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STOPING METHODS AND COSTS

BY

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CONTENTS

ILLUSTRATIONS

STOPING METHODS AND COSTS [1]

By Chas. F. Jackson [2] and E. D. Gardner [3]

INTRODUCTION

This bulletin is one of a series of Bureau of Mines reports dealing with mining methods, practices, and costs. A study of this subject was undertaken by the Bureau in cooperation with a large number of mining companies in 1928. This study continued for 5 years, during which engineers of the Mining Division of the Bureau visited most of the larger mines and many smaller ones in the United States, as well as several mines in Canada and Mexico.

As a result of this study, 75 reports dealing with underground mining methods and costs at different mines were published as information circulars. Circulars also were issued on 6 open-pit mines and 2 gold-dredging operations. These circulars were written by the managers, superintendents, and engineers in charge of the operations, in some instances in collaboration with Bureau engineers. They were prepared according to a standard outline drawn up by the Bureau to obtain uniformity in textual treatment and in the arrangement and basis of setting up cost and operating data.

In addition to the circulars on individual mines, summary circulars were prepared by Bureau engineers dealing with each of the principal underground mining methods.[4] These were based upon the circulars describing individual mines, upon extensive field studies by the authors, and upon data obtained from articles in the technical press and in the transactions of technical societies.

Other circulars and bulletins dealt with special mining problems and different phases of underground practice.[5]

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[1] Work on manuscript completed April 1934.

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[4] Jackson, Chas. F., Mining Ore in Open Stopes, Central and Eastern United States: Inf. Circ. 6183, Bureau of Mines, 1929 (revised April 1931), 30 pp.
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Gardner, E. D., Undercut Block-caving Method of Mining in Western Copper Mines: Inf. Circ. 6360, Bureau of Mines, 1930, 44 pp.
Jackson, Chas. F., Mining by the Top-slicing Method, with Some Notes on Sublevel Caving: Inf. Circ. 6410, 1931, 51 pp.
Johnson, C. H., and Gardner, E. D., Cut-and-fill Stoping: Inf. Circ. 6688, Bureau of Mines, 1933, 58 pp.
Gardner, E. D., and Vanderburg, Wm. O., Square-set System of Mining: Inf. Circ. 6691, 1933, 74 pp.

[5] Jackson, Chas. F., Some Notes on Underground Transportation: Inf. Circ. 6326, Bureau of Mines, 1930 (revised June 1931), 44 pp.
Jackson, Chas. F., and Knaebel, John B., Underground Chute Gates in Metal Mines: Inf. Circ. 6495, Bureau of Mines, 1931, 22 pp.
Gardner, E. D., and Johnson, J. F., Shaft-sinking Practices and Costs: Bull. 357, Bureau of Mines, 1932, 110 pp.
McElroy, G. E., Ventilation of the Large Copper Mines of Arizona: Bull. 330, Bureau of Mines, 1930, 145 pp.
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Crane, W. R., Essential Factors Influencing Subsidence and Ground Movement: Inf. Circ. 6501, 1931, 14 pp.
Knaebel, John B., Sampling and Exploration by Means of Hammer Drills: Inf. Circ. 6594, Bureau of Mines, 1932, 30 pp.

OBJECT AND SCOPE OF BULLETIN

This bulletin deals with stoping methods and costs and summarizes the data in earlier publications and those obtained during investigations in the field that apply particularly to stoping. The object is to discuss and analyze the several methods of stoping, with the principal variations of each, the conditions under which each is applicable, the factors that should be considered in the selection of a stoping method under different sets of conditions, the inherent advantages and disadvantages of each method, and stoping costs in dollars and in units of labor, materials, and mechanical power.

It is realized that stoping operations are affected by and in turn affect related underground operations, such as exploration, development, tramming, and drainage, and even milling operations, and that these various operations must be coordinated to obtain maximum efficiency in the mining of ore. It is, therefore, impossible to discuss stoping comprehensively without considering related operations. However, stoping will be divorced from other phases of mining, insofar as it is possible to do so, in order to compare the different stoping methods more clearly as to procedure and details of practice, applicability to different conditions, inherent advantages and disadvantages, and relative efficiency.

In studying different stoping methods and their variations it should be remembered that physical conditions and characteristics of an ore deposit which cannot be altered by the operator determine the stoping methods to be employed and that in comparing methods at different mines these characteristics also must be compared to reach correct conclusions. The discussions in this paper will be presented with the idea of linking the operating conditions imposed by nature to the practices employed to cope with these conditions.

DEFINITIONS AND CLASSIFICATION OF STOPING METHODS

DEFINITIONS

A stope is defined as follows:[6]

1. An excavation from which the ore has been extracted, either above or below a level, in a series of steps. * * * Usually applied to highly inclined

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Wright, Charles Will, Management of Labor in Successful Metal-mine Operations: Inf. Circ. 6050, Bureau of Mines, 1932, 35 pp.

Vanderburg, William O., Factors Governing the Selection of the Proper Level Interval in Underground Mines: Inf. Circ. 6618, Bureau of Mines, 1932, 17 pp.

Jackson, Chas. F., and Knaebel, John B., Sampling and Estimation of Ore Deposits: Bull. 356, Bureau of Mines, 1932, 155 pp.

Jackson, Chas. F., Some Notes on Methods and Costs of Equipping and Developing Prospects: Inf. Circ. 6693, Bureau of Mines, 1933, 24 pp.

Belgart, John R., The Cost of Developing to the Operating Stage and Equipping a Small or Medium-sized Mine in the Tri-State Lead and Zinc District: Inf. Circ. 6594, Bureau of Mines, 1932, 18 pp.

Keast, A. J., and Jackson, Chas. F., Method and Cost of Exploring, Equipping for Development, and Developing the Central Patricia Group of Claims, Northern Ontario: Inf. Circ. 6681, Bureau of Mines, 1933, 13 pp.

Emens, W. H., and Jackson, Chas. F., Methods and Costs of Developing and Equipping the Ashley Gold Mine, Matachewan Gold District, Ontario: Inf. Circ. 6707, Bureau of Mines, 1933, 27 pp.

Keller, Albert E., and Gillingham, E. C., Employee-timekeeping System and Mechanical Pay-roll Methods at Britannia Beach, British Columbia: Inf. Circ. 6622, Bureau of Mines, 1932, 6 pp.

Bishop, Fred L., and Keller, Albert E., Procedure of the Purchasing and Supply Departments of the Miami Copper Co., Miami, Ariz.: Inf. Circ. 6623, Bureau of Mines, 1932, 12 pp.

[6] Fay, Albert H., A Glossary of the Mining and Mineral Industry: Bull. 95, Bureau of Mines, 1920, p. 652.

or vertical veins. Frequently used incorrectly as a synonym of room, which is a wide working place in a flat mine.

2. To excavate ore in a vein by driving horizontally upon it a series of workings, one immediately over the other or vice versa. Each horizontal working is called a stope (probably a corruption of step), because when a number of them are in progress, each working face being a little in advance of the next above or below, the whole face under attack assumes the shape of a flight of stairs.

Stoping is, then, the act of excavating ore, either above or below a level, in a series of steps. In recent years the term has acquired a somewhat broader meaning, as it is employed to cover the removal of ore not only by a series of horizontal workings but by vertical or inclined workings as well.

In this paper the term "stoping" is employed in its broadest sense to mean the act of excavating ore by means of a series of horizontal, vertical, or inclined workings in veins or large, irregular bodies of ore, or by rooms in flat deposits. It covers the breaking and removal of the ore from underground openings, except those driven for exploration and development. The removal of ore from drifts, crosscuts, shafts, winzes, and raises, which are excavated to explore and develop an ore deposit, is incidental to the main purpose for which stopes are driven and is not a stoping operation in the sense here employed. Exploratory and development openings are driven to prepare a mine for extraction of the ore by stoping.

Ore deposits vary greatly as to their physical characteristics and the economic problems involved in their exploitation. Various methods of stoping have been devised for extracting the ore safely and economically from deposits of different types, and a nomenclature has been developed for use in referring to the different methods. Some of the terms adopted are descriptive and require no interpretation, while others, particularly those applied to variations of the principal methods, are not.

CLASSIFICATION

Numerous classifications of stoping methods have been proposed to clarify the nomenclature and to show the fundamental differences and relationships among the different methods. These have been made upon numerous bases and have been set up in tabular or chart form. Some have been simple, while others have included all variations of the principal methods. One of the latter type includes 175 different methods and, like other elaborate classifications, was devised principally for instruction in the classroom. The writers, in common with many others, prefer a simple classification of the principal, fundamentally different methods and consider the other methods merely variations of these.

In 1923 the mining methods committee of the American Institute of Mining and Metallurgical Engineers adopted a classification, after careful consideration and discussion of the subject.[7] This classification was based upon the direction of stoping and the method of support and included a number of the principal variations.

[7] Mining Methods Committee, American Institute of Mining and Metallurgical Engineers, Classification of Mining Methods: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, p. 10.

Other classifications have been based upon the method of support, direction of working, sequence of operations, or methods of handling the broken ore.

In 1928 the Bureau of Mines adopted a preliminary classification which was made a part of the outline for papers on mining methods and costs mentioned in the introduction. In the final analysis, the method of stoping that can be used for mining a given deposit depends upon those characteristics that determine the area or span of back and walls which will be self-supporting during the removal of the ore; the nature, size, and interval between supports required to maintain the back and walls of the excavations; and the requirements for permanently supporting the overlying and surrounding rocks and surface to prevent movement and subsidence. This is true regardless of whether the deposit is large or small, regular or irregular in outline, flat, inclined, vertical, low grade, or high grade. These factors are recognized as having an important bearing upon the details and variations of the principal method employed, but the fundamental factor and the one which must invariably be considered is that of support of the stopes. It therefore seems logical to base a classification of stoping methods upon the method of support of the back and walls of the stopes while the ore is being extracted.

The classification adopted by the Bureau in 1928 does not take into consideration the direction of stoping, such as "underhand", "overhand", "horizontal", or "rill" stoping; the sequence of working, as by "advancing" or "retreating"; or the methods of handling ore, as by "branch raises", etc. However, these considerations are recognized as important in distinguishing between variations of the principal methods. The Bureau classification, with minor changes, follows:

Classification of stoping methods

A. Stopes naturally supported.
  1. Open stoping.
    (a) Open stopes in small ore bodies.
    (b) Sublevel stoping.
  2. Open stopes with pillar supports.
    (a) Casual pillars.
    (b) Room (or stope) and pillar (regular arrangement).
B. Stopes artificially supported.
  3. Shrinkage stoping.
    (a) With pillars.
    (b) Without pillars.
    (c) With subsequent waste filling.
  4. Cut-and-fill stoping.
  5. Stulled stopes in narrow veins.
  6. Square-set stoping.
C. Caved stopes.
  7. Caving (ore broken by induced caving).
    (a) Block caving; including caving to main levels and caving to chutes or branched raises.
    (b) Sublevel caving.
  8. Top slicing (mining under a mat which, together with caved capping, follows the mining downward in successive stages).
D. Combinations of supported and caved stopes. (As shrinkage stoping with pillar caving, cut-and-fill stoping with top slicing of pillars, etc.)

A similar classification, also based on the method of support but arranged differently, was adopted by the Demographical Division of the Bureau in 1930 for use in obtaining accident statistics in metal mines. This classification is shown below.

Method of underground mining	Method of support	
	Temporary	Permanent
Open stope (including room-and-pillar and sublevel stoping.	Pillars.	Pillars.
Shrinkage.	Square-sets.	Filling.
Cut-and-fill.	Square-sets and filling.	No permanent support.
Square-set.	Stulls.	
Block caving.	Stulls and filling.	
Sublevel caving.	Filling only.	
Top slicing.		

In reporting accident statistics the mining companies were requested to check the mining methods employed, the method of temporary support, and the method of permanent support.

A. STOPES SUPPORTED NATURALLY; OPEN STOPES

Stopes naturally supported are those in which no regular artificial method of support is employed, although occasional props or cribs may be used to hold local patches of insecure ground. The walls and

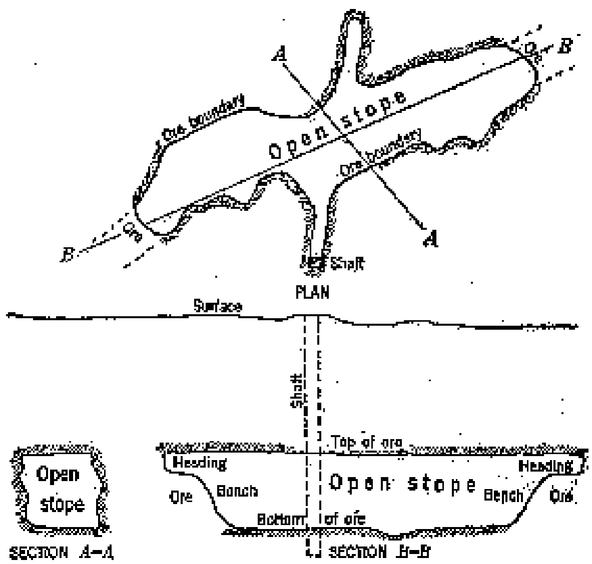


FIGURE 1.—Open stoping without pillars in small ore body.

roof are self-supporting, and open stopes can be used only where the ore and wall rocks are firm. The simplest open stopes are those in which the entire ore body is removed from wall to wall without leaving any pillars. (See fig. 1.) The stoping of ore in this manner is confined to relatively small ore bodies, since regardless of the firmness of the ground there is a limit to the length of unsupported span which will stand without breaking.

In sublevel stoping the ore is excavated in open stopes, retreating from one end of the stope toward the other. (See fig. 2.) The ore body is developed first by a series of sublevel drifts above the main haulage level. These sublevels usually are 20 to 25 feet apart vertically but in some instances are 40 or 50 feet apart. They are connected by a starting raise at one end of the stope and by a manway raise for entrance to the sublevels and stope face at the other end. Chute raises connect the haulage level to the lowest sublevel, at which the tops of the chute raises are belled out to form millholes. Beginning at the starting raise the ore is benched down from the sublevels; the broken ore falls into the millholes, whence it is drawn off through the chutes. The stope face is kept nearly vertical as it is benched backward toward the manway raise. In the commonest variation of the method the bench below each sublevel is farther back from the starting end of the stope than the bench above, so that the stope face has the appearance of a steep inverted stairway. In other variations the stope face may overhang at a flatter angle, may be carried vertically, or (in softer ground) may slope slightly outward toward the bottom. In any event the miners always are working under solid, undisturbed ground while breaking the ore out into the open stope. Within a given stope pillars are left only if ground is encountered which is too low in grade to be mined profitably. In wide ore bodies transverse sublevel stopes may be mined between regular pillars.

In open stopes with pillar supports the length of unsupported span is reduced by leaving pillars of ore in place. Pillars may be of the casual type, their position and size being determined by ground conditions and their arrangement not conforming with any predetermined, regular geometrical pattern (fig. 3); or they may be of a standard size and arrangement, conforming to a regular room-and-pillar or checkerboard pattern (fig. 4).

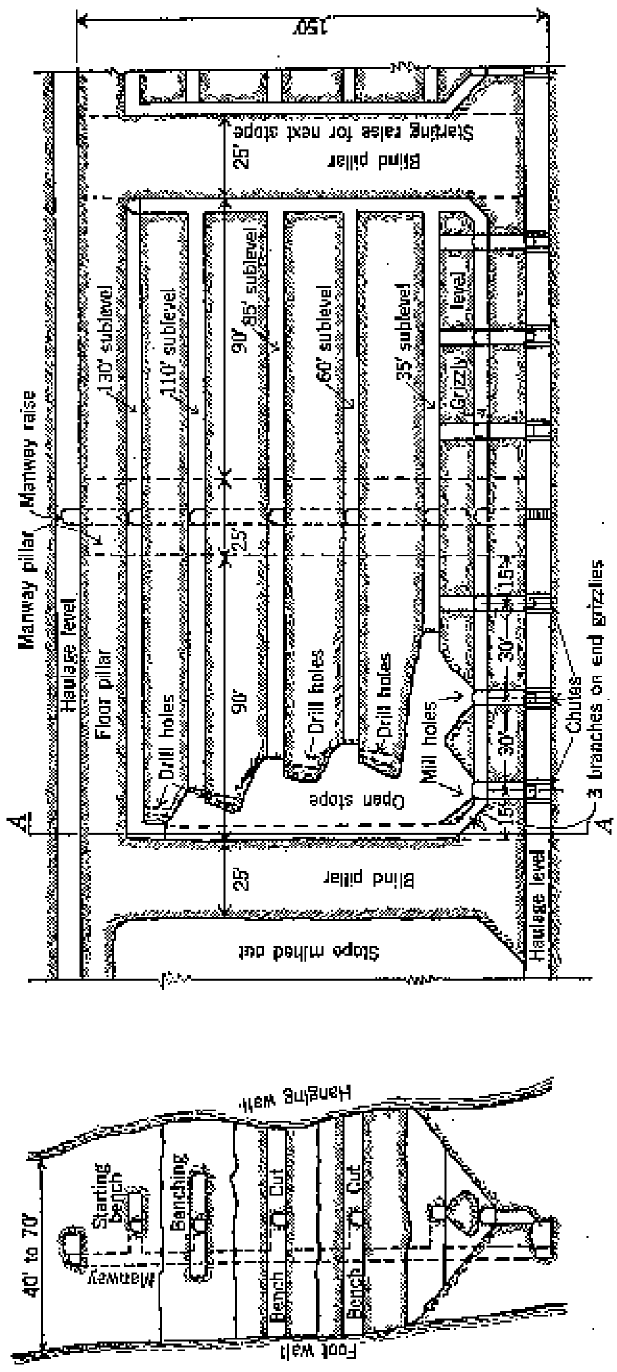
In mines where open stoping is employed the stopes may be filled with waste after they have been worked out, but since this filling is not introduced during excavation of the stopes it does not affect the stoping method.

B. STOPES SUPPORTED ARTIFICIALLY

SHRINKAGE STOPING

Shrinkage stoping is placed first under stopes supported artificially since it is more like open stoping than the other methods so classified. Indeed, there is some question whether shrinkage stoping should not be placed under open stopes. In shrinkage stoping broken ore is left in the stopes primarily to form a floor for the miners to stand upon while they are breaking down the back. On the other hand, this broken ore undoubtedly often provides a certain measure of temporary support to the walls of the stopes, therefore shrinkage stoping has been placed more or less arbitrarily under stopes supported artificially.

In this method of stoping the ore is mined out in successive flat or inclined slices, working upward from the level. After each slice is blasted down enough broken ore is drawn off from below to provide a working space between the top of the pile of broken ore



VERTICAL LONGITUDINAL SECTION

VERTICAL PROJECTION A–A

FIGURE 2.—Sublevel stoping.

and the back of the stope. (See fig. 5.) Usually about 40 percent of the broken ore will have been drawn off when the stope has been mined to the top.

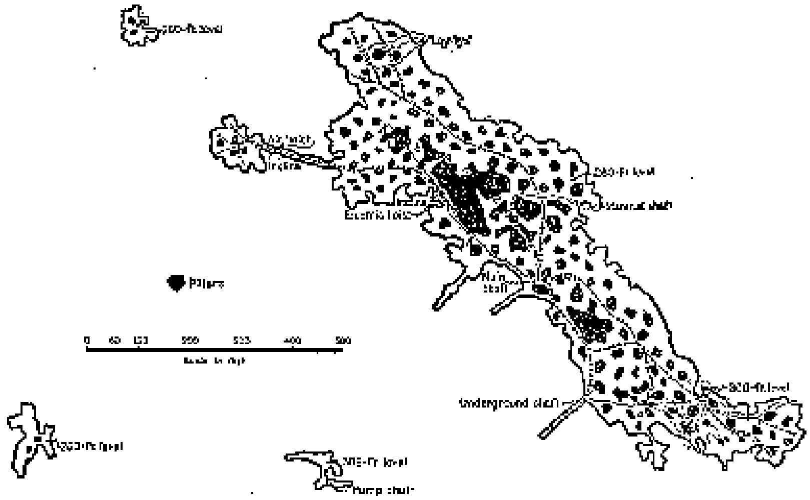


Figure 3.—Open stoping with casual pillars.

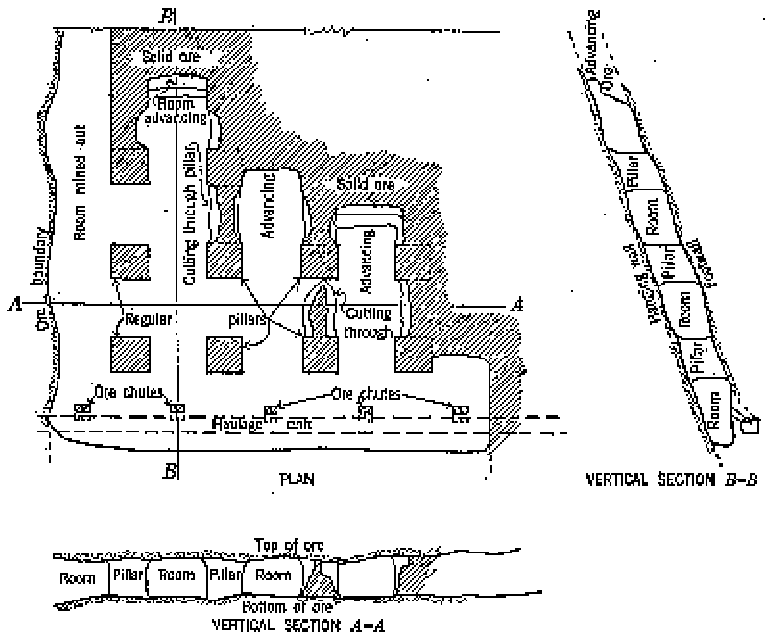


Figure 4.—Example of open stoping with regular pillars.

Shrinkage stopes often are excavated by taking slices along the vein (especially in narrow veins) from one end of an ore shoot to the other, without leaving any pillars for supporting the walls. Sometimes (especially in wide veins) the ore is mined in a series

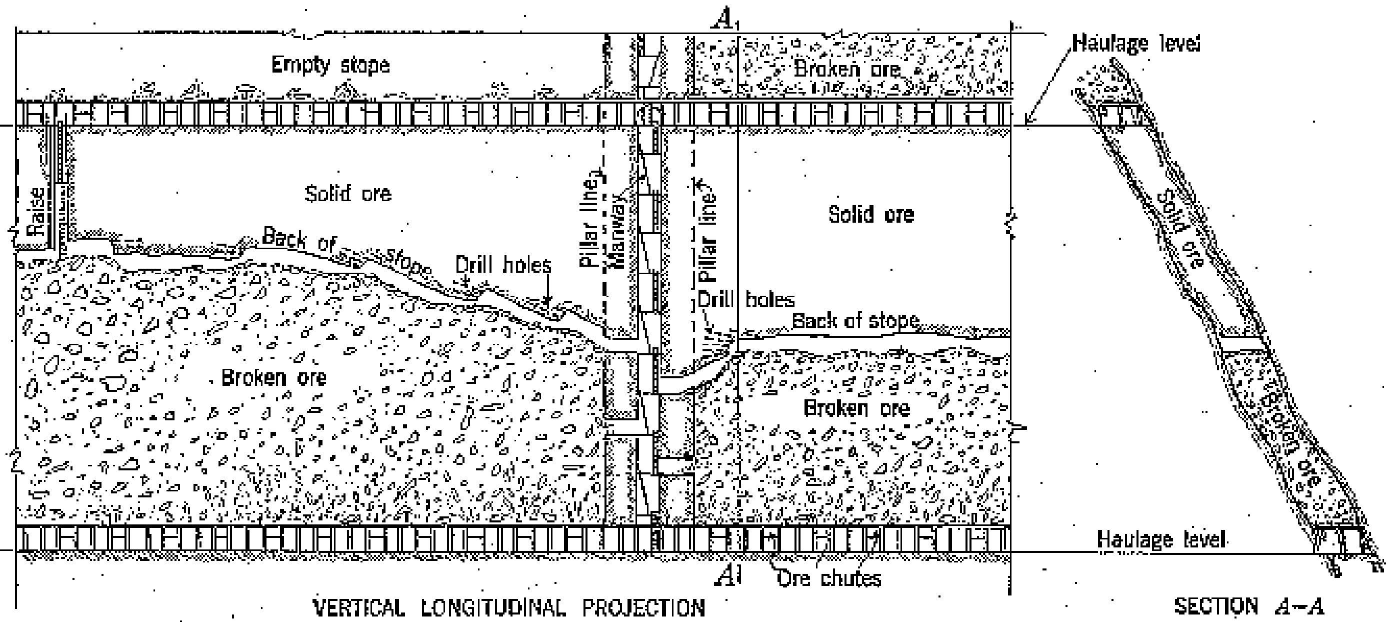


FIGURE 5.—Example of shrinkage stoping; stoping on drift sets.

of transverse stopes of limited size, each stope being separated from the next by a pillar of solid ore to reduce the length of the unsupported span. In some instances casual pillars may be left to support local areas where the walls are weak, and in others pillars of lean ore or waste within the ore body are left. The latter are left primarily because it does not pay to mine them, but at the same time they reduce the length of the unsupported span and assist in supporting the walls and back.

After a shrinkage stope has been mined out to the top the broken ore is drawn off, leaving the stope empty. The stope may be filled with waste later, either because the empty stope affords a handy place to dispose of waste rock from development, or because it is an essential part of the stoping method. Thus it may be necessary to fill two empty stopes to extract the ore from an intervening pillar with safety.

Stulls and props often are employed to support insecure patches of ground temporarily in shrinkage stopes.

CUT-AND-FILL STOPING

In cut-and-fill stoping the ore is excavated by successive flat or inclined slices, working upward from the level, as in shrinkage stoping. However, after each slice is blasted down all broken ore is removed, and the stope is filled with waste up to within a few feet of the back before the next slice is taken out, just enough room being left between the top of the waste pile and the back of the stope to provide working space. (See fig. 6.) The term "cut-and-fill stoping" implies a definite and characteristic sequence of operations: (1) Breaking a slice of ore from the back, (2) removing the broken ore, (3) introducing filling—then (1) breaking again, and so on. The filling is introduced primarily to support the walls of the stope and may consist of waste sorted from the ore in the stope, waste rock from development work, rock from waste stopes excavated to provide filling material, rock from surface glory holes, or sand and gravel or mill tailings. Stulls, props, or cribs often are employed to support local patches of insecure ground.

STULLED STOPES IN NARROW VEINS

The walls of narrow veins frequently are supported by stull timbers placed between the foot and hanging walls, which constitute the only artificial support provided during the excavation of the stopes. Stulls may be placed at irregular intervals to support local patches of insecure ground, in which case the stopes are virtually open stopes. Sometimes the stulls are placed at regular intervals both along the stope and vertically, in which case stull stoping should be considered a distinctive method. (See fig. 7.) Filling may be introduced where necessary to provide better and more lasting support for heavy ground. If close filling is required and the regular cycle of operations characteristic of cut-and-fill stoping is followed the method is classified as cut-and-fill stoping.

SQUARE-SET STOPING

The term "square-set stoping" is applied to that method of mining in which the walls and back of the excavation are supported by regular framed timbers forming a skeleton enclosing a series of con-

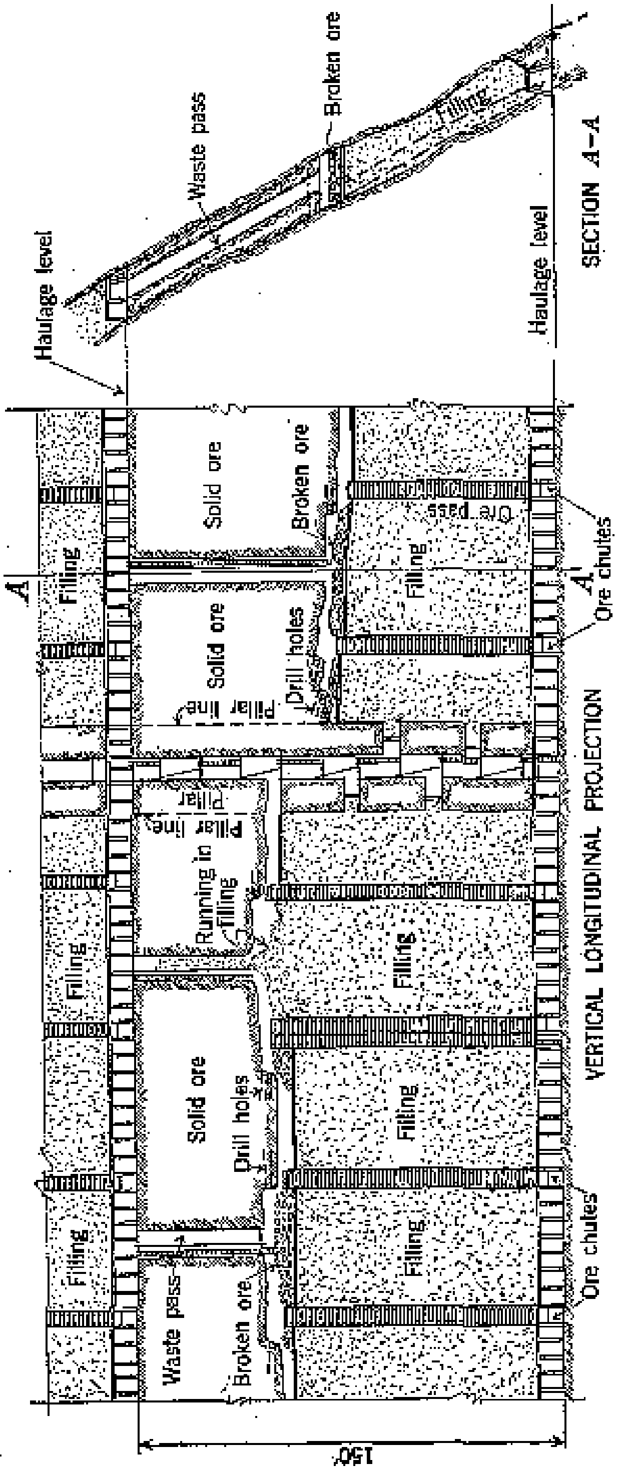


FIGURE 6.—Example of horizontal cut-and-fill stoping.

nected, hollow, rectangular prisms in the space formerly occupied by the excavated ore and providing continuous lines of support in three directions at right angles to each other. (See fig. 8, *A*.) The ore is excavated in small, rectangular blocks just large enough to provide room for standing a set of timber. The essential timbers comprising a standard square-set are respectively termed "posts", "caps", and "girts" (or "ties"). The posts are the upright members, and the caps and girts are the horizontal members. The ends of the members are framed to give each a bearing against the other two at the corners of the sets where they join together. The stopes usually are mined out in floors or horizontal panels, and the sets of each successive floor are framed into the sets of the preceding floor. (See fig. 8, *B*.) Sometimes, however, the sets are mined out in

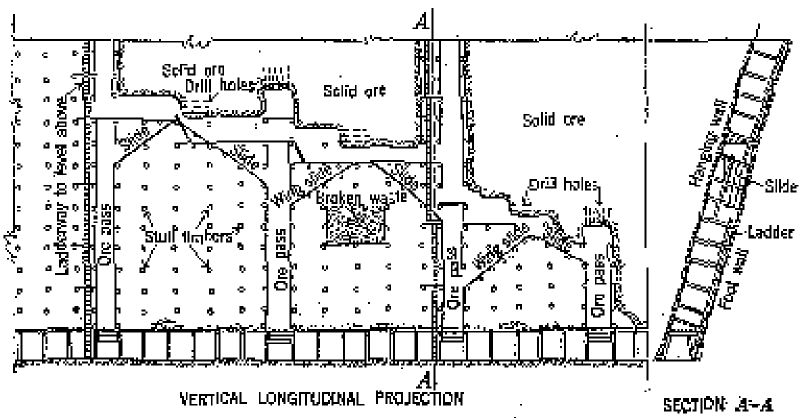


FIGURE 7.—Example of stull stoping in narrow vein.

a series of vertical or inclined panels. Square-set stoping usually is accompanied by filling of the stoped ground, and often in heavy ground the sets are filled with waste shortly after they are put in, leaving only a small volume of unfilled stope at any one time.

C. CAVED STOPES

There are two distinct types of caved stopes. In the first, the ore is broken by caving induced by undercutting a block of ore. In the second, the ore itself is removed by excavating a series of horizontal or inclined slices, while the overlying capping is allowed to cave and fill the space occupied previously by the ore. The first type comprises the caving methods of mining, while the second comprises the top-slicing method.

BLOCK CAVING

In block caving (fig. 9) a thick block of ore is partly cut off from surrounding blocks by a series of drifts, one above the other, or by boundary shrinkage stopes; it is then undercut by removing a slice of ore or a series of slices separated by small pillars underneath the block. The isolated, unsupported block of ore breaks and caves under its own weight. The broken ore is drawn off from below, and as the caved mass moves downward, due to continued drawing of broken ore from below, it is broken further by pressure and attrition. The overlying capping caves and follows the broken ore downward. In the earliest applications of the caving

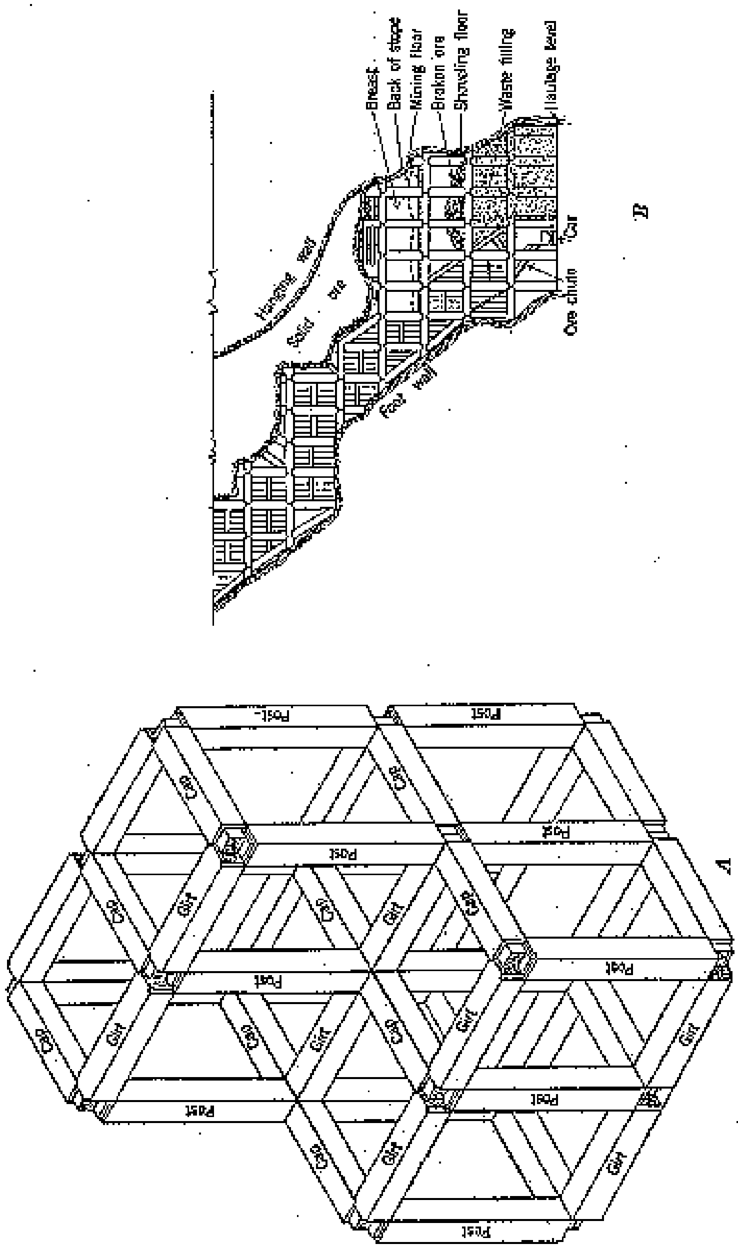


FIGURE 8.—Example of square-set stoping: A, Square-set timbering; B, vertical transverse section through typical square-set stope.

method the block was undercut on or immediately above the haulage level, and the caved ore was shoveled into cars in drifts driven under the cave or spiled through it. This system entailed the driving and

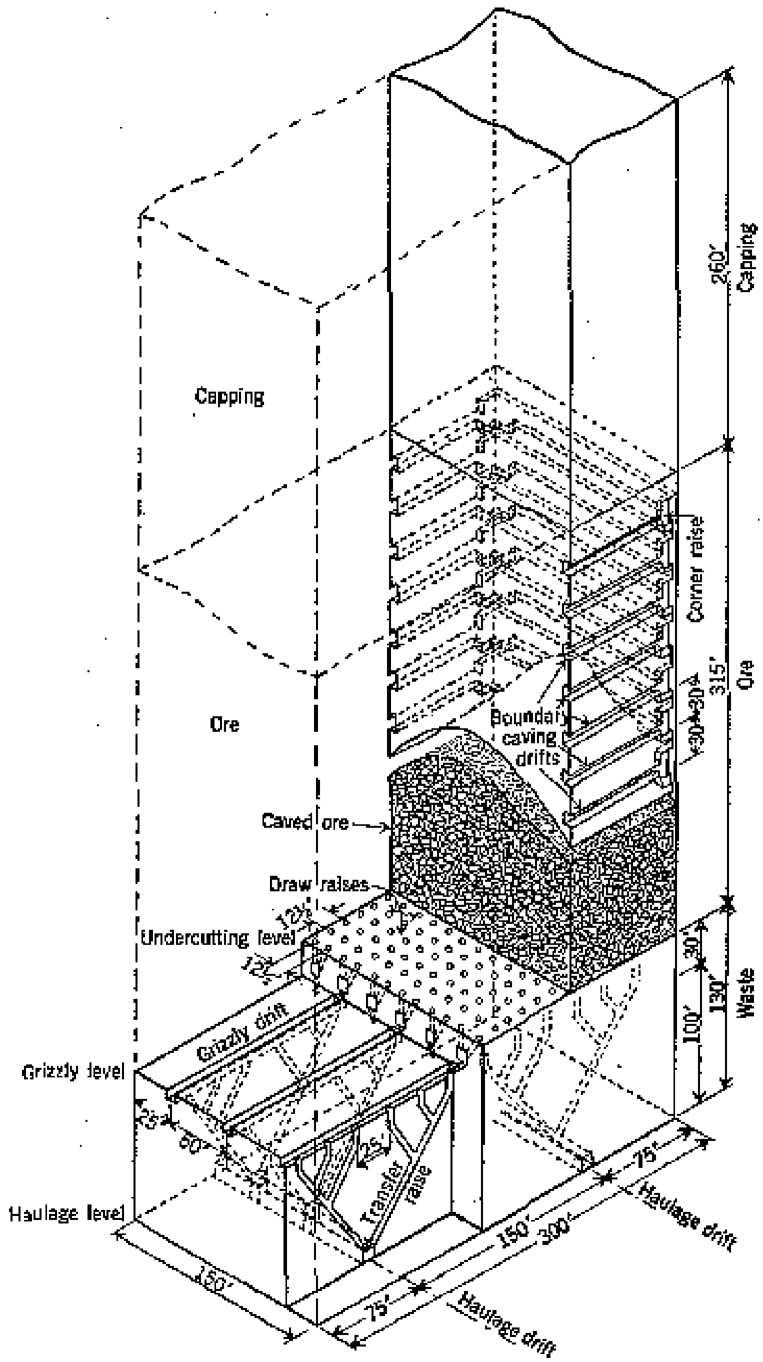


FIGURE 9.—Example of block caving (after F. W. Maclennan).

maintenance of a large number of drifts to recover the ore and has been superseded by caving to chutes or branched raises. At present the block of ore is usually undercut some distance above the haulage level, so that by driving a number of inclined branched raises a large undercut area can be tapped at closely spaced points from relatively few main-level haulage drifts, which may be placed well below the influence of the pressure induced by the caving block. (See fig. 9.) With this system hand shoveling is virtually eliminated.

SUBLEVEL CAVING

In sublevel caving, relatively thin blocks of ore are caused to cave by successively undermining small panels. The ore deposit is developed by a series of sublevels spaced at vertical intervals of 18 to 25 or 30 feet and occasionally more. Figure 10, *A*, *B*, and *C*, shows complete development of a block. This is not all done before stoping operations are begun, but usually only one or two sublevels are developed at a time, beginning at the top of the ore body. The sublevels are developed by connecting the raises with a longitudinal subdrift from which timbered slice drifts are driven right and left opposite the raises to the ore boundaries or to the limits of the block. Usually alternate drifts are driven first, and caving back from them is begun and continued while the intermediate slices are being driven. The caving is begun at the ends of the slices by blasting out cuts, as shown in figure 10, *D*, and retreating in the same manner toward the raises. The broken and caved ore formerly was shoveled into cars and trammed to the raises, but in recent years it is dragged to the raises by power scrapers. Successively lower sublevels are developed and caved back until the entire block has been mined. This method is intermediate between block caving and top slicing, since part of the ore is mined as in top slicing and part is caved.

TOP SLICING

The term "top slicing" is applied to the method of mining in which the ore is extracted by excavating a series of horizontal (sometimes inclined) timbered slices alongside each other, beginning at the top of the ore body and working progressively downward; the slices are caved by blasting out the timbers, bringing the capping or overburden down upon the bottom of the slices which have been previously covered with a floor or mat of timber to separate the caved material from the solid ore beneath. (See fig. 11.) Succeedingly lower slices are mined in a similar manner up to the overlying mat or gob, which consists of an accumulation of broken timbers and lagging from the upper slices and of caved capping. As the slices are mined out and caved this mat follows the mining downward, filling the space occupied previously by the ore.

D. COMBINATIONS OF SUPPORTED AND CAVED STOPES

The question has been raised frequently as to whether combinations of supported and caved stope methods should be classed as distinctive methods. The writers believe that certain such combinations should be. Thus, where a stoping system is laid out in advance of operations in which a series of stopes separated by regular pillars are to be mined by a supported stoping method, such as shrinkage, and the pillars are to be mined by caving, a distinctive stoping method seems to exist. Another example is the method in which block caving is preceded by excavating boundary shrinkage stopes around the block to be caved or the block is undercut by a series of shrinkage stopes. However, where less

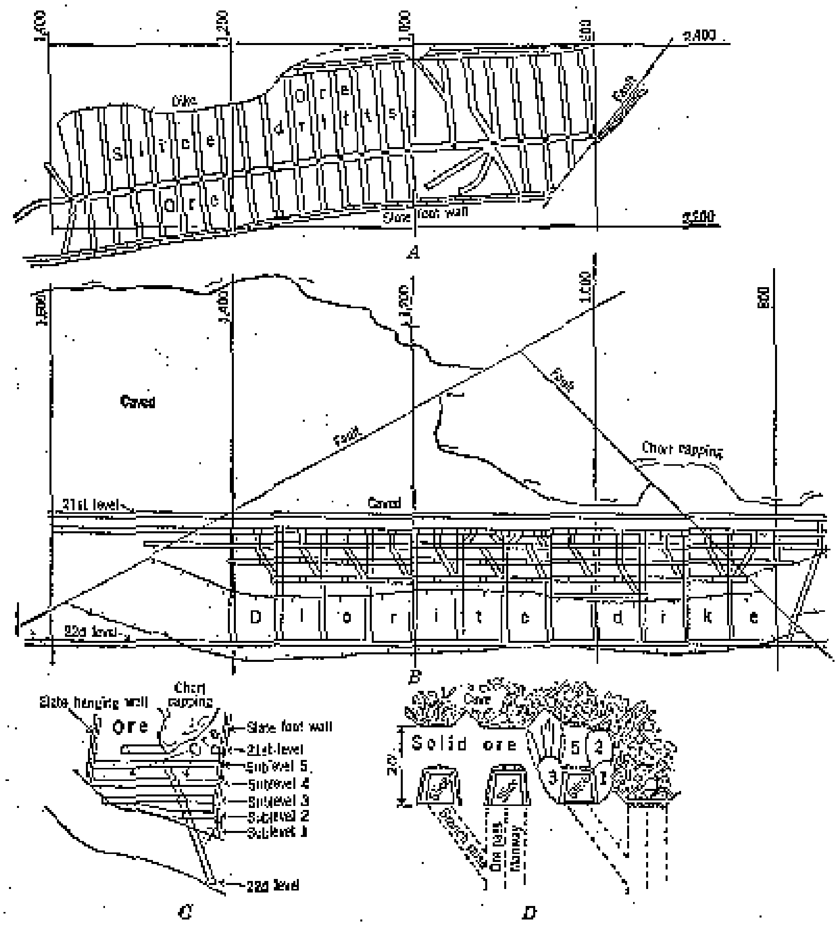
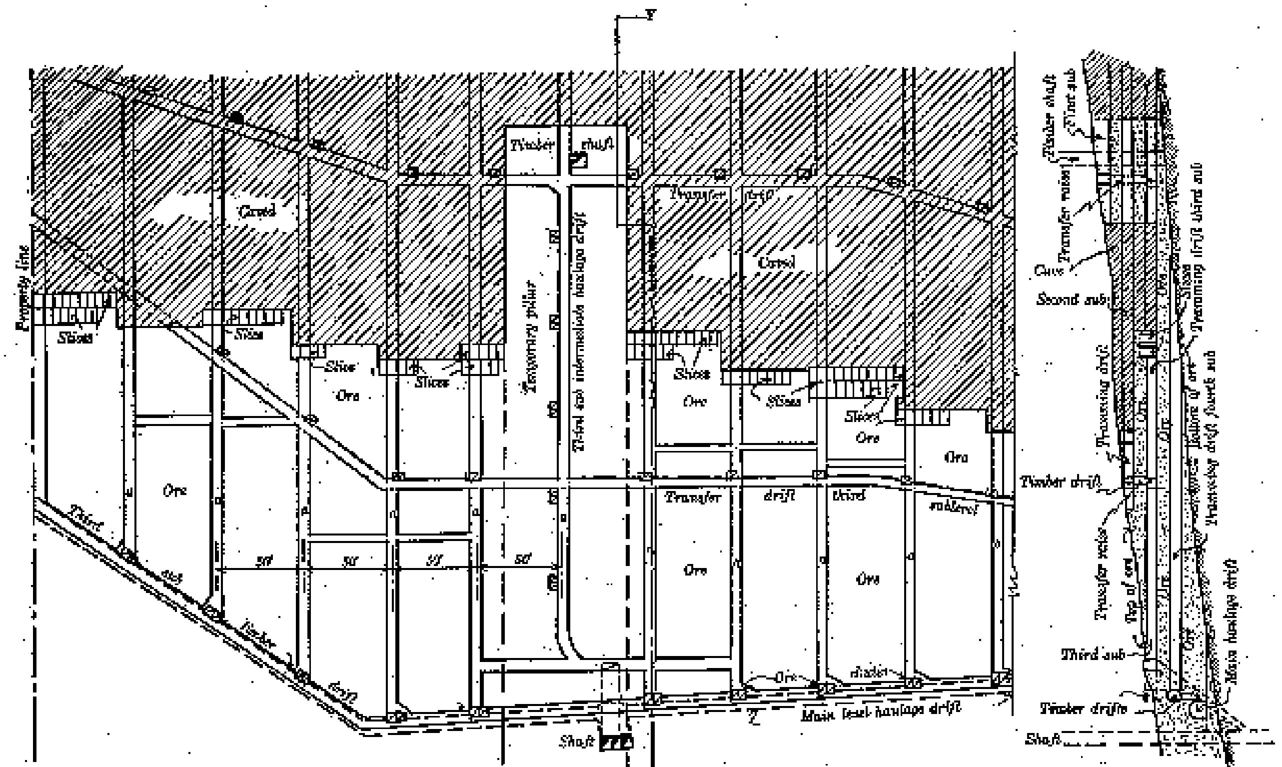


FIGURE 10.—Example of sublevel caving: A, Plan of sublevel; B, vertical longitudinal section; C, vertical cross section; D, cross section, showing details of caving back, cuts blasted out in order shown.

than 2 percent of the ore in a block is drawn from shrinkage stopes (now the practice is most mines employing block caving) it would be illogical to consider it a combination method. A third example is a system in which the pillars between mined-out cut-and-fill stopes are mined by top-slicing or square-setting. On the other hand, where a number of different methods are used independently for mining different ore bodies or different parts of the same ore body in a mine because of differences in the physical character of the ore and wall rocks, it would not be considered a combination method of stoping.



NOTE: A indicates tramming drift

A. PLAN OF PART OF THIRD SUBLEVEL

B. CROSS-SECTION ON X-Y

FIGURE 11.—Example of parallel top slicing, scraping to cars; flat-dipping ore body, two to four sublevels thick.

CLASSIFICATION SUMMARY

The simple classification adopted by the Bureau of Mines embraces only the principal methods of stoping and is based upon support or lack of support for the stopes and the method of support employed. The terms applied to the methods are in each instance those with which mining men are generally familiar; variations of the principal methods, to which different names have been applied locally, are not included.

From the foregoing brief discussion of the classified methods it is evident that there is not always a sharp distinction between the different classes of stopes. This is well-illustrated by shrinkage stoping, which may be considered either as open stoping or as stoping with artificial support, depending upon whether in any given instance the broken ore simply provides a footing for the men to work on or whether it provides essential support to the stope walls. Moreover, a stulled stope may be considered an open stope if the stull timbering is irregular and is used only to support occasional weak sections in the stope, or as an artificially supported stope where the stulls are installed regularly and systematically throughout the stope area for support of the walls. Some stulled stopes even might be considered caved stopes where stulls are used along a retreating longwall face for temporary support and the hanging wall is permitted to break the timbers and come down behind the working face. During the past 10 years the authors have participated in, listened to, and read numerous discussions on the classification of stoping methods but have yet to see a classification to which all authorities would agree or one in which the classification of certain of the principal methods is not open to some question. That adopted by the Bureau seems to be the least open to argument of any presented. Finally, the principal value of a classification is to clarify the basic principles of the different methods and their distinguishing features, and it is believed that of the Bureau of Mines does this.

PRODUCTION BY PRINCIPAL STOPING METHODS

The tonnage of ore mined by each stoping method cannot be considered a measure of its importance, since each is important to those types of ore bodies and mining conditions under which it is applied. A table of ore tonnages mined by the various methods is of interest, however, as it indicates the importance of the different types of ore bodies to which each method is applicable; when considered in conjunction with the cost data and data on units of labor and supplies given later the table supplies a rough measure of the labor, timber, explosives, and power costs for the stoping of ore.

Table 1 shows the production of the principal metallic ores mined in the United States in 1930, by each of the principal stoping methods. The year 1930 has been selected rather than a later period, since 1930 was the last year during which practically all of the producing companies were operating. During 1931, 1932, and 1933 most of the smaller high-cost producers and many larger producers were closed or their output was greatly curtailed. Figures for 1930 therefore give a truer picture of production as regards stoping methods than those for later years.

The figures in table 1 are admittedly incomplete. They have been summarized from reports of the mining companies, which are confidential as far as individual mines are concerned. However, production figures from all of the larger companies and several hundred smaller ones are included, and it is believed that the table covers at least 90 to 95 percent of the tonnage produced by underground methods in the United States during 1930. Where more than one stoping method was employed the percentage of ore mined by each method usually was given, so that it was possible to allot the tonnages to the proper stoping method. However, these tonnages include ore removed in development work, and there are no available data showing the percentage of development ore. The output of mines reporting development production only has been omitted.

TABLE 1.—*Tonnages*[1] *of metallic ores mined in the United States, by principal stoping methods, 1930*

	Gold and silver ores	Copper ores	Lead, zinc, and lead-silver ores	Iron ores [2]	Other ores [3]	Total
Open stoping (including small ore bodies and large bodies with casual pillars and with regular pillars)	377,200	3,774,500	14,661,000	9,789,000	393,000	28,994,700
Sublevel stoping	70,800	684,500		2,397,000		3,052,300
Total open stoping	448,000	4,459,000	14,661,000	12,186,000	393,000	32,052,000
Shrinkage stoping	[4] 6,433,000	1,665,000	1,169,000	1,615,500	77,500	[4] 9,847,000
Cut-and-fill stoping	45,000	1,409,000	1,077,000	105,000	10,000	2,739,000
Stulled stopes		873,000				873,000
Square-set stoping	872,000	3,149,000	1,900,000		81,000	6,002,000
Block caving		15,209,500	31,000			15,240,500
Caving pillars	408,500			108,000		516,500
Sublevel caving				6,918,000		6,918,000
Top slicing		85,000	464,500	16,790,500		17,340,000
Total	[4] 7,206,500	26,639,500	19,395,500	37,712,000	560,500	91,620,000

[1] Short tons.
[2] Original figures converted to short tons for totaling with other tonnages.
[3] Manganese, bauxite, cinnabar, pyrites.
[4] Includes Alaska-Juneau, 3,924,600 tons.

APPLICABILITY OF DIFFERENT STOPING METHODS AS DETERMINED BY PHYSICAL FACTORS AFFECTING STOPE SUPPORT

The statement already has been made that the applicability of the different stoping methods depends fundamentally upon the degree to which the ore and wall rocks will stand unsupported and how the methods meet the requirements for support of the back and walls of the stopes.

A certain stoping method may be highly desirable as to simplicity, low cost, and high production rate, but its use may be impossible due to the limitations set by the physical characteristics of the ore deposit as they affect support.

A number of other factors that govern the selection of a stoping method will be discussed later. These factors often are of the utmost importance to successful mine operation, but they are determinative only within limits governed by the physical characteristics of the ore and wall rocks and the support of the stopes. Usually they determine the variations of the principal methods and details of practice rather than the methods themselves. If the stopes cannot be kept open while the ore is being extracted, it is obvious that a method otherwise desirable cannot be employed.

Before beginning a broader and more involved discussion of the factors governing the selection of a stoping method it therefore seems logical to discuss briefly the applicability of different stoping methods as determined by physical factors affecting stope support.

SPECIFIC FACTORS

STRENGTH OF ORE AND WALL ROCKS

Strength of ore and wall rocks is the first important characteristic of the ore deposit that affects the stoping method, as it determines the size of excavation, the length of time it will remain open, and the character of support required.

The strength of the ground as it is considered here depends upon the inherent strength of the rock itself, the fractures and planes of weakness and their arrangement and spacing, major structural features such as faults or bedding planes, and the element of time. A small block of rock of uniform texture and structure which is not cut by fracture or joint planes may be much stronger than a larger mass lacking uniformity and cut by such planes of weakness. The arrangement and interval between planes of weakness also affect the strength of the ground. Thus a series of joints or bedding planes, all running in the same direction—that is, parallel to each other—may make the rock weak in its resistance to stresses in one direction while it is strong at right angles thereto. If the rock is cut by numerous planes of weakness at various angles to each other the ground may be structurally weak, although the rock itself may be inherently strong. Near fault planes the ground may be very weak, whereas at some distance from the faults it may be strong.

Frequently, fresh rock is strong, but after it has been exposed to the air for some time it may slough or swell and become heavy. Sometimes the stresses imposed on a strong rock arch may weaken it gradually until it will no longer stand. Thus it appears that the time element must be considered in relation to strength of the ground in mining operations.

SIZE OF DEPOSIT

A small ore body with firm walls often can be stoped out from wall to wall without any support to the stope other than that afforded by the walls themselves. With larger ore bodies in the same kind of ground it may be necessary to provide added support in the form of pillars, filling, or timber because of the greater area of wall or back exposed by the excavation of the ore. In other words, there is a limit to the unsupported span or arch that will stand by itself, even in very strong ground, therefore the size of the ore body helps to determine the stoping methods applicable.

SHAPE AND DIP OF DEPOSIT

An ore body may be large in total volume but may be narrow in one dimension and wide in two dimensions (tabular) or narrow in two dimensions (pipe deposit). In a tabular deposit dipping at a high angle the unsupported horizontal span is small, and it may be possible to stope the ore from wall to wall without introducing supports if the walls are moderately firm. In stoping a similar deposit dipping at a low angle the area of back or horizontal span may be so great that it is necessary to support the hanging wall with pillars or by other means.

In stoping, a deposit which is regular in shape may require less support than one which is very irregular, other conditions being the same. Thus, in mining an irregular deposit the walls of the stopes may be very uneven, with overhanging slabs or projecting columns of wall rock extending into the stopes. Such irregularities may require support, whereas if the walls were of the same inherent strength but smooth and regular they might stand without added support. This factor is of particular importance in very deep workings; in the deep or "rock-burst" zone it has been found that corners and projections are affected most by rock bursts.

It therefore appears that the shape and dip of the deposit may have an important influence upon the method of support in stoping, hence upon the stoping method.

DEPTH BELOW SURFACE AND CHARACTER OF OVERBURDEN

It happens frequently that as the depth of mining operations increases the mining method must be changed, due to requirements for increased support. Near the surface the rocks are not under as great stress as at depths where the superincumbent load of the overlying rock places a strain on the walls of the excavations.

The character of the overburden and the condition of the ground above the workings have an important bearing on rock stresses at depth. The weight of a thick mass of unconsolidated overburden is sustained by the underlying rock, and if the latter is honeycombed with mine excavations the weight is transmitted to the pillars, walls, or other supports, either directly or by the thrust of the rock arches. Similarly, if a pressure block is formed over the workings by huge masses of rock becoming severed from the surrounding formations, great pressures may be transmitted to the supports in the mine. Pressure blocks of tremendous size may be present as a result of natural phenomena, such as faults, or as a result of mining.

DIRECTION OF PRESSURE

The direction of pressure has an important bearing upon the method of support in stoping. Thus, in a steeply dipping tabular deposit, filling generally will furnish adequate support against side pressure from the walls, whereas in a wide deposit, where the greatest pressure acts vertically downward on the backs of the stopes, filling cannot be relied upon to prevent movement. In the latter, filling cannot be introduced tightly enough against the back to support permanently any great weight that may come upon it. Furthermore, filling usually shrinks in volume after it is placed, thus leaving open spaces and unsupported areas of roof. Therefore, a cut-and-fill method can be employed where the pressure is from the sides and the back is strong and self-supporting, whereas if the pressure is from above, square-setting supplemented by filling, or some other method of holding the back, may be necessary.

REQUIREMENTS FOR PERMANENT SUPPORT

The requirements for permanent support of the stopes may determine the method of stoping. Thus, if it is necessary to prevent caving and subsidence of the surface because of permanent surface improvements, or to prevent the water from a lake or stream from entering the mine, a caved-stope method could not safely be employed. Likewise, the position of shafts and other workings relative to the stoping area may be such that a caved-stope method, or one that would permit some ground movement, could not be employed.

APPLICABILITY OF PRINCIPAL METHODS

The applicability of the principal stoping methods from the standpoint of support is discussed briefly in the following section. Other factors governing the applicability of the different methods will be discussed later in this bulletin.

STOPES SUPPORTED NATURALLY; OPEN STOPING

Open stoping is applicable to deposits of strong, firm ore having strong, firm walls. In table 1, 32,052,000 short tons of ore are shown to have been mined by open stoping in 1930, out of a total of 91,520,000 short tons mined by all underground methods in the United States during that year. Of the metallic ores mined by underground methods, 36 percent, therefore, was recovered by open stoping. About 75 percent of the open-stope tonnage was produced in lead, zinc, copper, and iron mines of the Eastern and Central States, indicating the predominance of ore deposits workable by open stoping in these sections of the country. Much of this tonnage came from flat-lying beds, such as those of the Tri-State and Southeastern Missouri districts, the Birmingham (Ala.) district, and eastern Tennessee.

In the narrower deposits (30 to 50 feet, maximum width) the ore often can be stoped the full width of the deposit without leaving any pillars. In wider deposits it usually is necessary to leave pillars to reduce the length of unsupported span over the stopes and prevent failure of the back.

In general, casual pillars are used in the thicker, flat beds, whereas regular room-and-pillar mining is used more for mining the thinner beds. With casual pillars it frequently is possible to utilize the lower-grade areas within the ore bodies for pillars, at least in part, thus permitting a larger proportion of the better grade of ore to be extracted than if the pillars were in the good ore. The method of casual pillars is generally applicable to the firmest type of ground in ore deposits of variable thickness where the area of the pillars can be proportioned to suit the conditions of the back and the height of the ore. The method may be employed where some sloughing of the back and pillars may occur after the stopes are worked out and abandoned but will not result seriously. Usually a somewhat higher percentage of the total ore in the deposit is recovered by first mining than with the room-and-pillar system.

The room-and-pillar method is particularly adaptable to thin, flat-lying beds of ore where the roof must be supported definitely and permanently and where the back is such that unsupported spans nowhere can exceed a predetermined maximum length—in other words, where it is not safe to rely on personal judgment for determining the amount of support required at individual local points. An example of this is afforded by conditions at some of the iron mines in the Birmingham (Ala.) district where there is a thick, porous stratum saturated with water about 150 feet above the ore bed. Here it is necessary to make sure that there will be no caving or breaks in the roof which would let the water from this stratum into the mine. Under such conditions a room-and-pillar system in which the pillars are spaced regularly and are of a size capable of resisting the roof pressure and preventing settlement generally is safer than a casual-pillar system.

In the sublevel stoping system the stope faces retreat from the ends of the ore body or, in transverse stopes, from a wall. The miners always are under solid ground and have a safe avenue of retreat through the subdrifts to the manway raises. Slabbing of the roof and walls at the worked-out end of the stope does not endanger the workmen and, if the stope is mined back rapidly, usually will not dilute the ore with waste. This system is applicable to tabular ore deposits dipping 50° or more and to wide, thick ore bodies in which the ore will stand well without timber and which have firm walls. It is applicable in weaker ground than could be worked safely by other open-stoping methods, as the miners are working under solid ground cut only by narrow sublevel drifts where caves or falls are not likely to occur and they do not have to go under the back of the stope itself. Large open stopes are worked successfully by this method in ground which can be drilled with auger steel. In fact, the method is applicable where the ore is too weak to be worked by shrinkage stoping, which is classified as an artificially supported stope method. The walls should be firm, however.

STOPES SUPPORTED ARTIFICIALLY

SHRINKAGE STOPING

Shrinkage stoping is applicable to bodies of strong, firm ore enclosed between firm walls which will not slough to any great extent after standing for a considerable time. The method is applied most often to thin, tabular deposits dipping at angles greater than 50°, in which there are few horses of waste (although sometimes these are left as pillars) and which have fairly regular walls.

It is also applicable to large, thick ore bodies in which the entire deposit is to be mined from wall to wall, regardless of horses or bands of low-grade or waste rock within the ore body, which are removed with the ore. In this type of ore body modifications of the simple method employed for shrinking in narrow, tabular deposits usually are employed. Over one-third of the tonnage mined in the United States and Alaska by shrinkage stoping in 1930 came from two operations of this type. Shrinkage stoping also is employed in wide ore bodies by excavating transverse shrinkage stopes, separated by regular pillars, using a system virtually identical to that used in tabular ore bodies for individual stopes. The pillars are left between the stopes to reduce the unsupported span, which otherwise would be too wide to stand.

For shrinkage stoping the ore must be strong and firm, since the back under which the men work is unsupported except for occasional props under patches of insecure ground. Although the broken ore left in the stope affords support to the walls until the stope is worked out and is drawn empty, the walls may slough or even collapse during and after drawing unless the walls are strong and firm. Sloughing of the walls will dilute the broken ore with waste, even though the walls do not collapse. Therefore, the walls must be firm for shrinkage stoping to be applied satisfactorily. In general, for shrinkage stoping the ore itself must be firmer and stronger than is required for sublevel stoping, while the walls may be somewhat weaker. Shrinkage stoping is applicable to narrower ore bodies than is sublevel stoping, although this is due more to cost considerations than to questions of support.

CUT-AND-FILL STOPING

Cut-and-fill stoping finds its best application in the mining of firm ore enclosed between walls, one or both of which are weak. The deposits may be of the tabular type, dipping 50° or more, or they may be wide, thick ore bodies. Flatter deposits may be worked by providing special means for transferring the ore from the face to the level. In general, this method is applicable to more irregular ore bodies than is shrinkage stoping and to deposits in which shrinkage stoping could be employed except for the fact that the walls are too weak. In cut-and-fill stoping the fill supports directly only the walls of the stopes and does not support the back. Failure of the walls, however, may undermine and induce failure of the back. The ore itself, therefore, should be strong for cut-and-fill stoping. Other conditions are frequently as important as those that affect support in the selection of cut-and-fill in preference to shrinkage stoping, and these will be discussed later.

STULLED STOPES IN NARROW VEINS

There is some question as to whether stull stoping should be considered one of the principal stoping methods. Stull timbering often is used in open stoping and in other stoping methods to support local patches of bad ground. However, a regular arrangement of stull timbers often is employed to support the immediate hanging wall in stoping thin, tabular deposits, constituting a distinct method of support. It is especially applicable to thin, tabular deposits having an immediate hanging wall which is weak and will slab off for several feet back to a stable hanging wall unless supported in some manner. In other words, when the ore is blasted out, if a slab of hanging wall soon will come away, endangering the miners and diluting the broken ore, this system sometimes is more economical than it is to let the hanging wall cave and sort the ore and waste in the stope. Where the weak hanging wall extends far back from the ore stull timbering cannot be depended upon to hold it up, but where there is only a comparatively thin slab of weak hanging wall, above which the ground is firm, regular stull timbering may be very effective, acting to steady the wall rather than to hold any great weight.

SQUARE-SET STOPING

Square-set stoping is adapted to the mining of regular or irregular ore bodies, commonly on dips steeper than about 45°, where the ore and/or walls are too weak to stand even over short spans for more than a very brief time and where caving and subsidence of the overlying rock must be prevented. From the standpoint of support alone, both temporary and permanent, the method is applicable to conditions under which no other satisfactory method has as yet been devised. Where the overlying strata may be allowed to cave and where the loss of some ore or some dilution with waste is not a serious objection, caved stope methods are adaptable to as bad or worse ground as square-setting. Other aspects of the applicability of the method will be discussed later. Square-set stoping is not used extensively any more, except where the ground is so heavy that the stopes must be closely filled with waste to provide permanent support soon after the timbers are placed, and where only a small volume of stope can be left unfilled at one time. The method often is used in combination with others for stoping out pillars between or over filled stopes. Moreover, square-setting often is used as an auxiliary method to other methods, as for supporting the sill and first floors of some cut-and-fill or shrinkage stopes.

CAVED STOPES

When caved-stope methods are employed they will result sooner or later in breaks to or subsidence of the surface, provided the stoped area is large and thick enough in relation to the depth and character of the overburden. Therefore, caved-stope methods are applicable only where there is no objection to caving of the overlying strata and subsidence of the surface.

BLOCK CAVING

In the caving methods the ore itself is broken by caving and by the weight and movement of the mass of caved ore. In the block-caving method each section of the ore body, constituting a caving block, is partly isolated from the surrounding ore or walls by boundary drifts and raises or by boundary stopes, and the supports beneath it are either cut away entirely or are weakened so that the block will crush them. The ore must be of such a character that as it caves and moves downward after the support and broken ore are removed from the bottom it will break up finely enough to permit drawing it off through raises (or drifts) below. Block caving is adaptable to ore which behaves in this manner and which occurs in deposits of large area and considerable thickness. The method is usually applied only to mining low-grade ore, as will be pointed out later.

SUBLEVEL CAVING

In sublevel caving the ore is stoped partly by excavation and partly by caving. It is mined in a succession of small, caved stopes, usually 18 to 30 feet high, beginning at the top of the deposit and working downward. The method may be applied to smaller deposits than block caving but is also applicable to large deposits and to soft ores which will stand fairly well for a brief time over short spans with light timber supports but will cave when an appreciable area is undercut. The capping (and later the gob) should hang up over short spans for a long enough time to permit removal of the caved ore beneath in safety and without undue dilution of the ore with waste. Schaus [5] has well summarized the requirements for sublevel caving as it is applied in the iron mines of the Gogebic range, as follows:

> The determining conditions to which this method is adapted are: (a) A dipping and pitching irregular ore body which does not lend itself to top slicing; (b) a medium-soft ore which breaks fine yet stands well and is not free caving; (c) a hard capping which caves in medium-sized blocks without much fines and which is easily controlled.

It can be applied where top slicing would be dangerous due to the hanging up of the gob. In block caving the broken ore must be free-running; that is, it must be of such a nature that it will not pack and form a semisolid mass which cannot be drawn off from below. Ore of this character usually is hard. On the other hand, due to the relatively thin back and small units caved at one time in sublevel caving, packing can be avoided when using this method, and it is therefore applicable to softer and somewhat stickier ores than is block caving. It also is applied to stoping some hard ores, however.

TOP SLICING

In top slicing the ore is removed by excavating a series of slices, one alongside the other, beginning at the top of the ore body. (See fig. 11.) After the ore is removed in slices the overburden is caved. Slices are taken in the same way at successively lower levels, the ore being removed up to the caved gob. The method is applicable to mining soft, weak ore which will stand unsupported for only a very short time even over spans of a few feet and which is overlain by an unconsolidated overburden or weak capping which will break up and cave as soon as the support is removed, tightly filling the space formerly occupied by the ore and leaving no open holes. It may be employed for mining tabular deposits lying at all angles of dip from flat to vertical, or in wide, thick ore bodies. It is used successfully in very irregular deposits. Top slicing may be employed for extracting pillars from between caved or filled stopes and for mining broken ore or old caves which would be unsafe to work from the bottom upward. The method is similar to sublevel caving in that the ore is mined from the top downward by a series of slices and differs from it in that the sublevels are not so far apart and the ore is excavated up to the capping or the mat above, whereas in sublevel caving only part of the thickness of the slice is excavated, the rest being caved down on the retreat.

SELECTION OF STOPING METHOD

It has been pointed out in the foregoing discussion that the selection of a stoping method is concerned fundamentally with the requirements as to support of the stopes. In a given instance any

[5] Schaus, O. M., Mining Methods and Costs at the Montreal Mine, Montreal, Wis.: Inf. Circ. 6369, Bureau of Mines, 1930, 20 pp.

method by which support will fail before the ore can be removed in supported stopes or by which the ore will not cave and break up finely enough in caving methods may be disregarded at once.

Sometimes selection of the stoping method best adapted to the conditions may be comparatively simple due to the fact that only one method can be applied or that the choice lies between one of two methods or variations of one method. At other times the problem is more complex, and the best method can be selected only after full consideration and careful weighing of a number of interrelated factors. Safety of operation and the health and welfare of the workmen are of course primary considerations; then come factors of several different types.

First are those related to the purely physical characteristics of the ore deposits and of the surrounding rocks and surface, which have been discussed briefly as they affect support but which also must be considered in relation to other factors. The important physical characteristics that affect the choice of a stoping method are: (1) Strength of ore and wall rocks; (2) shape, horizontal area, volume, and regularity of the boundaries of the ore body; (3) thickness, dip and/or pitch of the deposit and individual ore shoots; (4) continuity of ore within the boundaries of the deposit; (5) depth below surface and nature of capping or overburden; (6) position of deposit relative to surface improvements and drainage.

Second are factors related to the nature of the ore itself, particularly its grade and unit value, hardness and friability, percentage of ore recoverable by the stoping method adopted, distribution of the valuable minerals within the ore body, and effect of the stoping method upon the percentage recovery of valuable minerals in the milling plant.

Third and closely related to the type and grade of the ore are economic factors, which involve consideration of the cost of stoping compared to the value of the ore, capital available, production rate, availability and cost of mine timber, possibilities for quick expansion or contraction of production rate, and degree with which the grade of ore mined may be adjusted to market conditions without loss of ore and consequent impairment of the value of the mine.

Fourth are factors that affect related mining operations, such as development, haulage and hoisting of ore, procurement and handling of filling material, disposition of waste rock, and handling timber, drills and drilling gear, drill steel, and explosives.

These several types of factors are interrelated, and in selecting a stoping method it is usually necessary to consider the influence of several factors and finally to disregard those of less importance to satisfy the requirements of more important ones.

PHYSICAL CHARACTERISTICS OF ORE DEPOSIT

The physical characteristics of the ore deposit include the broader physical aspects of the deposit rather than the composition of the ore itself. The effect of these characteristics upon the method of support and hence upon the stoping method has already been discussed.

In this connection the strength of the ore and walls and the size, horizontal area, and shape of the deposit have been shown to have paramount importance. A caving method cannot be employed if the horizontal area is so small in proportion to the strength of the ore that the ore will not cave when support is removed from beneath it. On the other hand, if the ore body is of large area the difficulty of providing temporary and permanent support is greater, and some sort of support in the form of pillars, filling, or timbering usually is required if a supported-stope method is employed. It has been pointed out that filling usually will provide ample side support to the walls but cannot be placed so as to afford much support to the back. The area of back that will stand permanently over a filled stope therefore is limited by the weakness of the back, and the larger the area the less is the probability that filling will support it adequately. Filled square-sets which catch up the back are much more effective than filling alone or square-setting alone.

In very thin, tabular deposits the size influences the stoping method quite apart from its effect on support. The width between walls may be too small to provide room in which to work, and it may be necessary to break into the walls to provide stoping width. If this is done the waste from the walls either goes out with the ore, reducing the grade of the material taken out of the mine, or is kept separate. In shrinkage stoping it cannot be kept separate. If the vein is flat the waste may be thrown to one side unless the hanging wall is so bad that space cannot be kept open in which to store it. If the vein dips at a steep angle either a regular cut-and-fill or a resuing system may be employed. In the ordinary cut-and-fill system ore and wall rock are blasted down together, the waste is sorted out and left in the stope for filling, and the ore is sorted and thrown into the ore chute. This can be done only where the appearance of ore and waste is quite different, so that they can be separated. Furthermore, unless the wall rock breaks into large pieces it cannot be picked out, and any fine waste will have to be sent out with the ore. In any event hand picking is expensive and slows down the rate of production from the stope.

In resuing (or "stripping") either the ore is broken down first and then the waste, or vice versa; usually the one which breaks easier is blasted first. The broken waste is left in the stope as filling, and the ore is broken down on flooring laid on the fill to prevent admixture of ore and waste. Resuing is applicable where the ore is not "frozen" to the walls and works best if there is considerable difference between the hardness of the ore and of the wall rocks.

If the ore boundaries are very irregular a flexible stoping system should be employed, by which irregularities in the ore may be followed, and the ore can be recovered from tongues and offshoots from the main ore mass. Thus shrinkage stoping might be applicable as far as stope support is concerned but might not be flexible enough to permit recovery of the ore from small shoots or irregular masses of ore projecting outward from the main ore body. Likewise, the use of a block-caving method in an ore body having very irregular boundaries would result either in the inclusion of considerable wall rock with the ore or in the loss of ore which would have to be left behind to avoid dilution with waste rock. However, if such a deposit were of large horizontal area the boundaries might be mined by a supported-stope method and the main mass caved.

The thickness and dip or pitch of the deposit and of the ore shoots have an important bearing upon the method of getting the broken ore out of the stopes, which in turn influences the stoping method. In this respect thickness and dip may be considered as similar characteristics. A thin, tabular ore body dipping at a steep angle furnishes a condition similar to that of a thick body of large horizontal area, since in both instances the vertical dimension of the ore is large and gravity can be employed to move the ore from the stope to the haulage level.

In a thin, tabular deposit dipping at an angle lower than the angle of repose of the broken ore, the ore in a shrinkage stope will hang up and cannot be drawn off. Near this critical angle the ore may hang up in places, leaving large openings over which hung-up ore may collapse suddenly and endanger the miners. In either case incomplete recovery of the broken ore would result with shrinkage stoping. To overcome this difficulty several alternatives could be employed, the one to be adopted depending largely on the strength of the hanging wall. If the hanging wall is firm an open-stope method is indicated, and the ore would be moved to the level by mechanical means, such as scrapers, go-devils, or conveyors. Either casual pillars or a series of regular pillars might be employed to reduce the unsupported span, or stull timbers could be used, spacing them at intervals so that the ore could be moved between them. If the hanging wall is bad a cut-and-fill or square-setting method might be applicable, in which the ore would be lowered to the level through chutes carried through the fill or in lines of square-sets kept open for this purpose, or it could be trammed in the stope to raises put up in the walls at an angle steep enough for the ore to run in them by gravity. With favorable hanging-wall conditions deposits of this type sometimes are worked by a retreating longwall system, in which the longwall face is carried at right angles to the strike (up the dip) and the hanging wall is allowed to cave or settle down behind the face as it retreats.

The pitch of individual ore shoots acts similarly to the dip of the vein in that a flat pitch prevents gravity movement of the ore from the stope face to the level unless steep raises are driven in the wall for transfer of the ore.

Wide variations in the angle of dip preclude the use of an inflexible stoping method. Thus a shrinkage method starting from the level on a steep pitch might work well until the vein flattens some distance above the level, at which point the broken ore would hang up.

The continuity of ore within the boundaries of the deposit has an important bearing upon the stoping method, particularly if continuity is considered with respect to the presence and distribution, or absence, of waste inclusions or "horses" within the deposit and the existence or absence of local offsets in the ore due to faults of small displacement. Alternating bands of ore and waste have a similar effect, although the method of overcoming the difficulty presented may differ.

Occasional large horses of waste can usually be left in place, and the ore can be stoped out around them with any supported stoping method. When they occur frequently and are relatively small individually the problem is whether to leave them in place, take them out separately and leave the resulting broken waste in the stope for filling, or send the waste out of the mine separate from the ore; or, if they cannot be kept in place but break down with the ore, to remove the broken waste with the ore, thus diluting it, or sort the waste from the ore. If shrinkage stoping were used under these conditions, leaving horses as pillars would cause some of the ore to hang up, with attendant loss of ore and danger to the miners, and if they are blasted out or fall out there is no opportunity to separate ore from waste and keep them separate in the stope. In narrow veins dipping at steep angles, where horses of waste break away from the walls and the walls are fairly strong, waste may be kept separate from the ore by sorting it out, or breaking it separately, and storing it on lagged stull timbers.

Sometimes tabular ore bodies are offset by a series of oblique or strike faults of small displacement. This may result in sections of the vein being thrown out over the back of lower sections, and in mining upward from the bottom the back of the lower sections may be weakened, and the footwall side may break back to the underlying stope when the upper sections are mined. If the displacement along the fault is in the reverse direction, the offset in the stope may result in caving of the lower portion of the stope and resultant dropping of the hanging wall in the upper portion. Such a condition existed at a mine with which the senior author is familiar and resulted in abandonment of shrinkage stoping in favor of cut-and-fill stoping in the narrower stopes and square-set stoping in the wider stopes.

Where the ore consists of alternating bands of ore and waste the choice lies between mining the entire body as a unit from wall to wall and removing ore and waste together, sorting ore from waste in the stopes, or breaking ore and waste separately. If there are several thin bands of ore and waste, it usually is not feasible to break ore and waste separately, and the ability to sort in the stopes will depend upon the size of the pieces into which the rock breaks and whether the ore and waste differ in physical appearance.

Depth below surface, nature of capping and overburden, and position of the deposit relative to surface improvements and drainage are factors that affect support. If the deposit is large and close to the surface, the choice may lie between underground or open-pit methods. However, this bulletin is not concerned with open-pit methods.

GRADE OF ORE

The grade of ore or value per ton has an important bearing upon the stoping method and must be considered in relation to percentage recovery of ore in the deposit and cost of mining.

If the ore is low grade, considerable may be left in the mine in the form of pillars, and the same care need not be exercised to recover a high percentage as would be advisable in mining a high-grade deposit if such a practice will effect appreciable savings in cost. To be more precise, if the added cost of stoping by a high-recovery method aggregates more than the value of the additional ore recovered then a lower recovery method would in general be the more profitable one to use. However, the unit values of the ores change with market conditions, and if ore is left in the mine during periods of low prices so that it cannot be recovered later a considerable loss may result during periods of higher prices when it might pay to go to greater expense to obtain a greater percentage of the ore.

With ores shipped as mined without any beneficiation or milling process the inclusion of appreciable waste with the ore in the process of stoping usually reduces the selling price considerably and increases the shipping cost by the freight paid on this valueless material. A clean method of stoping would be preferred under these conditions. Iron ore is a good example of this, as penalties in price increase with the content of silica and other impurities over a certain amount. High-grade pyrites furnishes another example. A number of nonmetallic ores are also in this class.

In caving methods of stoping there is always some loss of ore and some dilution of ore with waste. Where top slicing is applicable high recovery and clean ore can be obtained by this method. With shrinkage stoping, ore will be lost and diluted with waste if the walls are irregular or slough off readily. With cut-and-fill stoping a high recovery can be obtained, irregularities in the ore body can be followed to a considerable extent (but not so well as with square-setting), and waste can be sorted and kept separate from the ore in the stopes. With open stopes, selective mining is possible, lean or barren sections often can be left in as pillars, and there is considerable opportunity for sorting waste from the broken ore, especially in deposits dipping at low angles.

DISTRIBUTION OF VALUABLE MINERALS WITHIN ORE BODY

There are five principal types of distribution from the standpoint of their effect on the stoping method: (1) Uniform dissemination of valuable mineral particles throughout the deposit; (2) erratic mineralization with the valuable minerals concentrated in bunches throughout the deposit like plums in a pudding, the surrounding mass being low in grade or barren; (3) bands or irregular and disconnected stringers of high-grade ore alternating with bands or stringers of waste rock; (4) deposits in which there are two or more grades of ore, or two or more classes of ore, distinguished from each other by the predominance of minerals of different metals, the different grades or classes of ore being associated rather closely; and (5) deposits in which the boundaries are gradational—that is, where there is no sharp line of demarcation between the ore and walls but the ore decreases gradually in value toward the boundaries of the ore body.

1. Where the valuable mineral is distributed uniformly throughout the ore body, a flexible selective stoping method is not essential. In large deposits of this type, which commonly are of low grade, such methods as block caving, sublevel caving, and modified shrinkage stoping or in fact any method which will fulfill requirements as to support and give the lowest operating costs often can be employed.

2 and 3. In deposits where the mineralization is erratic and is of types 2 or 3 the first question is whether to employ a selective or a nonselective stoping method. In selective stoping the stopes are confined to the bunches or lenses of ore, and care is taken to mine as little waste or noncommercial material as possible. In nonselective

mining the deposit is mined out from wall to wall and the ore is separated from waste on the surface by hand sorting, milling, or both.

In selective mining the object is to obtain a relatively high grade mine product; this usually entails the use of a much more expensive stoping system and high exploration and development costs in searching for and developing the separate bunches, stringers, lenses, and bands of ore. If the value of the ore recovered will bear these combined costs a selective method is worthy of consideration. In general, selective methods are applicable where the valuable sections of the deposit are rather large, comparatively few in number, and separated by relatively large volumes of waste. Under these conditions removal of the intervening bodies of waste would so dilute the ore that it would be too low grade to handle and treat.

In nonselective mining the object is to secure a low cost, generally by using a cheap stoping method combined with large-scale operations. This method can be used in deposits where the individual stringers, bands, or lenses of high-grade ore are so numerous and so irregular in occurrence and separated by such thin lenses of waste that a selective method cannot be employed. A very good example of this practice is furnished by the Alaska-Juneau method of mining, which will be described later.

The selective methods of stoping are: Square-set stoping, open stoping in low-dipping beds, and cut-and-fill stoping. Some variations of cut-and-fill stoping are more selective than others. Nonselective methods include: Caving methods, top slicing, some forms of open stoping, and shrinkage stoping under most conditions.

4. In deposits containing two or more classes or grades of ore selective stoping methods are employed if for commercial and economic reasons it is desirable to keep the various classes of ore separate. If the different classes of ore are very closely associated so that changes from one to another are abrupt, a high degree of selectivity is necessary. Square-set stoping and some types of open stoping are applicable. Horizontal cut-and-fill stoping may be used by providing separate chutes for each class of ore, but this method is not as flexible as square-set stoping. Shrinkage stoping does not permit sorting in the stopes.

5. In deposits with gradational boundaries a nonselective method may be employed for extracting most of the ore if the ore body is wide and uniformly mineralized except at the boundaries. If the marginal low-grade material is thick compared with the commercial ore it is advisable to employ a method whereby the low-grade material can be stoped later if market conditions change to such an extent that this becomes of commercial grade. If a caved-stope method is employed it would usually be impossible, or at least unprofitable, to attempt to return and mine the marginal ore. A cut-and-fill, or filled square-set, method would allow a return to the old workings for recovery of this ore, but the cost of these methods might be prohibitive during the earlier mining or be unjustified by the probability of the marginal ore ever becoming of commercial grade. Where a nonselective method is cheap and milling is not expensive it may be desirable to employ such a method, taking out the best of the marginal ore during the first mining and deliberately abandoning the possibility of ever going back to stope for the balance of it.

EFFECT OF STOPING METHOD ON MILL RECOVERY

The mining method may affect the percentage of valuable mineral recovered in the milling plant, especially when certain sulphide ores are treated by flotation. Some sulphides oxidize rapidly upon exposure to the air, and if left broken in the stopes for only a short time (sometimes as little as 2 or 3 days) the sulphide particles will acquire a thin film of oxide which will interfere with their recovery in the flotation plant. In such cases it is necessary to employ a stoping method by which the ore can be removed from the stopes and sent immediately to the mill as fast as it is broken. In shrinkage stoping the broken ore usually remains in the stope for a long time, and the sulphide particles will film if they are susceptible to oxidation. Under certain conditions partial oxidation of sulphide minerals may occur in a column of caved ore in block caving to an extent that will reduce recovery in the flotation plant.

With other methods of mining the ore can usually be removed as fast as it is broken.

ECONOMIC FACTORS

The relation of some of the economic factors to the physical characteristics of the ore deposits and the grade or value of the ore already has been discussed.

COST OF STOPING

Obviously if the value of the ore will not cover the cost of mining, treatment, and transportation and leave a reasonable profit a mine cannot continue to operate; in general, it may be said that a low-cost stoping method is essential for mining low-grade ores.

The cost of stoping is only one of the items in the cost of mining (which, in turn, is one of several cost items in the total cost of ore production); nevertheless it is one of the largest, and often the largest, single item. In this connection the cost per ton of ore stoped is important but not as important as the total cost of production (including mine operations, milling, transportation, and overhead) per unit of metal recovered, as it is not tons of ore but the contained metal which the buyer pays for. A stoping method which permits a low mining cost per ton of ore may result in a higher total production cost per unit of metal recovered than a stoping method which has a higher mining cost per ton of ore. The low-cost-per-ton method may so dilute the ore with waste that the combined cost of breaking, handling, tramming, hoisting, crushing, and milling this waste may be greater than the saving in stoping cost over a higher-cost-per-ton method. If the ore is shipped direct to a smelter without first being concentrated the freight on the barren material must be added to the cost per unit of metal, and penalties may be imposed by the smelter for treatment of ores containing considerable impurities.

CAPITAL AVAILABLE

The amount of capital available may be important in determining the method of stoping to be employed. Some methods, particularly block caving, require large capital expenditures for general development and stope preparation before ore production from the stopes is begun. Open stoping and simple types of shrinkage stoping require comparatively small preliminary expenditures for stope development, as stoping can be begun almost as soon as the development levels have penetrated the ore bodies. Sublevel stoping requires somewhat larger preliminary expenditures than other types of open stopes. Modified shrinkage methods, such as the method employed at the Alaska-Juneau mine, require greater expenditures for development than the simpler shrinkage systems.

CAPITAL REQUIREMENTS

Capital requirements and rate of production are closely related. If a mine is to be developed for a high rate of production, capital expenditures for equipment and mine development on a scale adequate to obtain such a rate obviously will be proportionately large. Low-grade ores usually must be mined on a large scale to obtain costs low enough to insure profitable mining, and there is no escaping large capital expenditures.

Factors not related to the method of stoping or rate of production may also make large capital expenditures necessary. Thus, if the ore bodies lie at great depths or are scattered so that a large amount of lateral development is necessary to open them up, considerable capital will be required to place the mine on a productive basis.

If capital is limited it may be necessary to adopt a stoping method that will bring the mine into production quickly, with the smallest possible initial outlay. For example, limited capital may make it imperative to employ shrinkage stoping even though sublevel stoping, cut-and-fill stoping, or some other method is better suited to the conditions, would result in lower costs, or would recover a greater percentage of the ore in the deposit with less dilution. Moreover, lack of capital might preclude the use of a cheap, nonselective mining method otherwise applicable to the deposit, because of the high cost of preliminary development and of an ore-dressing plant to separate waste from the ore on the surface. Frequently an open-stope method is employed in small mines in preference to shrinkage stoping, because the immediate demand for revenue does not permit tying up broken ore in the stopes.

COST OF MINE TIMBER

Availability and cost of mine timber may influence the choice of a stoping method in some districts. Thus, if timber is expensive, cut-and-fill stoping may be chosen instead of square-set stoping, although the latter method might permit a higher percentage of ore extraction. Again, the nature of the timber available locally might influence the choice between square-setting and top slicing. In top slicing the stope timbers remain in place only a short time and usually need not be of as good a grade as for square-setting; top-slice timber may be available locally at small cost, whereas timber suitable for square-setting may have to be shipped in and may be costly.

MARKET CONDITIONS

When changes occur in market conditions a mining method flexible enough to meet these changes promptly is an advantage. Changes in conditions may call for sudden expansion or contraction in the scale of operations, for changes in the grade of ore mined, or for both. A method that will meet changes in rate of output best may not permit the desired changes in grade of ore mined satisfactorily.

Large expansion of output in mines employing slow stoping methods, such as cut-and-fill stoping and square-set stoping, can be brought about only by keeping a large amount of development ahead of actual stoping, so that if a sudden increase in production is demanded additional stopes may be brought into production quickly. The same is true with the simple forms of shrinkage stoping, with top slicing and sublevel caving, and to a smaller extent with other methods. Advance development naturally ties up capital in unproductive workings and may entail considerable additional maintenance cost.

With some stoping methods production from individual stopes can be forced when desired. Thus, with sublevel stoping, it is possible to work a number of machines on each of the benches in the stope at the same time, whereas ordinarily only one machine would work on each bench, or only two or three benches in a stope would be worked simultaneously. In other types of open stopes, particularly if they are large, production can be forced by adding to the number of machines normally employed in each stope. With block caving best results are obtained with a regular rate of drawing; the rate should not be so rapid that it causes piping through of waste or that the caved ore does not have time to break up before reaching the draw points, nor should it be so slow that excessive weight will be thrown on the extraction openings and thus result in high maintenance costs. Within the limits prescribed by these considerations expansion of output can be forced to some extent with block caving.

With most mining systems, particularly those employed in heavy ground, a sudden contraction in rate of output is apt to bring more serious consequences than an expansion. If output is to be curtailed by reducing the number of active stopes, a number of partly mined stopes must be left standing. With some methods of stoping and in some classes of ground idle stopes will cave, with attendant loss of ore or dilution of ore with waste, and later reopening may be expensive and dangerous unless considerable work has been done in putting them in shape for a protracted shut-down. Furthermore, underlying and adjacent development and extraction openings may require considerable maintenance during the idle period.

If the output is curtailed by continuing to operate all the stopes on a reduced scale it may be possible to maintain them in good condition, but a small production scattered over a large number of stopes increases unit costs for haulage, handling materials and supplies, and maintenance.

Difficulties attendant upon curtailment of operations as it affects stoping and general condition of the mine usually are less serious in open-stope mines than in other types, since the walls and ore are strong and firm, and often the stopes will stand for years with little caving or slabbing.

Cessation of operations for a considerable time in uncompleted shrinkage, cut-and-fill, square-set, or caved stopes usually requires that more or less work be done so that the stopes will remain in good condition for resumption of operations later. The backs of shrinkage or cut-and-fill stopes may have to be caught up with props, regular sets, or cribs placed on top of the broken ore or filling. Square-set stopes are usually filled completely (except for the necessary ore and waste passes and manways) before operations cease, and some lining sets and braces may have to be put in.

In block-caving sections, which are being drawn, cessation of drawing is apt to result in the broken ore packing so that upon resumption of drawing considerable difficulty will be encountered, and extraction drifts and pillars between finger raises may crush if the suspension is prolonged.

With sublevel caving it is usually necessary to complete caving back in all slices in which it has been begun and to timber securely any intervening slice drifts. With top slicing it is necessary to continue active slices until no small pillars are left between them and the line between the cave and the solid ore is straightened out; then, if all the slices are blasted down and caved full, the mine will usually remain in good condition for a considerable time.

When wide fluctuations in the demand for ore are anticipated it is, therefore, wise to consider their possible effects upon the condition of the mine, and a stoping method which will be flexible with sudden expansion or contraction of operations is desirable.

The effect of changes in the prices of metals upon the grade of material which may be classed as ore has been touched upon in a previous section, where it was pointed out that a flexible stoping system which will permit reentry into low-grade sections of the mine during periods of high metal prices is desirable.

FACTORS AFFECTING RELATED MINING OPERATIONS

Stoping is closely related to other underground operations, such as development, haulage and hoisting of ore, procuring and handling filling, and handling timbers, drill steel, and other supplies.

DEVELOPMENT

It has been pointed out that some stoping methods require considerable stope development before the stopes can begin to produce ore and that, if capital is not available for such development, a stoping method requiring less preliminary expenditure may have to be employed.

Development and stoping also are related as regards the adaptation of the development plan to variations of each general stoping method. Thus, in a wide, steeply dipping vein the ore may be stoped by either of several methods, the ore being mined in longitudinal stopes taking out the full width of the ore or in a series of transverse stopes. In the former the haulageways will consist of one or more longitudinal drifts in the ore or in the walls on each level, whereas in the latter a series of extraction drifts driven across the vein under the stopes and connected to a main longitudinal haulageway will be required.

In other instances the general scheme of developing levels may be the same or similar, whether sublevel stoping, shrinkage stoping, cut-and-fill stoping, top slicing, or sublevel caving is employed, whereas the development details may be quite different. A simple shrinkage stope is usually developed by a single longitudinal drift on the footwall on each level, whereas with cut-and-fill stoping or square-setting it may be desirable to provide a separate hanging-wall drift for introduction of waste filling, particularly if the dip of the vein is flat. The shrinkage stope would be provided with a series of comparatively closely spaced chutes for drawing off the broken ore and with only occasional through raises to the level above for travel of men, handling of supplies, and ventilation; these raises are frequently put through from the back of the stope after stoping has been going on for some time. Cut-and-fill stopes and square-set-and-fill stopes usually require that at least one raise be put through to the level above for introducing filling before stoping is begun, and a number of such raises may be required for each stope, the distance between them depending upon the details of the stoping practice. The distance between ore chutes in cut-and-fill stoping varies with the details of practice but usually is greater than in shrinkage stoping.

Sublevel stoping, sublevel caving, and top slicing require considerable sublevel development, as does the sub-level-inclined variation of the cut-and-fill method, and the amount of this and of raise development will depend upon the details of stoping practice, as well as upon the size and dip of the ore body and the general stoping method employed.

The vertical distance between the main haulage levels also affects or is affected by the stoping practice. It usually is necessary to adopt a comparatively short level interval to prospect and develop erratic ore bodies or veins thoroughly, whereas if the ore bodies are regular and continuous from level to level the interval may be considerably greater. If the levels are close together the height of the stopes between them obviously is correspondingly short. If the stopes are begun some distance above the back of the level, leaving an arch pillar over the drift and under the bottom of the stopes, the effective stoping height is reduced further. If a substantial floor pillar also must be left below the level above, the stoping height will be reduced still more, and the cost of development per ton of ore stoped will be high. The effective stoping height, and hence development charges, are reduced by increasing the level interval, by stoping directly on level timbers, and by reducing the thickness of the floor pillars. Where the haulage drifts are in the footwall or the vein is narrow and has strong walls so that the stopes and tracks can be supported on stull timbers it may be possible to stope through from level to level without leaving arch or floor pillars, thus obtaining a maximum stoping height and a minimum of development per ton of ore stoped.

Sometimes a given stoping method may be applicable if the level interval is short which could not be used if the level interval were long. Thus shrinkage stoping might be practicable with a short level interval, as the stopes can be quickly completed before exposure to the air has had time to cause slacking and sloughing of the walls, whereas with a long interval it might be necessary to use cut-and-fill stoping because the stope walls would slab off and get heavy before the stope could be mined up to the top and the broken ore drawn off. For similar reasons the use of a rapid stoping method by which each stope can be developed and mined out quickly would permit a greater level interval in some instances than would a slow stoping method, thus reducing the amount of development per ton stoped.

Where top slicing or sublevel caving is employed several different methods of development can be used which are governed not only by the physical characteristics of the deposit but also by the method of handling the ore between the stope faces and the haulage level and by the method of driving the slice drifts. Some variations require only a comparatively small number of raises, while others require more. Likewise, the details of practice in sublevel caving largely determine the number of haulage crosscuts, main ore passes, and branch raises.

The foregoing discussion indicates briefly some of the ways in which stoping methods and their variations are related to mine development. The section on stoping costs includes a table showing the amount of development work required at typical mines and its cost per ton of ore produced by different stoping methods and under various conditions.

HAULAGE AND HOISTING OF ORE

For efficient operation, ore broken in the stopes must be trammed and hoisted as economically as possible. Up to a certain point concentration of stoping operations within a small section of the mine is desirable, but if carried to extremes it may result in congestion on the haulageways. Up to this point concentration of operations is favorable to low haulage costs, as haulage units are kept busy, ton-mileage is reduced, and supervision of haulage is facilitated.

Some methods, such as sublevel stoping, certain variations of large-scale shrinkage stoping, and block caving, are adapted to a high output from individual stopes. In tramming from such stopes an entire train or several trains in succession may be loaded from a single chute, whereas when the production comes from a large number of chutes or stopes scattered over a wide area considerable switching may be necessary to obtain a trainload of ore, which obviously adds to the haulage costs.

With some methods of stoping in which production is intermittent or only a small production can be obtained daily from each stope, a large number of stopes is necessary to maintain a high rate of production from the mine. Thus in some variations of cut-and-fill stoping production from each stope is intermittent, and no ore is drawn while the stope is being filled. In other variations of this method, breaking and removal of ore and filling are continuous operations, one operation going on in part of the stope while the others are being conducted in other parts.

In shrinkage stoping only about 40 percent of the ore is drawn as it is broken, and usually only a small amount of ore is drawn daily from each chute before the stope has been completed and final drawing of the broken ore is begun.

Where tramming distances are short and only a small tonnage is drawn from each stope at one time hand tramming often is the cheapest method. On the other hand, where large tonnages must be drawn daily from individual stopes or where the haulage distances are long, some form of mechanical haulage is usually the most economical.

Some stoping methods require the ore to be drawn frequently but only in small amounts at a time, as in shrinkage stoping prior to final drawing, or in top slicing and sublevel caving after the active slices have reached an elevation only a short distance above the haulage level and the capacity of the ore chutes is therefore small. In the latter instance, the chutes must be kept empty so that operations will not be retarded in the slices.

When ore is hauled to the shaft in large trains and the hoist is taxed to handle the required tonnage, shaft pockets of ample capacity must be provided for storage, so that the trains may be dumped quickly and the empty cars returned to the stopes and so that hoisting will not be delayed.

From the foregoing it appears that the stoping method may determine the method of tramming and the hoisting facilities needed and affect the cost of haulage and hoisting.

FILLING MATERIAL AND SUPPLIES

Where cut-and-fill or square-set-and-fill methods are employed filling must be available when and in the amounts needed. In some instances enough waste may be provided from sorting in the stopes and from development headings in rock. In other instances filling from these sources may have to be augmented by waste material from special waste stopes, surface quarries, surface gravel pits, or mill tailings, or it may have to be supplied entirely from such sources. In deciding upon a filling method of stoping, therefore, it is important to consider the available sources of filling material. Sometimes a source of cheap filling material may be readily available, but the procurement of filling material has been a serious problem at some mines.

When filling is procured from outside the stopes its transportation and handling to and in the stopes sometimes are problems. The haulage of waste and ore through the same drifts may complicate haulage since, unless special provisions are made, tramming of waste may interfere with tramming of ore and vice versa.

Handling timber and supplies in congested haulageways may further complicate underground transportation and hoisting. If a large amount of timber is required due to the stoping method in use, it may be necessary to provide a special timber shaft, or a special timber compartment and cage in the main hoisting shaft, to avoid delays and congestion.

SAFETY, HEALTH, AND WELFARE

The largest single cause of serious accidents in underground metal mines has been falls of ground. Prevention of accidents from this cause is possible only if the stoping method affords adequate support for the ground and protection to the workmen at the working face. This point was stressed previously when it was pointed out that the basic factor in selecting a stoping method is stope support. Accident-prevention work does not cease, however, with the adoption of a safe stoping method, as accidents will occur with any stoping method unless safe practices are employed in every detail of the work. Nevertheless, if the general stoping method is not the safest one to use in a particular instance accident prevention is impossible, no matter how much attention is given to details of practice.

It is not within the scope of this paper to discuss the safety, health, and welfare of the workmen, and much that will occur to the reader on these subjects will remain unsaid. Their importance is recognized by all operators today, and it is trite to say that the welfare of the workmen and their families depends upon safety and health in their occupation. It therefore is necessary in selecting a stoping method to consider its effect on the safety and health of the men.

The stoping method may affect not only the accident rate but also the miners' health. If the mining method and details of practice entail poor ventilation at the working face the health of the men will be impaired by work in air deficient in oxygen, high in temperature, laden with harmful gases produced by blasting, and carrying a high dust count.

Good ventilation can be obtained with any of the principal stoping systems but can be provided more easily with some methods than with others. Open stopes are usually easy to ventilate, and ordinarily natural ventilation is all that is required, except perhaps at great depths below the surface. Shrinkage stopes, cut-and-fill stopes, and square-set stopes often are not ventilated as easily as others, but good ventilation can be obtained by means of an adequate number of openings large enough to permit circulation of air from the level below the stope to the stope face and from the stope face to the level above. Forced ventilation may or may not be required.

In block caving, ventilation must be supplied to the boundary shrinkage stopes if such stopes are employed, to the undercutting level, and particularly to the grizzly level and draw points where excessive dust often is produced. Suitable arrangements can be provided for circulation of air in all these places, and forced ventilation may or may not be required.

Slices in sublevel-caving and top-slicing methods are perhaps the most difficult faces to ventilate. Fortunately these methods have found their greatest application where the mines are comparatively cool (except where the mines are deep and a thick timber mat which generates heat has accumulated) and the ore is damp and produces comparatively little dust. In caved slices there is no direct outlet from the face to the level above, or other separate return for the air coming to the face, and fans may be required to blow air into individual faces.

SUMMARY

From the foregoing it is evident that although, after considerations of safety and health of the miners, support is the basic factor in selecting a stoping method the importance of many other factors must be considered and weighed before the best method or methods for any particular ore deposit can be determined. These factors are interrelated and frequently present a complex problem.

DISCUSSION OF DIFFERENT STOPING METHODS WITH EXAMPLES OF PRACTICE

The different stoping methods and their variations can be described and discussed best by presenting examples of practice, together with cost and unit data obtained from actual mining operations.

In this section each principal stoping method will be described, illustrated, and discussed separately, its inherent advantages and disadvantages will be pointed out, and examples of practice and cost data will be given, with the mining conditions under which the method is applied.

OPEN STOPING

The simplest form of stope is the open stope, in which the entire ore body is excavated from wall to wall and from top to bottom without leaving any pillars of ore, as shown in figure 1 (p. 5), which depicts a small, nearly horizontal ore body with irregular outlines. Tabular ore bodies dipping at steep angles may be mined from wall to wall and from bottom to top, but in such deposits the details of operation differ somewhat and are not as simple as in flat deposits. In steep ore bodies stulls and staging must be installed for the men to stand on while drilling.

In flatter, tabular deposits handling of the broken ore is an added problem, and some support may have to be supplied to the hanging wall, as a large area of hanging wall may be exposed, even though the ore body is thin. Such support may be supplied by pillars or stull timbers.

In large, horizontal or flatly dipping, irregular deposits where pillars are left to support the backs of the stopes the ore is stoped as in small deposits of the type shown in figure 1, each area between pillars being equivalent to a small single stope similar to that shown in the figure. Figure 3 (p. 8), in which the black areas represent pillars, is a plan of such a deposit showing the stoped areas with the boundaries of the ore body and the pillars.

OPEN STOPING IN HORIZONTAL OR LOW-DIPPING ORE BODIES

Deposits dipping at angles so low that broken ore will not run by gravity on the footwall are placed in this category. Often development and preparation for stoping in deposits of this type, mined by open-stope methods, are quite simple. The outlines of the ore bodies usually are at least partly determined by drilling or test pitting from the surface before underground work is begun, and their development requires merely the sinking of a shaft in or near the ore to the bottom of the ore body and crosscutting to the ore if the shaft is outside the ore boundaries. As soon as the ore is penetrated a short distance stoping can be begun, and further development is carried on simultaneously with the stoping, or the ore even may be developed from the stope faces as they are advanced.

When the ore is not more than about 12 to 14 feet thick it is usually stoped out from top to bottom in one slice or breast. In thicker ore it is customary first to take out a slice or heading 7 or 8 feet high directly under the top of the ore and then to bench or stope down the ore between the bottom of the heading and the bottom of the ore or floor of the level. The heading is kept a short distance in advance of the bench or stope. This system is termed "heading and bench" or "heading and stope" mining. Some stopes have been mined in this manner to a height of 200 feet. Where the ore is of medium thickness (14 to about 20 feet) a slice sometimes is breasted out first along the bottom, then the upper half of the ore is drilled and blasted by standing on the pile of broken ore from this breast. The heading and bench system is the most common method, however.

When the ore is thin enough so that it can be taken from floor to roof in one slice the stope is advanced by drilling and blasting like the heading in the heading-and-bench system.

Figure 12 shows details of typical heading-and-bench stoping practice. Figure 12, *A*, shows stope or bench holes drilled flat from the stope face. This is the usual practice in the Tri-State lead and zinc district where the ground is seamed, vuggy, and fractured and where flat holes are more easily drilled and cleaned than vertical holes and fewer holes are lost due to stuck steel.

Figure 12, *B*, shows stope or bench holes drilled nearly vertical from flat benches. This practice is typical of the Southeast Missouri lead district.

Figure 12, *C*, shows vertical bench holes where the heading is driven in steps. Figure 12, *D*, pictures a heading and stope in the Tri-State lead and zinc district shown diagramatically in figure 12, *A*.

Where flat benches are carried, as in figures 12, *B*, and 12, *C*, and holes are drilled vertically, muck blasted from the heading will pile up on the benches, and the latter must be cleaned off before the bench holes can be drilled. Where the holes are drilled as in figures 12, *B*, and 12, *D*, the splitter holes usually can be drilled without much cleaning of the stope face while the muck is being shoveled out from the toe of the stope. The drilling system to be used, however, depends principally upon the way the ground drills and breaks. Unless the character of the ground is such that bench holes are impracticable they usually are cheaper to drill, as hand-held pluggers can be used and no drill mounting is required.

The broken ore is loaded into cars and trammed to the shaft. It is loaded by hand shoveling, power shovels, or power scrapers. If the tonnage of ore broken by each stope round is small hand loading may be the most economical method, in which case loading tracks must be moved or lengthened frequently to keep them close to the muck pile. Figure 3 (p. 8) shows the track layout in a Tri-State district mine.

Where enough muck is broken to keep a loading machine busy power loading usually is cheaper than hand loading, unless labor is very cheap or much sorting of waste from the ore is required. If shovel-type loaders are employed the tracks must be kept close to the muck pile, and unless the stope is very high they will have to be moved or lengthened frequently. If scrapers are employed trackage only need be changed occasionally, as the ore can be dragged economically to a central loading point from faces within a radius of 150 to 200 feet or sometimes considerably more. First cost of scraping equipment is usually only one-half to one-fifth as great as that of shovel loaders, while power and labor costs are usually about the same. The largest single item in scraping cost, aside from labor, is that of replacing drag cables; other maintenance costs are usually lower for scraping equipment than for power shovels. However, some operators prefer power shovels to scrapers for the particular conditions under which they operate.

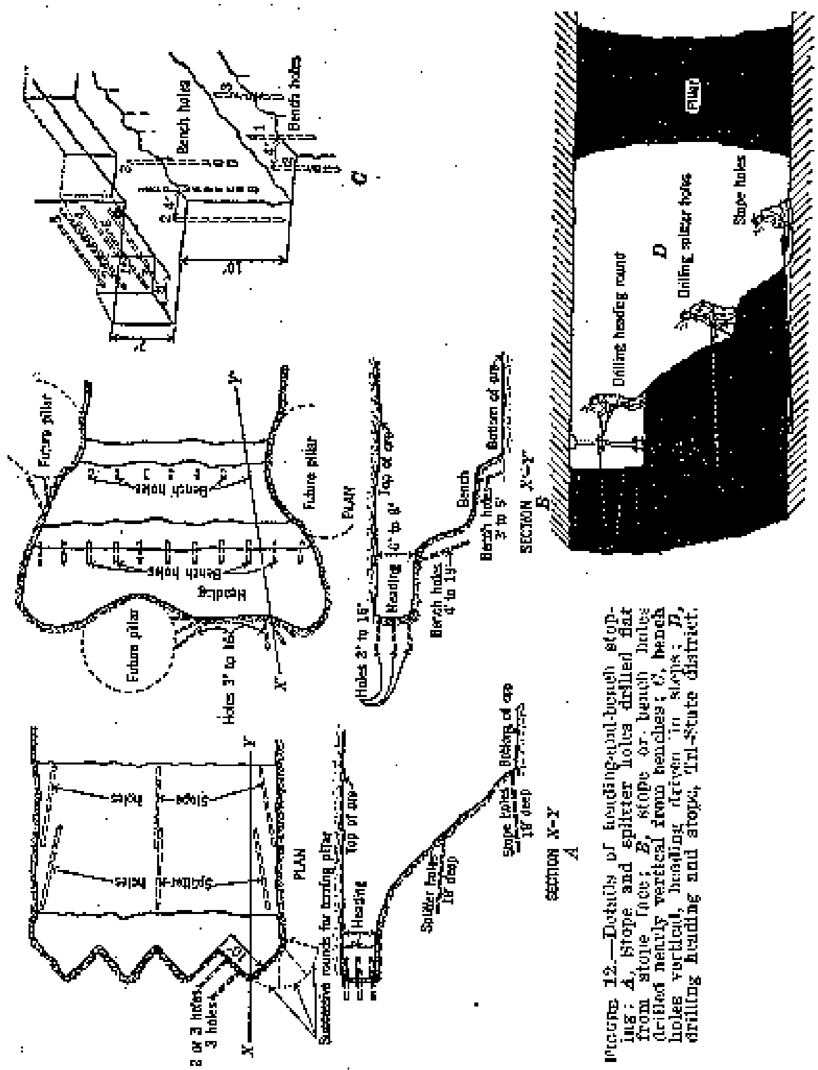


FIGURE 12.—Details of heading-and-bench stoping: A, Stope and splitter holes drilled flat from stope face; B, stope or bench holes drilled nearly vertical from benches; C, bench holes vertical, heading driven in stope; D, drilling heading and stope, Tri-State district.

Hand loading generally has been preferred in the Tri-State district, although one of the earliest scrapers for underground work in the United States was developed there. A high output per man has always been obtained by hand loading, and as many of the mines have had a short life the outlay for power loading equipment has been considered unjustified. In some of the larger and newer operations of the district power loaders or scrapers have been installed or their installation is planned, and with a revival of mining more mechanical loading probably will be done in the future than in the past.

In the Southeast Missouri lead district most of the loading is done by power shovels, which are preferred to scrapers in that region. At Mascot, Tenn., scrapers are preferred to power shovels, and very little hand loading is done.

Where power loading is practiced efficient operation depends upon adequate car supply and transportation facilities to avoid delays in loading; large cars and mechanical haulage are preferred.

Figure 13, *A*, depicts an electrical shovel loading from the toe of a stope in the Tri-State district and figure 13, *B*, hand loading

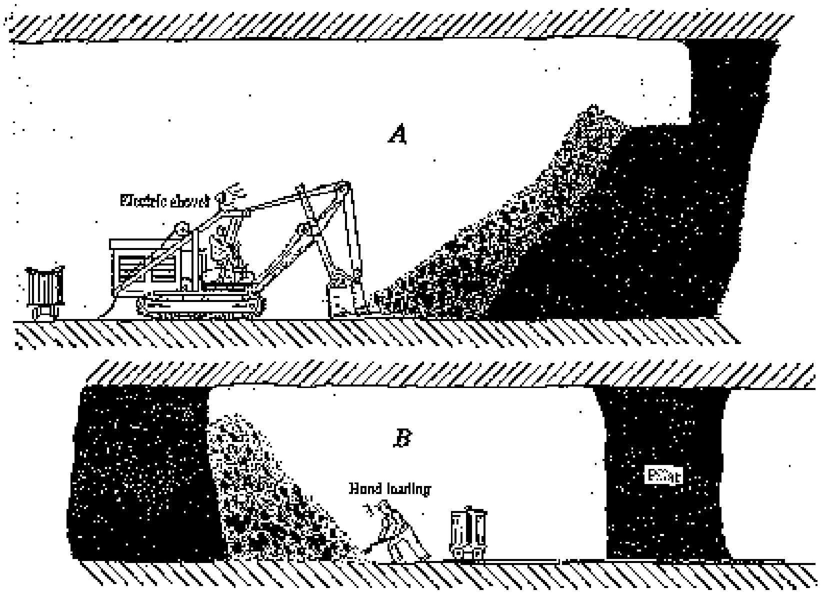


FIGURE 13.—Loading ore from heading-and-bench stopes in Tri-State district: *A*, Loading by electric shovel; *B*, hand loading.

in the same district. Note the "cans" into which the ore is loaded. These cans hold only 1,200 pounds of ore and are not large enough for the most economical use of power loaders.

Figure 14, *A*, shows a special type of electric shovel loading into cars in a stope in a southeast Missouri mine, and figure 14, *B*, a scraper loading from heading and bench into cars which are run under a loading slide. The latter installation is in a large tunnel, but the same method could be applied in stoping. The muck is dragged from the heading over the edge of the bench and then dragged from the bottom over the slide and into the cars.

Figure 15 shows a scraper dragging ore from a stope into a mill-hole at Mascot, Tenn. When the floor of the stope is at the elevation of the haulage drift the same equipment is used to drag the ore from the stope to a loading slide.

TRI-STATE DISTRICT

Nearly all the ore deposits of the Tri-State district are in the Boone formation of lower Mississippian age. This formation, believed to have been originally a limestone, has been dolomitized

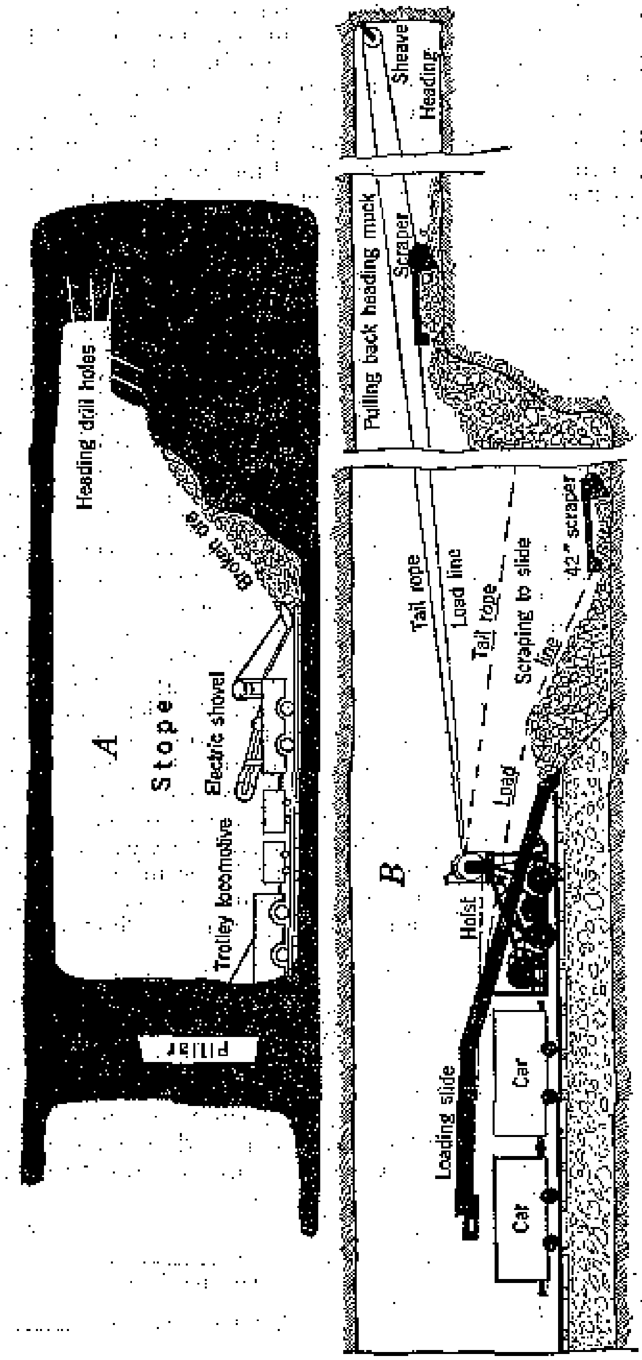


FIGURE 14.—Power loading: A, Special-type electric shovel loading into cars, southeast Missouri; B, scraper loading from heading and bench. (After St. Louis Power Shovel Co.)

and silicified and is now made up of flat-lying beds of limestone, dolomite, and chert, nodule beds, and one or more oolite beds. The exploited part of the Boone formation has been divided into 16 distinct beds, ranging from 4 to 55 feet in thickness, of which 6 have provided the principal ore-bearing horizons. All of the ore deposits are related to structural features and occur in zones of shearing, shattering, and brecciation and where there has been considerable silicification of the original limestone. Most of them lie within a horizon 200 feet thick, although a few high-grade but small deposits extend upward into the overlying Chester formation. They occur at shallow depths, the bottom of the ore seldom being over 350 feet below the surface and usually not over 300 feet. The top of the favorable formation usually is 95 to 175 feet below the surface. The ore beds range in thickness from 5 to 22 feet. Only part of the bed may contain ore. The lateral boundaries of the ore bodies are very irregular, and the line of demarcation between ore and waste may be quite sharp; in many instances, however, the transition from ore to waste is gradual. The height of the ore varies widely, and individual ore faces range from 6 or 8 to nearly 100 feet in height. Stopes 30 to 60 feet high are typical of the district. The ore minerals are sphalerite and galena, and the average grade of ore mined in the district over a period of several years has been around 4½ percent of metallic zinc, with a smaller percentage of lead.

The ore and walls usually stand well, except in rubbly or bouldery and badly fractured ground, or where the roof is shaly. Unsupported spans of 30 to 60 feet are common in the stopes. Pillars of ore 20 to 60 feet in diameter and 40 to 100 feet apart from center to center are left to support the roof in the wider deposits. From 15 to as much as 40 percent of the ore is left in pillars on first mining, depending upon the nature of the ground and the height of the ore. Ultimate extraction usually ranges from 80 to 90 percent. Many of the mines have been small, and mines of 400 to 500 tons capacity per day have been typical of the district. Recently a number of much larger mines have been developed. Some mines are equipped with cars, but loading into cans of 1,200 pounds capacity and hoisting the cans to the surface in small shafts has been standard practice for many years.

The method of stoping by the heading and stope-hole system has been described already. Figures 1, 3, 12, *A*, 12, *D*, and 13 illustrate practice in the Tri-State district. Most of the ore is loaded by hand and either is hand-trammed to the shaft or hand-trammed to a lay-by, whence it is hauled to the shaft by mules or locomotives.

The average stoping cost at seven typical mines in this district, taken from figures reported for different periods from 1927 to 1930, inclusive, was $0.504 per ton. The distribution of stoping costs is estimated as follows:

	Cost per ton
Labor, breaking	$0.144
Labor, loading	.129
Supervision	.020
Compressed air, drills, drill steel, and air lines	.102
Explosives	$0.104
Power (for power shovels)	.005
Total stoping cost	.504

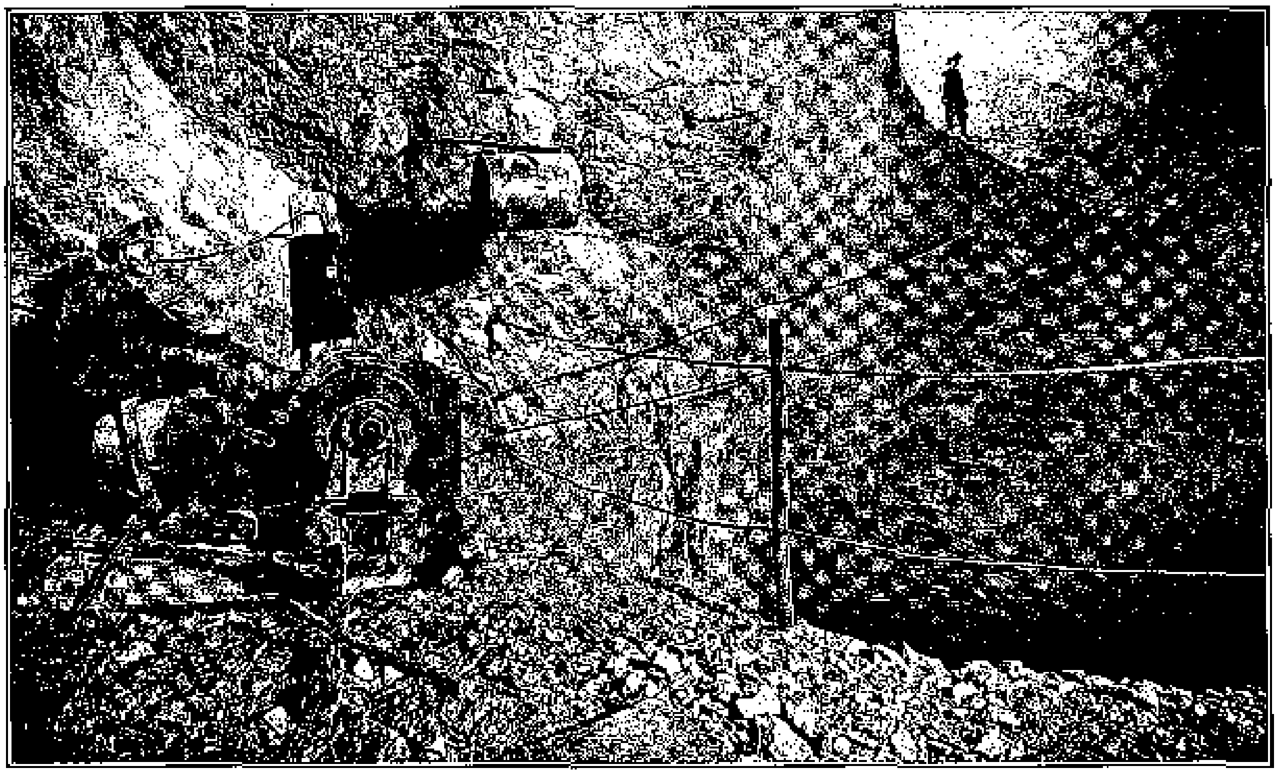


FIGURE 15.—Scraper dragging ore from a stope into millhole, Mascot, Tenn.

At five of these mines the daily output averaged 400 tons per day, at one it was 725 tons per day, and at another about 1,200 tons per day. The stoping cost ranged from 45 to 55 cents per ton. The basic figures from which these data were taken did not separate the development and stoping costs, and the development has been estimated and deducted from the total to reach the above figures. Hand tramming was included with loading in the original figures, and the author has deducted 25 percent of the reported loading costs arbitrarily for tramming. The costs given do not include tramming, haulage, hoisting, pumping, and general charges and are believed to approximate actual stoping costs at typical mines of the district from 1927 to 1930, inclusive.

Wage rates varied at different mines with the selling price of zinc concentrates and were as follows:

	Rate per 8-hour shift		
Machine runners	[9] $4.25	to	$4.75
Machine helpers	[10] 3.75	to	4.25
Trammers and mule drivers	3.50	to	4.00
Blacksmiths	4.25	to	6.00
Hoistmen	3.75	to	5.75
Locomotive operators	4.25	to	5.00
Locomotive brakemen	3.50	to	3.75
Powdermen	4.00	to	5.00
Roof trimmers	4.00	to	4.75
Trackmen	4.25	to	4.50
Shovel operators			5.00
Shovel operators on contract per can [11]			.095
Hand loaders on contract do [11]	.085	to	.14
Hand loaders on contract per 1½-ton car	.38	to	.415

[9] 1 mine, $3.50.
[10] 1 mine, $3.00.
[11] 1,200 pounds of ore.

The productivity of stope labor at the seven mines, measured in man-hours per ton, is shown in the following table. The figures include a little development work in some instances, but the amount is so small that it does not affect the stoping figures seriously.

Productivity of stope labor

Mine	Man-hour per ton of stoping					Tons per man-shift
	Breaking	Loading (by hand)	Hand tramming in stopes	Power-shovel loading and tramming from shovels	Total	
1	0.377	0.305	0.102		0.784	10.2
2	.297	.259	.086		.642	12.5
3	.339	.197	.065		.401	10.9
4	.286	.260	.090		.636	12.6
5	.223	[1] .241	[1] .070	[1] 0.163	.502	15.9
6	.218	.204	.068		.490	16.3
7	.229	[2] .163	[2] .055	[2] .160	.427	18.7
Average	.262	.234	.076		.564	14.4

[1] Average hand and power-shovel loading and tramming, 0.279.
[2] Average hand and power-shovel loading and tramming, 0.198.

Tonnages broken per machine-shift

Mine		Tons per machine-shift
1	In headings	40
	In stope faces	60–75
2	Average (headings and stopes)	60
3	Do	62.5
4	In headings	30–40
	In stope faces	80–100
	Average (headings and stopes)	62
5	Do	64.2
6	Do	76.7
7	Do	62.2

Consumption of explosives at the different mines was as follows (the figures include a small amount of powder consumed in development work):

Mine	Kind and grade of explosive	Pounds per ton broken
1	(?)	0.750
2	Ammonia, 33 percent	1.265
3	Gelatin dynamite, 20 percent	.875
4	Gelatin dynamite, 30 percent	.594
5	Ammonia, 40 percent	.805
6	Gelatin dynamite, 30 percent	.741
7	Ammonia, 40 percent	1.120

Further details of practices and costs in the Tri-State district are given in information circulars dealing with individual mines.[12]

SOUTHEAST MISSOURI DISTRICT

In the Southeast Missouri lead district the ore is mined in open stopes by the heading-and-bench system. The ores are found in sedimentary rocks of Middle Cambrian age, chiefly in the Bonneterre limestone, which consists of about 400 feet of magnesian limestone, some of which is argillaceous and interbedded with layers of dark-gray and black shale. The Bonneterre formation rests upon the porous La Motte sandstone and is capped by the Davis shale, which is about 160 to 200 feet thick. The ore-bearing beds are nearly horizontal, although they dip up to 20° locally. Galena is the principal mineral, but at a few places enough sphalerite occurs to make it profitable to produce both lead and zinc concentrates. At other places there is no zinc in the ore. The ore is found principally in the lower 100 or 150 feet of the Bonneterre limestone, although it occurs at various horizons from top to bottom of the formation. In some places the workable horizons are separated by barren or low-grade rock, and at others the deposits are workable over continuous

[12] Netzeband, Wm. F., Method and Cost of Mining Zinc and Lead at No. 1 Mine, Tri-State Zinc and Lead District, Picher, Okla.: Inf. Circ. 6118, Bureau of Mines, 1929, 11 pp.; Method and Cost of Mining Zinc and Lead at Mine No. 2, Tri-State District, Picher, Okla.: Inf. Circ. 6121, Bureau of Mines, 1929, 11 pp.; Method and Cost of Mining Zinc and Lead at No. 3 Mine, Tri-State District, Crestline, Kansas: Inf. Circ. 6174, Bureau of Mines, 1929, 10 pp.

Banks, Leon M., Mining Methods and Costs in the Waco District: Inf. Circ. 6150, Bureau of Mines, 1929, 10 pp.

Keener, Oliver W., Method and Cost of Mining at Buer Mine, Tri-State Zinc and Lead District: Inf. Circ. 6159, Bureau of Mines, 1929, 9 pp.; Methods and Costs of Mining at Hartley-Grantham Mine, Tri-State Zinc and Lead District: Inf. Circ. 6256, 1930, Bureau of Mines, 8 pp.

Anderson, Carl N., Mining Methods and Costs at the Interstate Zinc & Lead Co.'s Hartley Mine, Tri-State Zinc and Lead District: Inf. Circ. 6656, Bureau of Mines, 1932, 16 pp.

heights of 200 feet. The galena occurs disseminated through dolomitic limestone and shaly beds, in horizontal sheets along bedding planes, in vugs and cavities, filling or lining the walls of joints and crevices, and as aggregates of cubes in channels and joints. The ores average about $3\frac{1}{2}$ percent lead, although some are much richer.

The ore bodies are irregular in plan and elevation. Figure 16 shows a plan of a portion of the workings in one of the mines and

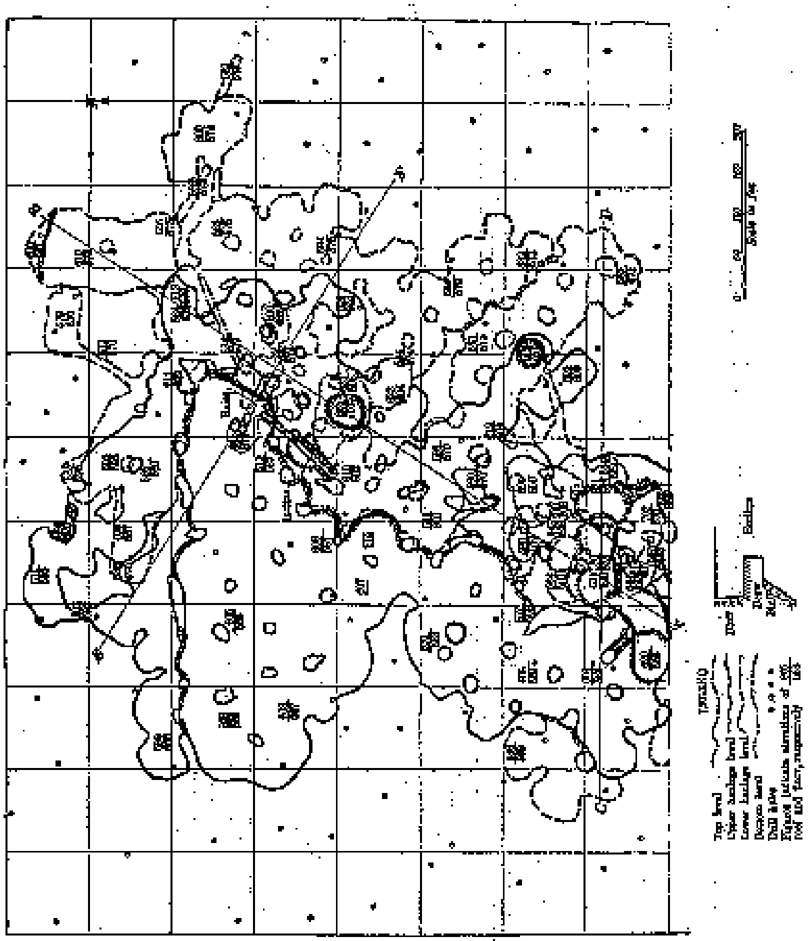


FIGURE 16.—Plan of portion of mine workings, Southeast Missouri lead district.

figure 17 cross sections of the same workings. Stoping widths range from a few feet along some of the upper, narrow fractures to several hundred feet. Some of the ore bodies are 700 or 800 feet wide and 1,200 feet long. In some parts of the district the ore bodies are almost continuous for several miles.

The formation varies considerably in strength in different localities and different sections of the same mine but on the whole is very strong, and the roof often will stand over spans of 100 feet between pillars, although the usual span is 40 to 60 feet. The walls are of

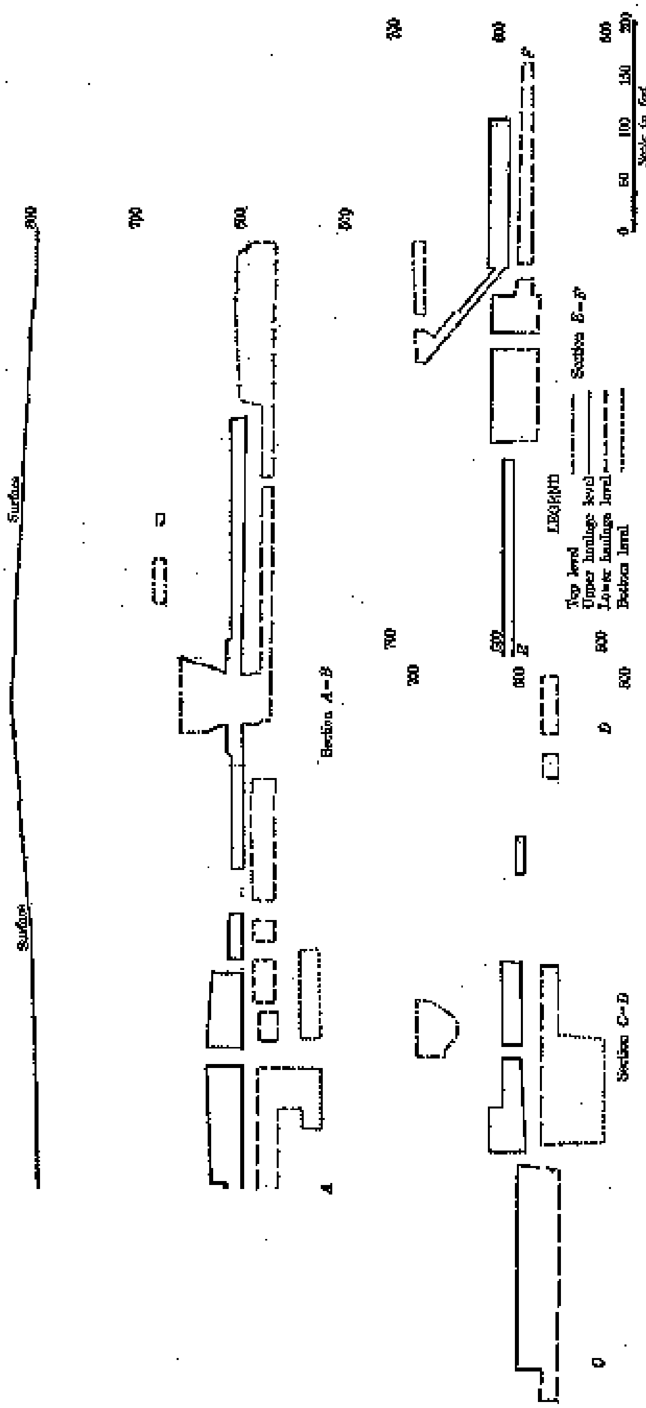


FIGURE 17.—Sections through mine, Southeast Missouri lead district.

the same character as the ore, and the ore-bearing areas pass laterally into barren rock, in some places abruptly and in others gradually. Irregular, roughly circular pillars are used to support the roof in wide ore bodies. Individual stope faces range from 7 to 200 feet or more in height. At present most of the production comes from depths of less than 750 feet.

The ore is stoped as previously described by driving a heading under the top of the ore and benching the stope face down to the level. (See fig. 12, *B*.) The stope is breasted out the full width between pillars, and when the face reaches a pillar area the pillars are "turned" as shown, branch stopes being driven to right and left between pillars.

At No. 8 mine of the St. Louis Smelting & Refining Co.,[18] the ground is strong but breaks well and is drilled by one-man, wet, jackhammer-type drills. The heading is carried 6 to 8 feet high directly under the top of the ore. At this mine the average height of the ore is about 20 feet, the maximum being 35 feet. Holes are drilled 4 to 18 feet deep, starting with a 1⅞-gage bit and decreasing one-eighth inch for each 3-foot change in steels. Stope holes carry 3 to 5 feet of burden and are blasted with 35-percent gelatin powder. Comparatively little blockholing is necessary, as the ore breaks well. An entire cycle—drilling, blasting, and mucking—is completed each working shift. Each tonnage contractor is allotted a stope and is responsible for the safety of the roof and pillars therein; no roof trimmers are employed, except under unusual conditions. The company furnishes all supplies and equipment necessary, and the driller loads and fires his own holes and is paid a stipulated price per ton of rock broken which varies with the height of the stope. Both hand shovelers and power-shovel operators work in pairs on a tonnage basis. At this mine the ore is loaded direct into either 1- or 2-ton cars and hauled to the shaft by trolley locomotives. A mechanical loader (fig. 14, *A*) is used for loading where the tonnage warrants its use. About 12 percent of the ore is left in the mine in the form of pillars.

In 1928, 168,089 tons of ore were mined at No. 8 mine from open stopes. Direct stoping costs (excluding tramming, hoisting, pumping, and general mine maintenance) for that year were as follows:

Direct stoping costs, No. 8 mine, 1928

	Breaking ore	Loading ore	Repairs and maintenance	Total
Labor	$0.1319	$0.2307	$0.0715	$0.4341
Supervision	.0152	.0152		.0304
Explosives	.0721			.0721
Power	.0092	.0009		.0101
Materials and supplies	.0207	.0119	.0086	.0412
Total	.2491	.2587	.0801	.5879

[18] Poston, Roy H., Method and Cost of Mining at No. 8 Mine, St. Louis Smelting & Refining Co., Southeast Missouri District: Inf. Circ. 6160, Bureau of Mines, 1929, 22 pp.

Base rates of pay for various occupations in 1928 were as follows:

Occupation:	Rate per 8-hour day	Occupation—Continued:	Rate per 8-hour day
Shift boss	$5.95	Hand shoveler (minimum, 18 tons)	$5.00
Assistant shift boss	5.35	General underground labor	5.00
Mechanical-shovel operator	5.80		
Company driller	5.05		

The rate of $5.80 for shovel operators applied only when mechanical loading conditions were very adverse. The operators usually received 8 cents per ton, and if their earnings exceeded $60 per week half the excess was returned to the company. Contract hand shovelers received $5 for loading 21 tons, $5.55 for loading 22 tons, and $0.28 for each additional ton over 22 tons. All breaking was done on contract. Each tonnage contractor was guaranteed the company rate of $5.05 per shift. For stopes 8 feet or less in height the price was 14 cents per ton broken, for stopes 8 to 20 feet high 13 cents, and for stopes more than 20 feet 10½ cents. These rates applied where the ore was loaded by hand; when the ore was loaded by mechanical shovel the rate was 1 cent less. Breaking and loading were paid for according to actual underground scale weights and not by car units.

The average man-hour per ton in breaking in stopes, including a little labor in drifting, during 1928 was 0.122, representing 65.5 tons per man-shift. The man-hour per ton in loading was 0.319, equivalent to 25 tons per man-shift. The total man-hour of stope labor per ton was 0.441 (18.1 tons per man-shift). The consumption of explosives averaged 0.565 pound per ton broken; three-quarters was a bulk dynamite (170 sticks per 50-pound box), and one-quarter was regular 35-percent gelatin dynamite.

The practice at another mine in the Southeast Missouri lead district has been described by Jackson.[14] Figures 16 and 17 show parts of the mine workings. The thickness of ore ranges from 7 to over 200 feet, but in 1929 no active stopes were over 35 to 40 feet high. In a given area the ore may occur at several overlapping horizons (fig. 17) or be confined to 1 or 2. In some places pillars are over 100 feet apart, while in other places the character of the ground makes it necessary to limit spans to 17 or 18 feet; the average spacing is 30 feet in the clear. Shaly layers tend to slack upon exposure; and vertical channels, slips, and joints constitute other sources of weakness in both the ore and roof of stopes. Where the ore occurs at more than one horizon and where stopes at various levels come under or over those above or below the pillars are matched to come under or over the pillars above or below.

The rock drills easily and is quite uniform in texture, and 80 to 100 feet of hole is a fair average footage per machine shift. Most of the drilling is done with light hammer drills. For drilling flat or upward-pointed holes the drills are equipped with pneumatic feed legs, and for stope and other down-holes and for drilling lifters they are hand-held. Where the ore is not over 9 or 10 feet thick it is breasted out the full height. In thicker ore a heading 7 or 8 feet high is breasted out immediately under the top of the ore in

[14] Jackson, Chas. F., Methods of Mining Disseminated Lead Ore at a Mine in the Southeast Missouri District: Inf. Circ. 6170, Bureau of Mines, 1929, 21 pp.

advance of the stope. (See fig. 18.) In the heading three holes are drilled above each other, but the middle or breast hole is drilled with about 6 inches less burden than the back hole and lifter. The burden usually is 3 to 4 feet. These holes are drilled 8 to 10 feet deep, occasionally deeper. In turning a pillar the holes are never drilled toward the pillar but are drilled tangentially, as shown at *X–X* (fig. 18) to avoid shattering it. After a stope is fully opened usually 2 and sometimes 3 headings are in operation. One driller works in each heading and the stope or "bluff" behind it. Stope holes are drilled 3 to 4 feet back from the face and spaced 6 to 8 feet apart. Those drilled from the floor of the heading are 6 and those in the lower benches 10 feet deep. In a heading or low stope

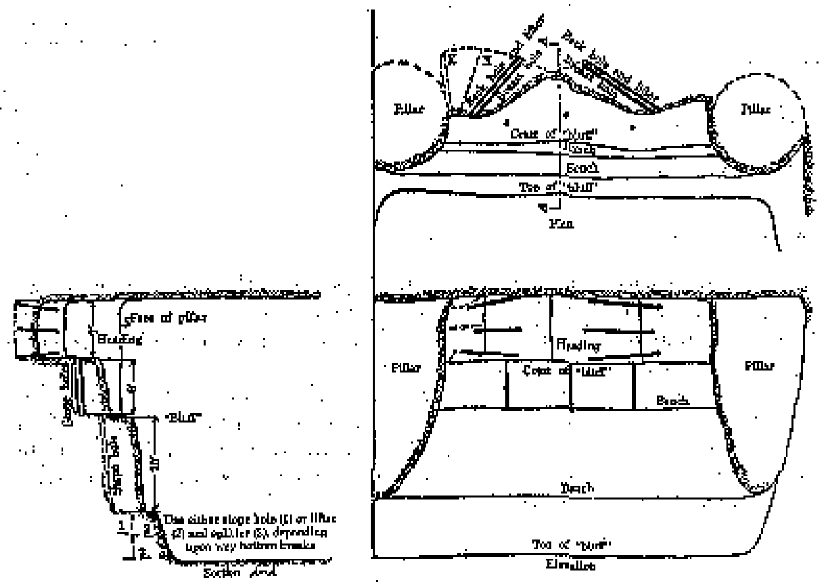


FIGURE 18.—Heading and stope or "bluff", southeast Missouri.

a driller usually will break about 30 tons per shift and more in higher stopes. In "bluff" mining 90 to 120 tons per shift are common. An average shift's work consists of about 12 holes, or 80 to 100 feet of drilling. The drillers work on contract and are paid a stipulated price per ton broken, as measured by the number of cars of 2½ tons capacity loaded out. In low stopes and hard ground the price in 1929 was 16 to 18 cents per ton, in stopes of medium height 12 cents, and in very high stopes as low as 8 cents.

In development work and stopes where not enough ore can be broken to keep a mechanical loader busy the ore is hand-loaded. The usual task or "score" for a hand loader is 20 tons. This score is reduced when loading is difficult or when there is a long tram to the "loop" or siding. Mechanical loaders are of the shovel or dipper type, are mounted on caterpillar traction, and have a full 360° swing. The shovel operator transfers his own cars to the loop and is paid for loading under a bonus system whereby he receives a regular wage for a given task or score and a bonus for tonnage in excess of this. The usual score is 42 cars of $2\frac{1}{2}$ tons each, but this is varied to suit conditions and ranges from 32 to 48 cars.

This mine is one of the oldest in the district, and much of the ore comes from extensions of old stopes and areas which were too low grade to work in earlier days. Current workings must be planned to fit the old workings and new pillars to "match" earlier ones, and the work is scattered. Labor performance in stopes, as measured in man-hours per ton during an average month in 1929, was as follows:

Labor performance in stopes

Occupation	Man-hour per ton	Tons per man-shift
Breaking	0.1625	49.2
Loading (hand)[1]	.4160	19.2
Loading (mechanical)[1]	.0240	331.1
Loading (hand and mechanical)[1]	.2330	34.3
Roofmen (trimmers)	.0340	235.2
Total	.4295	18.6

[1] Includes mule haulage by loaders between face and loops.

The consumption of explosives in stopes for the same period was 0.459 pound of 40-percent ammonia gelatin per ton of ore broken. Mechanical loading required 0.2 kw.-hr. per ton of ore and air compression 3.7 kw.-hr. per ton.

A unique feature of the practice in this mine and other mines of the district is the use of scaffolds suspended from the roof from which the back may be inspected and trimmed and test holes drilled in the roof. Details of this feature are given in Bureau of Mines Information Circular 6170.

MASCOT, TENN.

At Mascot, Tenn., zinc ore averaging 2.9 percent zinc (1929) is mined by a millhole variation of the heading-and-bench system. The ore body has an average dip of about 20°, is wide and long, and ranges from a few feet to 150 feet in thickness. It occurs in a dolomitic limestone. The ore mineral is sphalerite of exceptional purity, which occurs as seams and veinlets with secondary dolomite, and cementing particles of brecciated limestone. The mineralized ground ordinarily will not stand long over wide spans without support, whereas the unmineralized rock comprising the roof and walls of the stopes often will stand almost indefinitely over spans of 100 feet or more. Locally, occasional thin shale partings between beds so weaken the ground as to cause it to fall, but such partings are not common. When they occur near the top of the ore the insecure ground is usually taken down, and some strong overlying bed is carried as a roof. The stoping practice has been described by Coy.[16]

The mine is developed by a vertical shaft 612 feet deep, and stopes are being mined both above and below the shaft level, which is at

[16] Coy, Harley A., Mining Methods and Costs, American Zinc Co. of Tennessee, Mascot, Tenn.: Inf. Circ. 6230, Bureau of Mines, 1930, 11 pp.

520 feet. The ore below the shaft level is developed by slopes. In the ordinary heading-and-bench stoping system individual stope units are advanced in one direction. In the millhole variation of this system a raise is put up to the top of the ore, a heading 7 feet high is advanced immediately under the bed selected for the roof of the stope, and this heading spreads out in all directions from the raise. The ore then is benched down by underhand stoping around the raise, forming a funnel-shaped stope. The ore runs by gravity to the bottom of the raise, whence it is drawn off through chutes into cars. The headings are fanned out until they reach pillar lines, and the benching continues until the slope of the "funnel" or millhole sides is flatter than the angle of repose of the broken ore (about 45°). The stope continues to widen until it meets the next stope between the pillars, and the broken ore is dragged to the millholes by power scrapers. As much as 100,000 tons is sometimes mined to a single millhole. Pillars are roughly circular in cross section and proportioned to the character of the ground and height of the ore.

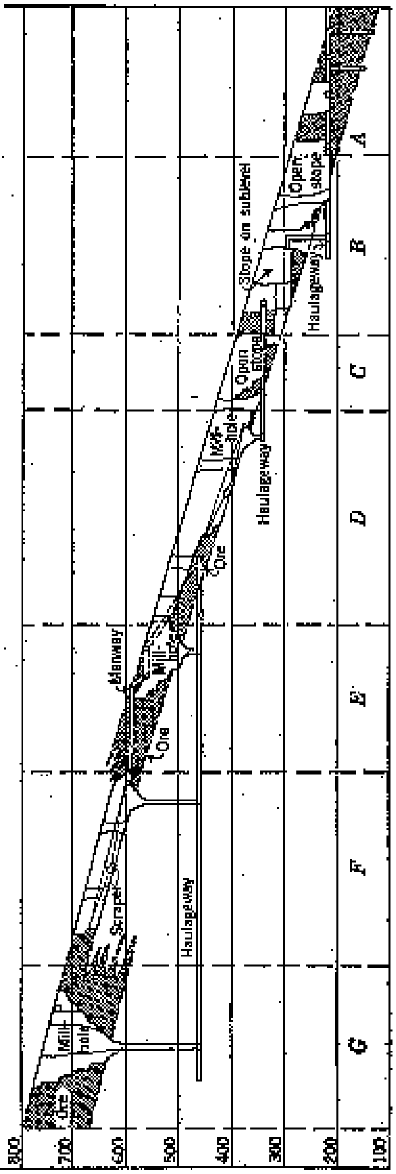


Figure 19.—Steps in development of mining processes, Mascot, Tenn.

Ordinarily they are 25 to 30 feet in diameter, and the stopes between are 40 to 60 feet wide. Figure 19 shows the development of the mining processes diagrammatically. *A* shows prospecting at low elevations by crosscuts and diamond drilling; *B*, sublevel operations employed in earlier days of the mine for cleaning up such ore as remained between haulageways after the toe of an open stope had reached the footwall; *C*, a stope opened up from a crosscut passing through the ore body; *D*, slushing ore to a millhole after the millhole sides have reached the angle of repose of broken ore; *E*, millhole operations before slushing becomes necessary; *F*, slushing in thin ore considerably above the haulage level; and *G*, millholing a thick ore body considerably above the haulage level. The method employed at this mine is particularly adapted to a large, rather thick deposit in which the ore itself does not necessarily stand well unsupported but which has a very firm and strong roof.

Figure 12, *C*, illustrates the method of driving headings and cutting the benches at Mascot. Heading holes are 6 to 10 feet deep and bench holes seldom over 10 feet. Thirty-percent gelatin dynamite is used in shooting all holes.

No reserve of broken ore is stored in the stopes or millholes, and any ore that may be left in them at the end of the day shift is drawn off on the following night shift. The working faces must be uncovered each day for inspection, to maintain the grade of the ore broken.

In 1929 considerable hand loading still was done in stopes opened out from the levels, but today (1933) very little hand loading is done in the stopes, and this is confined to small tonnages in clean-up work. Virtually all of the ore is loaded from chutes or by scrapers, and scrapers are used in the stopes to drag the ore to millholes, when it will not run by gravity. Scraper hoists are driven electrically by 25-hp. direct-current motors, and arc-shaped manganese-steel scrapers with teeth, 6 feet wide, 21 inches high, and weighing 1,760 pounds, are employed. Improvements in scraping practice have made it possible to scrape ore from 2 or 3 stopes to a single millhole, thus reducing the number of long raises through rock to reach the bottoms of the stopes. Ore is scraped in one place a distance of 450 feet, and the average scraping distance is about 175 feet.

Ore is broken and loaded by contract. The stope contractor breaks and delivers the ore into cars at a stipulated price per ton; the company furnishes drills, steel, and power, and the contractor furnishes the labor and pays for the explosives. Slushing contracts cover breaking, dragging, and chute pulling at a stipulated price per ton.

From December 27, 1928 to October 30, 1929, this mine produced 528,626 tons of ore; 16.87 percent of this was loaded by hand on contract, 3.01 percent by hand on company account, 75.21 percent from millholes on contract, and 1.64 percent from millholes on company account. Stoping costs for this period are as follows:

Stoping costs, Mascot, Tenn., 1929

Labor	\$0.130
Compressed air, drills, and steel	.042
Explosives	.045
Other supplies	.005
Total	.222

Labor performance, as measured in man-hours per ton, was as follows:

Labor performance in stopes, Mascot, Tenn., 1929

Occupation	Man-hour per ton	Tons per 9-hour man-shift
Drilling and blasting	0.214	42.1
Timbering	.009	900.0
Shoveling and scraping	.093	96.7
Total	1 .316	28.5

[1] Chute pulling not included.

MARQUETTE RANGE, MICHIGAN

At a hard-ore mine on the Marquette range in Michigan[16] the ore is mined in open stopes separated by regular pillars, forming a checkerboard stope-and-pillar pattern. Here the development is not as simple as in the examples just given.

The ore occurs in an irregular, badly folded, and faulted bed of iron formation, ranging in thickness from a few to 100 feet, with an average thickness of 25 to 30 feet in the larger ore bodies (fig. 20, *A* and *B*). It is overlain by quartzite and slate, and the footwall is jasper and altered diorite. The dip of the ore is variable but in general is too low for the ore to run by gravity on the footwall.

The capping over the ore body must be permanently supported because the formation is overlain by 50 to 100 feet of quicksand and because the ore extends long distances under the business section of a town.

The ore is a hard, high-grade, specular hematite which is tough and hard to drill. It stands well and does not slack on exposure to the air. Seams of rock 1 or 2 inches to 1 or 2 feet thick often occur in the ore and must be sorted out by hand underground. The hanging wall, which in most places is a slate, slacks when exposed to the air; therefore, a small amount of ore is left to protect it whenever possible. Overlying the slate, which seldom is over 15 feet thick and usually much less, is a strong, hard quartzite which forms an excellent capping over the stopes. Displacement of the ore bodies by faulting and their irregular size have made it impossible to block out large ore bodies for mining. Flexibility in stoping method is required to permit following the ore.

Where the thickness of the ore will permit, rooms 25 feet wide by 25 feet high are driven by breast or heading-and-bench stoping, and floors 25 feet thick are left between levels, except on the upper levels where they are only 15 feet thick. Pillars between the stopes are 25 feet square. About 72 percent of the ore is extracted from the ore bodies; the balance is tied up in pillars.

The rooms are advanced by carrying a heading cut, 7 or 8 feet high and as wide as the stope, 10 feet ahead of the bench (fig. 20, *C*), which is taken up afterward by lifting holes drilled horizontally. The heading cut is driven by the use of slabbing holes, and wherever possible advantage is taken of slips. When the heading cut is far enough ahead the face is drilled again but not blasted, and the bench is cleaned off. Lifting holes 8 to 10 feet deep are drilled 5 or 6 feet apart in horizontal rows with a burden of about 5 feet. The whole breast then is blasted at one time. Since the bottom holes do not extend in as far as those above a toe is left as a start for a new bench, and the broken ore lies on this toe in a pile high enough to allow the miners to reach the back conveniently. The back and breast then are trimmed, and drilling for the next cut is begun. This system has many advantages; it facilitates trimming the back, always leaves a bench or pile of broken ore for the miners to stand on, and provides a fairly continuous supply of broken ore.

[16] Eaton, Lucien, Method and Cost of Mining Hard Specular Hematite on the Marquette Range, Michigan: Inf. Circ. 6138, Bureau of Mines, 1929, 14 pp.

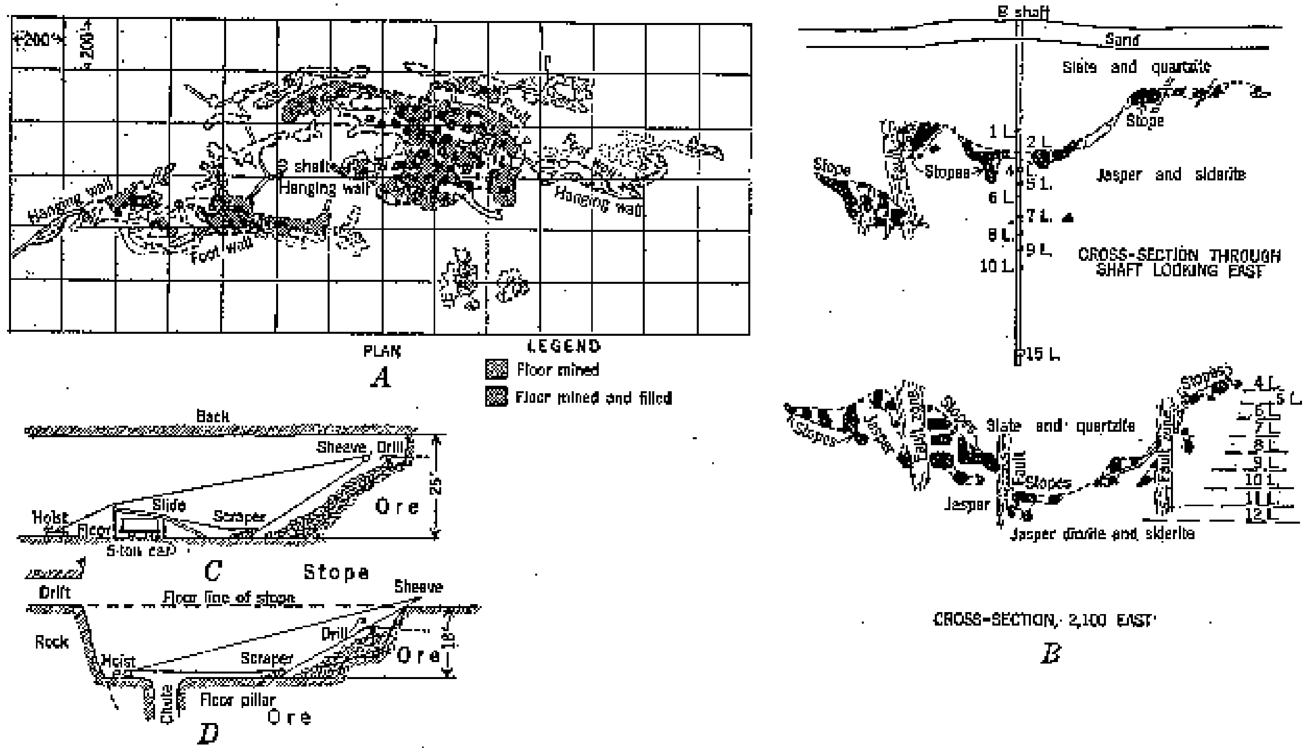


Figure 20.—Open stoping at hard-ore mine, Marquette range, Michigan: A, Plan of ore bodies on second level; B, north-south cross sections; C, heading-and-bench or breast stoping, scraping to car; D, removing floor pillar, scraping to chute.

When the ore does not extend up to the next level the stope must be finished by back stoping. A stage is built of ladders and 3-inch planks, and from it as many holes as possible are drilled in the back. The holes are charged, the stage is torn down, and the holes are then fired. The stage again is erected on top of the broken ore, and the operation is repeated until the pile of broken ore is high enough to reach the back without staging. The stope is continued to the top of the ore and then is advanced longitudinally over the stope below, breaking the ore down by horizontal holes drilled in rows; these holes often carry a 6-foot burden. Shovelers begin at the end of the pile of broken ore and load it out, following up the mining ahead of them.

In mining floor pillars, practice in 1929 was to work on a sublevel instead of mining the entire 25 feet and dropping the ore to the level below. (See fig. 20, *D*.) A raise was put up, a chute installed, and the ore around the raise milled into it to a depth of 18 feet. When the ore no longer would run into the chute by gravity it was shoveled by hand at first, and after the stope had been widened out it was dragged by scrapers. As the footwall is flat the ore would have to be rehandled several times if it were shoveled by hand. Raises usually were about 100 feet apart, and the scraper could reach 50 feet in either direction from the chute easily. When all the ore had been mined down to the floor of the sublevel the remaining 7 feet were mined in one lift, breaking the ore down to the level below.

About 5 or 6 hours of drilling was done each shift, and the machines averaged about 25 to 30 feet per shift. In full-size stopes about 1 ton of ore was broken per foot of hole; in breast holes the break was less and in bench holes more. Usually 1 miner worked in each stope, but occasionally there were 2 miners using 2 machines. They were paid by contract according to the number of 2.5-ton cars trammed from the stope; the size of the stope and the hardness and toughness of the ore were taken into account in setting the price. Contractors were charged for all explosives used and for carbide, picks, shovels, and other hand tools. The ore broke in large pieces, and considerable sledging and blockholing were required.

Most of the ore was loaded into cars or dragged to chutes by scrapers, but where there was much rock to sort out and where the life of the stopes was short, hand loading was preferable. When cars were loaded by hand on contract the shovelers were paid a certain price for filling a 2½-ton car and 1 cent additional for every 45 feet hand-trammed.

Contract prices in 1928 were as follows:

Contract prices per car of 2½ tons (hand loading and tramming)

	Large piles	Stopes	Drifts	Chutes
Hand loading	$0.47	$0.52	$0.57	
Chute loading				$0.20
Tramming, per 45 feet	.01	.01	.01	.01

On sublevels and where a chute was within reach the broken ore was moved from the breast to the chute by scrapers. (See fig. 20, *D*.) These scrapers were of hoe type, were 48 inches wide and 7 feet long, and weighed 1,500 pounds. Scrapers were handled by 25-hp. double-drum electric hoists. One miner and one scraperman constituted the usual stope crew; the scraperman helped the miner to set up and tear down, and the miner helped the scraperman rig up his scraper blocks. Contract prices per car of 2½ tons for both breaking and scraping ranged (1928–29) from $0.90 to $1.50. In a few places scrapers loaded into 5½-ton cars over semiportable loading slides. (See fig. 20, *C*.) Four cars (22 tons) were loaded per hour, including switching and other delays.

During 1928, when 420,000 long tons (2,240 pounds) were mined and hoisted, stoping costs were as follows:

Stoping costs per long ton hoisted

	Per ton		Per ton
Labor	$0.514	Explosives	$0.105
Supervision	.028	Timber	.002
Compressed air, drills, and steel	.104	Other supplies	.011
Power	.006	Total stoping cost	.770

Labor performance in stopes, as measured in man-hours per ton, was as follows:

Occupation	Man-hour per ton	Tons per 8-hour man-shift
Breaking	0.342	23.4
Timbering and filling	.053	150.8
Shoveling (including scraping)	.310	25.8
Total	.705	11.4

Explosives used in stoping amounted to 0.705 pound of 50 percent ammonia dynamite per ton broken.

MINEVILLE DISTRICT, NEW YORK

At Mineville, N. Y., the ore is mined in open stopes with casual or irregular pillar support. Here the ore bodies are different from any described heretofore in that, although some thick ore has been mined (over 200 feet in one area), the ore mined in recent years has been 3 to 40 feet thick and averages only 10 feet, and the ore beds dip at 20° to 30°.

The ore is magnetite, running from 25 to as high as 69 percent iron, but it averages only 42 percent iron as mined. It occurs in beds interstratified with beds of gneiss. The productive beds extend several hundred to several thousand feet in depth on the dip, with correspondingly large lateral dimensions. Figure 21 is a plan of the Old Bed mine. Trap dikes cut the ore bodies, and they are markedly folded and faulted. In places there is no sharp line of demarcation between ore and rock, and magnetite is found disseminated through gneissoid beds in scattered grains, in sufficient quantity to be locally of commercial grade. In other places, especially along the footwall, the line between ore and rock is sharp.

The mine is developed by long inclined shafts and auxiliary inclines along or in the footwall from which stoping levels are turned off following the ore contours at 30- to 100-foot intervals. Stopes are

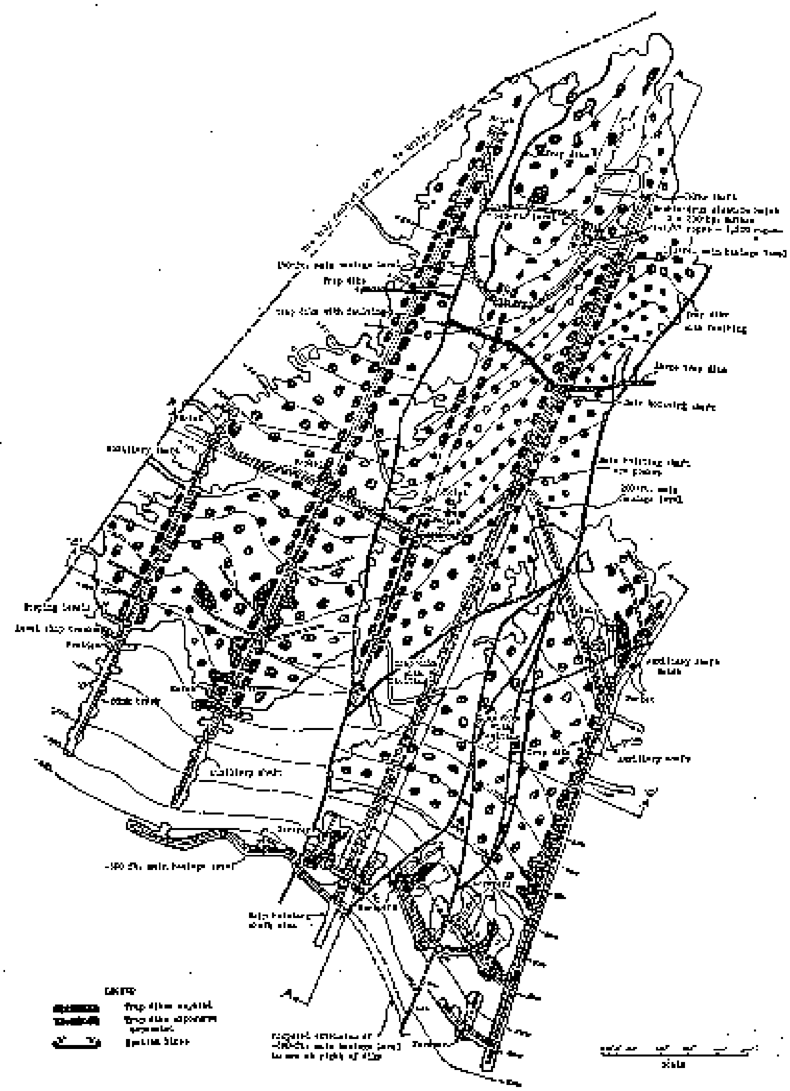


FIGURE 21.—Plan of Old Bed mine, Mineville, N. Y.

breasted out the full thickness of the ore, leaving pillars 20 to 60 feet in diameter, depending upon the height and character of the roof. These pillars are spaced about 50 feet apart center to center. The roof is arched between pillars to prevent slabbing off of ore and rock. Pillars contain about 25 percent of the ore in the mined areas, but later it will be possible to recover a considerable proportion of this ore.

Most of the ore is loaded mechanically or is scraped into loading chutes. Scrapers and power shovels are employed. The former are electrically operated, and the latter are operated by compressed air. The ore breaks in large chunks and slabs, and the scrapers are 48 inches wide and 26½ inches high, have a long bail (7 feet 6 inches), and weigh 1,400 pounds. Six full loads will fill a 7-ton car, although 8 to 10 loads is the average. Scraper hoists are

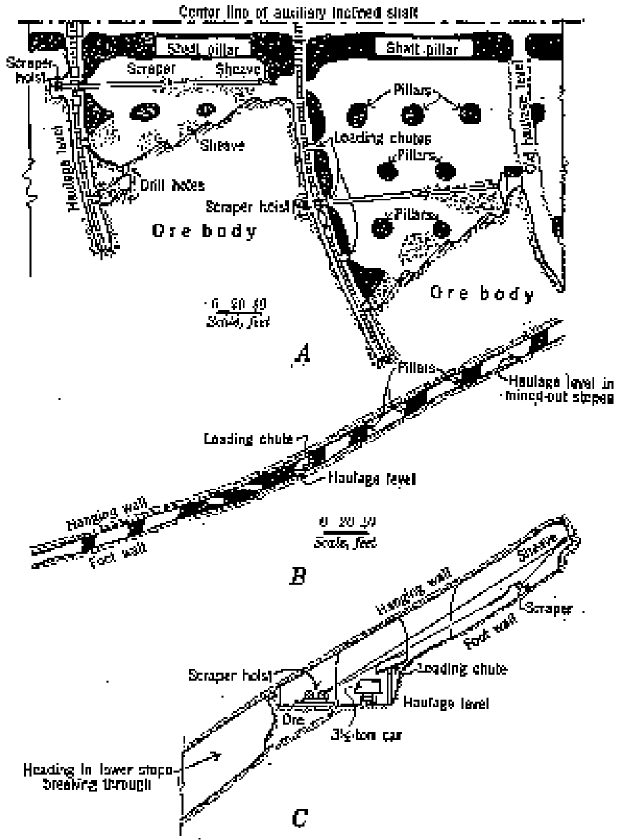


FIGURE 22.—Scraper stopes, Mineville, N. Y.: *A*, Plan of a stope; *B*, cross section through stope; *C*, enlarged cross section showing scraping to chute.

powered by 25-hp. direct-current motors. Maximum scraping distance is 330 feet, the average being 180 feet. Figure 22 shows a plan and sections of scraper stoping operations, the method of drilling the faces, and scraping into cars and chutes.

The rock is hard and breaks large. Miners in the stopes break an average of 30 tons per man. In 1927, 14 cents per ton was paid each driller, but if the average tonnage fell below 30 tons the basic wage of 36 cents per hour ($2.88 per shift) was paid. The average wage for all men at the Harmony and Old Bed mines in 1927 was $5.63 per shift.

Where ore was loaded and trammed by hand the muckers were paid 28 cents per car (1-ton average filling) for 10 cars, 29 cents per car for 11 cars, and 30 cents for 12 or more cars.

Direct stoping costs per ton for 1927, as given by Cummings[17], were as follows:

Direct stoping costs, Old Bed and Harmony mines, 1927

	Cost per long ton
Labor	$0.164
Supervision	.014
Compressed air, drills, and steel	.113
Explosives	.122
Other supplies	.003
Total direct stoping cost	.416

Labor performance, as measured in man-hour per ton, was as follows:

Occupation	Man-hour per long ton	Long tons per man-shift
Breaking (drilling and blasting)	0.236	33.9
Shoveling (hand and power loading)[1]	.480	16.7
Total	.716	11.2

[1] Includes some hand tramming.

EDWARDS MINE, NEW YORK[18]

At the Edwards mine zinc ores occur in crystalline dolomites interbedded with quartzites, schists, and gneisses. The ore bodies are lenticular masses 5 to 25 feet thick and 100 to 200 feet long, and persist down the dip to a vertical depth of 1,700 feet or more. The avarage dip is 40° to 45° but locally ranges from nearly horizontal to nearly vertical. The outlines of the ore bodies are quite smooth and regular. The walls are generally "commercial" ones, although the gradation from massive ore to virtually barren material is sometimes quite abrupt. The average grade of the ore is about 17 percent zinc but in places reaches 35 percent or more. Some dilution occurs in mining due to the carrying of minimum stope widths, which often make it necessary to mine some wall rock so that the average grade of ore hoisted is about 12½ percent zinc. Both ore and wall rocks stand well for a long time, except where parallel slips occur and where the walls contain considerable serpentine which spalls in irregular slabs. The tendency of walls to fail from these causes increases as the mine becomes deeper.

Figure 23 shows the method of stoping where the dip is flat enough to require the use of scrapers to move the ore from the face to the chutes. Open stopes are advanced by both overhand and underhand methods, depending upon the distance above the level, the homo-

[17] Cummings, A. M., Method and Cost of Mining Magnetite in the Mineville District, New York: Inf. Circ. 6092, Bureau of Mines, 1928, 12 pp.

[18] Knebel, John B., Mining Practice at the Edwards Mine of the St. Joseph Lead Co., St. Lawrence County, N.Y.: Inf. Circ. 6586, Bureau of Mines, 1932, 25 pp.

geneity of the ore, its thickness and dip, and the condition of the walls. Often a stope is worked by a combination of overhand and underhand stoping. In overhand work chute raises are driven on the footwall at intervals of 30 to 40 feet. These are driven up the dip 15 to 30 feet, a subdrift is turned off 10 or 15 feet above the back of the level to connect the raises, and the raises are belled out. Stopes are then breasted out above the subdrift level. The stopes are breasted out at least 6 feet high along the hanging wall, and if the ore is thicker the bottom is usually benched up from below afterward. The system is thus virtually an inclined heading-and-bench system. Pillars are left as required for roof support, and often low-grade or barren ground can be utilized for pillars. Ordinarily spans of 40 feet or more will stand during the life of a stope, and the pillars need not be very large. In some places

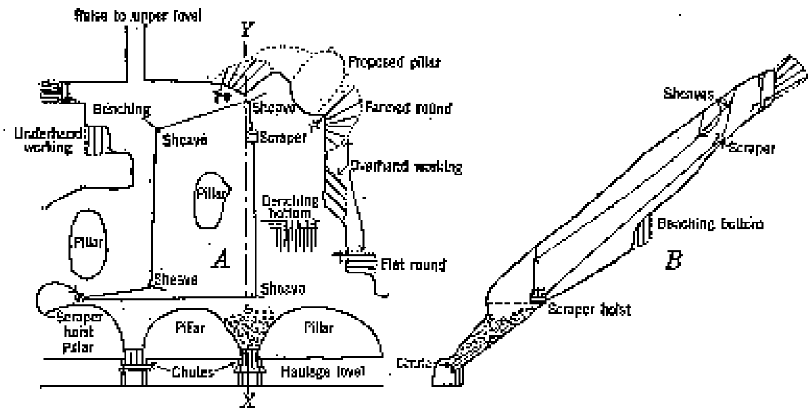


FIGURE 23. Open stoping with scrapers, Edwards, N. Y.: A, Vertical longitudinal projection; B, vertical section X–Y.

in the lower levels strong jointing parallel to the ore body is combined with a system of intersecting fractures which cut it obliquely, producing a blocky and dangerous hanging wall. In such places regular breasts 15 feet wide are driven up the dip between regular pillars 40 feet wide. These pillars are recovered later by cut-throughs and slabbing on the retreat.

Drilling is done principally with mounted one-man machines, although stopers sometimes are employed, especially for cutting around pillars in steep stopes. A maximum of nine 8-foot holes is drilled per heading or breast in stopes. For taking up the benches of ore vertical holes are drilled (at an angle of about 45° with the footwall), and the ore is broken into the stope, beginning at the bottom of the stope and advancing up the dip.

Underhand working is employed principally to mine the upper portions of blocks which have been worked overhand from below. It is also used in combination with overhand working, and a stope may be worked at various stages by several alternations between the two systems. Underhand work is cheaper, permits better selection, and results in easier breaking, but it is not so safe where the ground is weak, and greater difficulty is experienced in cutting around pillars. Before an underhand stope is begun a raise must be driven to the level above or to the top of the ore. The ore is then benched down around the raise in benches 7 feet or more high. Benches are usually about 6 feet wide and are drilled with three rows of holes, each carrying a 2-foot burden. Underhand stoping is applicable to steep ore bodies where the ore and walls are strong.

The angle of repose of the broken ore is 42° to 45°. Where the dip is less scrapers are employed to move the ore from the face to the chutes. (See fig. 23.) Scrapers are used for hauls up to 300 feet long. The use of scrapers in flat stopes has greatly increased the output per man and resulted in appreciable savings in mining costs.

The productivity of labor in stopes in 1930 was as follows:

Productivity of labor, Edwards mine, 1930

Occupation	Man-hour per ton	Tons per 8-hour man-shift
Breaking	0.456	17.5
Timbering	.072	111.2
Hand shoveling	[1].208	[1]38.5
Scraping	[1].090	[1]88.2
Total	.832	9.6

[1] Based on total tonnage of ore from stopes.

MICHIGAN COPPER DISTRICT

In the Michigan copper district open stopes with pillar support are employed where the dip is flat, whereas in the steeper lodes shrinkage stoping or cut-and-fill stoping is used. In the Conglomerate lode stulls are employed to support the hanging wall, and the method usually is classed as open stoping with stull support. In this bulletin the author has classified stopes with a regular system of stull support as a distinct stoping method under supported stopes, and the practice will be described later under "Supported stopes." Two types of pillar-supported stopes are employed in the Michigan copper mines and have been described by Crane [19]: (1) Large open stopes with irregular pillar support and (2) long, narrow open stopes supported by narrow pillars.

In both types a retreat system is employed; that is, the levels are developed to the property lines or ends of the ore, and stoping retreats from the ends toward the shaft. Chutes are installed at regular intervals and are driven up a short distance where they are connected by a subdrift or small stope. In the first type of stope the ore is breasted out on each side of the raise until adjacent stopes meet and merge into a single large stope which is advanced up the dip, leaving irregular pillars of ore or lean rock as required for local support of the hanging wall. (See fig. 24, *A*.) In the second type of stope adjoining stopes do not merge but are separated by a long, thin pillar. (See fig. 24, *B*.) In both types of stope an arch pillar is left over the back of the level.

[19] Crane, W. R., Mining Methods and Practice in the Michigan Copper Mines: Bull. 306, Bureau of Mines, 1929, 182 pp.

The lodes vary in thickness and where open stopes with pillar support are employed dip at angles of about 30° to 40°. The ore is native copper occurring in basic amygdaloidal beds, which lie between distinct hanging- and foot-wall trap rocks; individual beds vary in thickness, averaging 20 to 30 feet. The mineralized portion of an amygdaloid bed often is very irregular, sometimes being on one wall, sometimes on the other, and sometimes in the middle of the bed, and only occasionally is a bed mineralized enough to

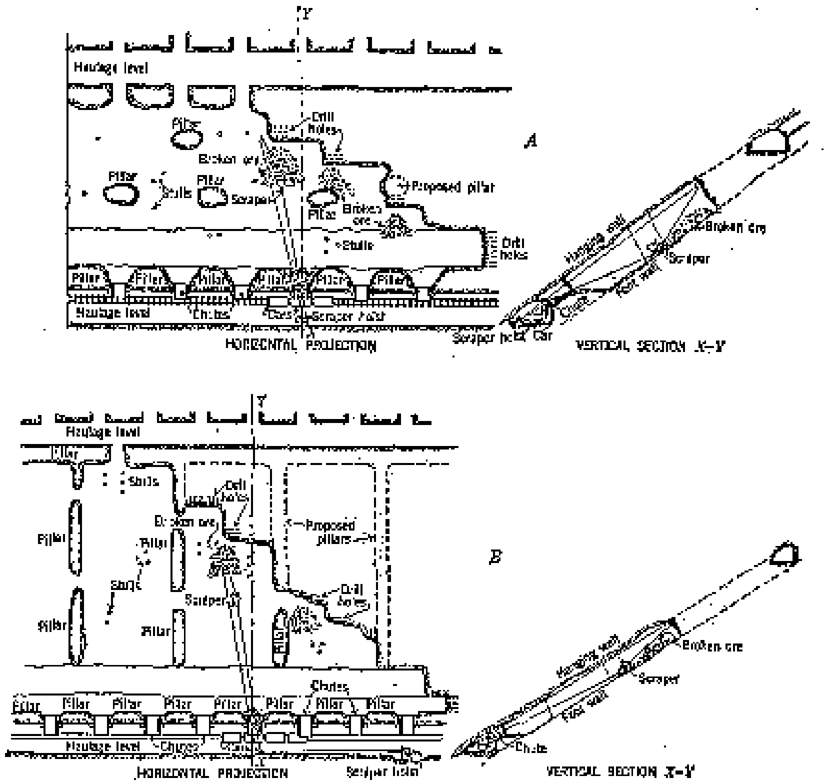


FIGURE 24.—Open stoping, Michigan copper mines: *A*, Large open stope with irregular pillars; *B*, long, narrow stopes separated by narrow pillars.

make ore from wall to wall. Usually there is no sharp line of demarcation between ore and rock. Workable lodes in the district range from 3½ to 50 feet in width. Probably the average width of lode worked today is 8 feet, although local enrichments may increase the width to 15 feet or more. The amygdaloid is not very hard to drill and breaks well, but bunches of native copper often make drilling difficult.

The ground in the open-stope mines stands well over considerable spans for some time except where badly shattered by faulting. The ore is low grade.

At the Osceola mine the lode dips 37° at the lowest levels, and the best ore is just below the hanging wall. Levels are driven 120 feet apart measured on the dip and are 12 to 16 feet wide along the hanging wall. Starting at the end farthest from the shaft, box holes are opened over the back of the drift on 25-foot centers; these are connected 10 or 12 feet above the level, leaving a pillar of this thickness over the back of the drift. Wide stopes then are breasted out along the hanging wall, leaving irregular pillars for support as required. Local bulges, rolls, or troughs in the ore are 25 to 30 feet thick in places. Where the ore is thick it is benched down from the hanging-wall heading or breast. Considerable lean ore is left as irregular pillars. In 1931, when the mine last was visited by the senior author, 26 tons per machine-shift (one-man machines) was the average break in the stopes. The broken ore was dragged down the flat, irregular footwall by 48-inch hoe-type scrapers handled by 25-hp. electric hoists.

At the Ahmeek mine the narrow stope and long, narrow pillar system was employed. The pillars ran up the dip and were only 5 or 6 feet thick between stopes. Thicker pillars caused the hanging wall to shear off and break, but by keeping them narrow and working on the retreat they gradually spall off or crush and settle under the weight of the hanging wall without breaking it. The ore was not as thick as at the Osceola, averaging only 7½ feet. Chutes were on the center lines of the stopes and were 21 feet apart. These were connected over the back of the level, leaving a pillar only 5 or 6 feet thick. The connection was driven about 20 feet wide measured up the dip, and from it the stopes were driven up the dip in line with the chutes. When the mine last was visited (1931) an average of 20 tons per machine-shift was broken in the stopes. The average dip was about 33°, which made it necessary to drag the broken ore to the chutes with power scrapers. Under a bonus system in force, a basic wage of $4 per shift was paid for drilling 35 to 50 feet of hole, the base footage depending upon the nature of the ground, and a bonus of 5 cents per foot was paid for each additional foot of hole drilled.

The breaking cost at the Osceola during January and February 1931 was $0.456 per ton and at the Ahmeek, $0.539 per ton.

VANADIUM MINE, RIFLE, COLO.

At the mine of the United States Vanadium Corporation, Rifle, Colo., a room-and-pillar system of open stoping is employed.[20]

The ore occurs in lenses in crossbedded sandstones, ranging in dip from 25° near the outcrop to 15° in the lower levels. The ore ranges from a few inches to 30 feet in thickness. The ore-bearing sandstone is 50 to 75 feet thick and generally forms a strong roof over the stopes. Much of the ore can be recovered without using extensive roof support, but support must be provided to the hanging wall to obtain a high percentage of extraction. The strength of the roof is such that in retreating progressive roof caving cannot be brought about, and adequate support must be provided to prevent heavy squeezes over large areas.

In first mining, rooms 25 feet wide are driven up to the next level (120 feet on the dip), leaving 25-foot pillars between the rooms. The

[20] Barwell, Blair, Mining Methods and Costs at the Vanadium mine of the United States Vanadium Corporation, Rifle, Colo.: Inf. Circ. 6602, Bureau of Mines, 1932, 9 pp.

broken ore is dragged down to the level by power scrapers. The face is advanced by driving a 12-foot breast, using a center V-cut, followed by slabbing to a width of 25 feet. The ore is tough, blocky, and highly abrasive.

After the rooms are mined from level to level the pillars are attacked, retreating from the end of a block of rooms. Artificial roof support generally is not required in mining the rooms, but occasional stulls are set to serve as a warning or to support small slabs of roof. Before the pillars are removed, however, large waste packs are built in preparation for a heavy squeeze, which may occur in recovering the last of the pillars; from 10 to 20 percent of the area is supported in this manner. Packs are 15 by 20 feet, made with stulls and heavy woven wire, and filled with waste rock from sorting in the stopes and from development. Pillars finally are blasted, and the ore is scraped out. Waste handled in the mine amounts to about 25 percent of the ore tonnage.

Self-rotated stopers with an adjustable stop board are used for drilling holes at angles of 15° to vertical and have proved more economical than mounted machines.

In mining thin ore (as little as 10 inches thick), stripping and sorting (resuing) are resorted to, and low-type scrapers are employed, which are 12 inches high and 5 feet long.

Most of the ore is scraped in medium- and thick-ore stopes, although an air-operated shovel is used to some extent in high ore. Chinaman-type chutes are used for loading cars on the levels. Considerable hand shoveling is done near the chutes and where sorting of ore in the stopes is necessary. In scraper mining three men make up a scraper crew; these men place the eyebolts for the cables, blockhole coarse chunks, and load and tram the cars to the level grizzly. In high ore 500 tons or more sometimes are broken in advance of scraping, but usually the ore is removed as it is broken. The output per scraper crew is 30 to 75 tons per shift.

Stoping costs during July, August, and September 1931, when 14,623 tons were mined and trammed, were as follows:

Stoping costs, Rifle, Colo.

	Cost per ton		Cost per ton
Labor	$0.74	Timber	$0.04
Supervision	.05	Other supplies	.10
Compressed air, drills, and steel	.09		
Power	.13	Total	1.35
Explosives	.20		

Costs in terms of man-hours per ton during the same period were as follows:

Costs in man-hours per ton, Rifle, Colo.

Occupation:	Man-hours per ton	Tons per man-shift
Breaking	$0.30	26.7
Timber and filling	.24	33.3
Shoveling	.90	8.9
Total	1.44	5.6

Consumption of explosives in stopes was 1.05 pounds per ton of ore.

OPEN STOPING IN DEPOSITS DIPPING AT HIGH ANGLES

Deposits dipping at angles steeper than the angle of repose of the broken ore are included in this category. The deposits are tabular or lenticular, wide in two dimensions and narrow in the third, or wide in the third dimension also.

GRANADA MINE, QUEBEC[31]

At the Granada mine the ore occurs in quartz veins, mostly in graywacke and conglomerate but in places close to or along porphyry sills. The sediments have a marked schistosity which strikes N. 70° W. and dips 45° to 55° to the north. The width, richness, and nature of the quartz veins are erratic; the greatest thickness of quartz is 12 feet, and some sections of the veins average 5 or 6 feet or less. The wall rock envelops the ore lenses in tight shells. Gouge is virtually absent, but the walls are slickensided. Although the wall rock contains a small amount of gold only solid quartz and stringer zones have proved to be ore so far. Assay values in the quartz are erratic because of the presence of coarse gold in shoots of high-grade ore.

Shrinkage stoping first was employed but was abandoned later for open stoping. The walls are strong, but the schistosity of the rock within 2 or 3 feet of the quartz makes the hanging wall likely to slab if left unsupported over large areas. This caused no trouble (with shrinkage stoping) until drawing of the ore was begun. The dip of the veins (45° to 55°) was not steep enough to permit even drawing over the irregular footwall, and after one-half to two-thirds of the ore had been drawn from the shrinkage stopes the remainder lay in the stopes, extending up as much as 100 feet between the chutes (20 to 25 feet apart.) Open stoping is advantageous because of the irregular tenor of the ore; ore can be drawn as mined or left in the stopes a few days, and by selecting the points from which the ore is drawn uniformity can be obtained in grade of ore sent to the mill. Where the quartz is over 3½ feet thick little waste is broken with the ore. Waste can be sorted in open stopes, using stulls covered with lagging to support discarded waste rock. During August 1932, a typical month, 12.3 percent of the rock broken was sorted out as waste and gobbed in the stopes and 5.2 percent by the trammers.

In beginning a stope two rounds are blasted out of the back of the drift, and the broken ore is mucked out, leaving the back of the stope 14 feet above the track level. Stulls are put in 6 feet above the track and 8 feet apart and lagged over tightly with poles, except that 5-foot spaces are left for chutes every 25 feet. The back is then drilled over, but before this slice is blasted inclined slides are built of poles to deflect the broken ore onto shoveling platforms alongside the chutes. After the round is blasted muckers remove the top lagging of slides, leaving an opening through which may be thrown waste sorted from the ore, the ore, of course, being shoveled into the chutes. At this stage a raise usually is driven to the level above.

[31] Loofbourow, R. L., Mining Methods and Costs at Granada Gold Mines, Ltd., Rouyn, Quebec: Inf. Circ. 6709, Bureau of Mines, 1933, 14 pp.

The stope then is advanced upward by drilling from staging on temporary stull timbers. When the stope is 30 to 50 feet above the back of the level another row of stulls is put in and lagged over 6 feet below the back, the chutes are extended up to this level, and stoping is continued as before. In a long stope the ore sometimes is shoveled into a car on the mucking level and trammed to a central chute. The minimum stoping width is 3½ feet; where the vein filling is less, the footwall usually is broken, leaving the hanging wall intact and therefore less likely to slab off. The mill superintendent estimates that 35 to 50 percent of the mill feed is wall rock. It is estimated that 98 percent of the ore is recovered, the only ore left being in 3- to 4-foot level pillars extending over about half the width of the stope to act as a footing for tracks.

During July 1932 the direct stoping costs per ton of ore hoisted were as follows:

Direct stoping costs, Granada mine, July 1932

	Cost per ton hoisted		*Cost per ton hoisted*
Labor	$1.556	Explosives	$0.606
Supervision	.087	Timber	.040
Compressed air, drills, and steel (including power)	.476	Total	2.765

As one-third of the rock broken in stopes was sorted out as waste, the cost per ton broken was $1.843.

During the same month the direct stoping cost in man-hours per ton was as follows:

Stoping cost in man-hours per ton, Granada mine

Occupation:	*Man-hours per ton*	*Tons per man-shift*
Breaking	0.809	9.9
Shoveling	.249	32.1
Total	1.058	7.6

Consumption of explosives in stopes was 3.927 pounds of 40-percent gelatin dynamite per ton hoisted, and consumption of timber was 1.65 board feet per ton.

MARY MINE, ISABELLA, TENN.

Figure 25 shows the stoping method employed at the Mary mine in the Ducktown (Tenn.) district.[22]

The method sometimes is termed "underhanding" and is an old Cornish system. It is similar in some respects to the system used at Mascot, but a regular or checkerboard stope-and-pillar pattern is employed. Pillars on each level are directly under those on the level above, forming a continuous pillar from bottom to top of the ore.

The ore occurs in lenses replacing certain folded and faulted beds in highly metamorphosed sediments consisting mainly of graywacke

[22] Kegler, Vern L., Mining Methods of the Ducktown Chemical & Iron Co., Mary Mine, Isabella, Tenn.: Inf. Circ. 6397, Bureau of Mines, 1931, 9 pp.

with arkose, graywacke conglomerate, mica schist, slate, staurolite schist, and garnet schist. The shape of the ore bodies is controlled largely by folding of the country rocks, which exhibit swells on the crests of anticlines and domes and pinches on the flanks. The ore bodies are displaced by postmineral faults. They strike N. 45° E. and dip 65° southeast. The Mary ore body is 2,000 feet long and ranges from a few feet to 150 feet in thickness. The wall rocks are

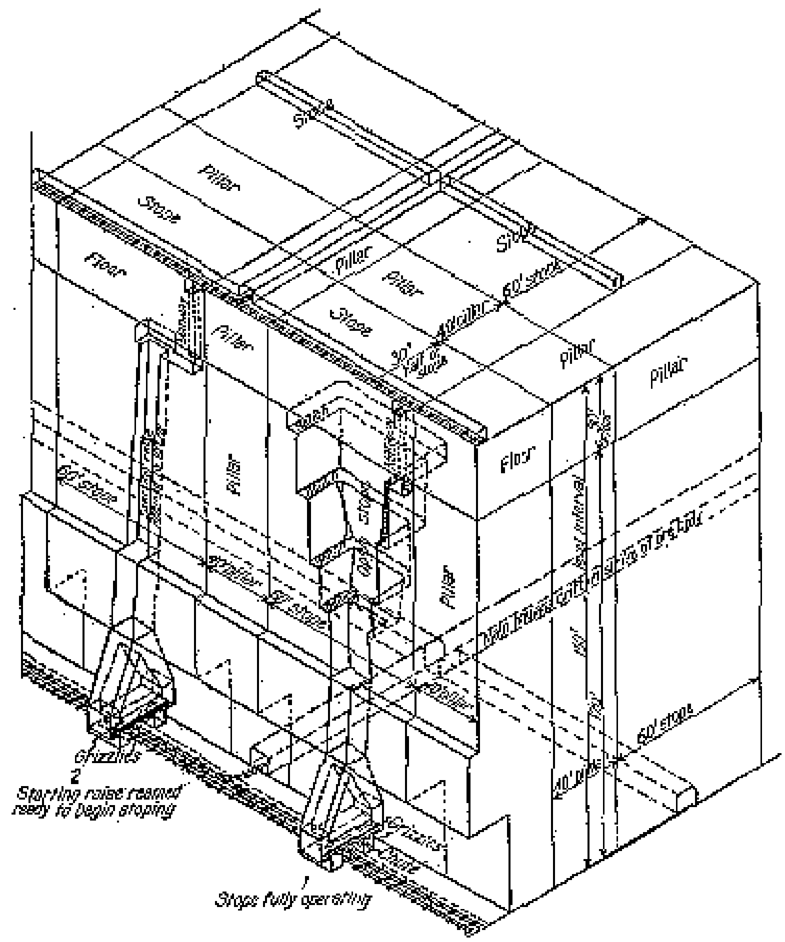


FIGURE 25.—Stoping method at Mary mine, Isabella, Tenn.

mica schist and graywacke and stand well over spans of 100 feet; in some places in the mine 150-foot spans have been self-supporting.

The principal ore minerals are pyrrhotite and chalcopyrite, and the ore is used primarily for the manufacture of sulphuric acid, the copper being a byproduct. The principal gangue minerals are quartz, amphibole, and diopside, with smaller amounts of dolomite, calcite, zoisite, epidote, and garnet. The ore is hard to drill, and after blasting some bulldozing is required.

The chamber and stope system, in which the ore is benched down as in heading-and-bench stoping, was employed almost exclusively, although some breast stoping was done where the ore extended only a short distance above the level. The regular pillar arrangement was not always adhered to strictly, as it was possible, by varying the arrangement somewhat, to leave much of the leaner ore in pillars and extract a higher percentage of better-grade ore. Sharp changes occur in the grade of the ore, due to dislocations of the ore body by faults.

A central raise is put up in each stope area, and this then is reamed out to 10 feet in diameter. Starting just under the floor pillar of the level above, which is 30 feet thick, a heading is driven around the raise. As this is widened the ore below is benched down with vertical holes. At first most of the ore falls directly into the raise, but after the stope is widened considerable ore is caught on the benches below and must be shoveled off. (This constitutes one of the disadvantages of the method.) Another disadvantage is the inability to inspect and trim the high roof over the lower benches after the stope becomes wide. It is necessary to take down very carefully any loose ground in the back while it is still accessible from the heading bench. The backs of all stopes are arched to lessen the danger of falling ground. One advantage of the system is the thorough exploration of the ore body before stoping.

After the stopes or "chambers" have been mined out to the pillar lines small sections of the pillars are cut out between levels connecting the stopes. Later the floors are extracted by benching them down into the stopes, and finally the pillars on the levels are trimmed to the minimum size consonant with safety. The method would not be suitable for mining a high-grade ore because of the large amount of ore left in pillars.

At the Mary mine stope labor was paid 15, 20, or 25 cents per 1-ton car of ore delivered to the level, the price depending upon the character of the ore and the area and height of the stope. Each crew of contractors usually had two drillers, who paid for the additional labor required in the stopes and for the explosives.

In 1928, 108,519 tons of ore were mined and hoisted, of which 99,551 tons came from stopes. During that year a considerable part of the tonnage was derived from marginal areas where the ore was very irregular and where considerable development was necessary with the stoping. As a result costs were higher than in the larger, more regular stopes. Direct stoping costs for that year are given below.

Direct stoping costs, Mary mine, 1928

		Cost per ton
Labor (stoping and blockholing)		$0.270
Labor (hand shoveling and tramming)	$0.319	
Deduct 25 percent for tramming on level	.080	
Labor (hand shoveling attributable to stoping)		.239
Supervision		.022
Compressed air, drills, and steel		.177
Explosives		.076
Total		.784

The productivity of stope labor, as measured in man-hours per ton, was as follows:

Productivity of stope labor, Mary mine, 1928

Occupation:	*Man-hours per ton*	*Tons per man-shift*
Breaking	0.426	18.8
Blockholing	.130	61.5
Shoveling and tramming, less 25 percent for tramming on level	.467	17.1
Total	1.023	7.8

The consumption of explosives in stopes was 0.42 pound of 40-percent gelatin dynamite per ton of ore broken.

BURRA-BURRA MINE, DUCKTOWN, TENN.[28]

At the Burra-Burra mine the sublevel stoping variation of open-stope mining is employed. The ore bodies are described as replacements of a limestone bed, thickened in places by thrust folds and strike faults, in wholly metamorphosed Lower Cambrian sediments consisting mainly of graywacke with arkose, graywacke conglomerate, mica schist, slate, staurolite schist, and garnet schist. The Burra ore body dips 75° southeast at the surface, flattens to 50° at the sixteenth level, and locally dips as low as 35°. It occurs over a length of 2,300 feet on the fourteenth level, where the minable thickness ranges from a few feet to a maximum of 180 feet. The immediate wall rocks are highly metamorphosed schists and graywackes, the bedding planes of which parallel the ore in dip and strike. The schistosity generally, but not always, follows the bedding.

The walls generally stand well, and spans of 100 feet are generally self-supporting, except where the ground is fractured due to folding. The ore itself is characterized by few prominent slips and joints which, where they do appear, are usually horizontal. The ore tends to adhere to the walls and, although it is not self-supporting over as long spans as the walls, is firm and hard and stands well. It breaks large, and considerable blockholing is required on the grizzlies under the stopes. In the primary zone to which mining has been confined in late years, the principal ore minerals are massive sulphides—pyrrhotite, pyrite, and chalcopyrite. The gangue minerals are chiefly lime-bearing silicates, quartz, and calcite with which the sulphide minerals are intergrown. The average grade of the ore in 1928 and 1929 was 1.6 percent copper, 24 percent sulphur, and 32 percent iron. The ore must be graded closely to produce the required percentages of sulphur in the direct-smelting ore and in the ore which goes to the flotation plant, it being primarily a sulphur ore used for the manufacture of sulphuric acid.

Before 1910 ore above the sixth level was mined by underhand stoping in open stopes with irregular pillars for support of the walls and backs. Spans of 50 feet or more between pillars were

[28] McNaughton, C. H., Mining Methods of the Tennessee Copper Co., Ducktown, Tenn.: Inf. Circ. 6149, Bureau of Mines, 1929, 17 pp.

common. Since then a large percentage of the pillar ore has been recovered. Later, shrinkage stoping was employed. Above the tenth level these stopes were carried longitudinally and virtually the full width of the ore. Below this level transverse stopes 80 feet wide with 30-foot pillars between were employed in wide ore and longitudinal stopes in narrow ore.

In 1925 the sublevel stoping method was introduced on the fourteenth level and is now the principal method employed, but in 1928 only 32.6 percent of the output came from sublevel stopes. During

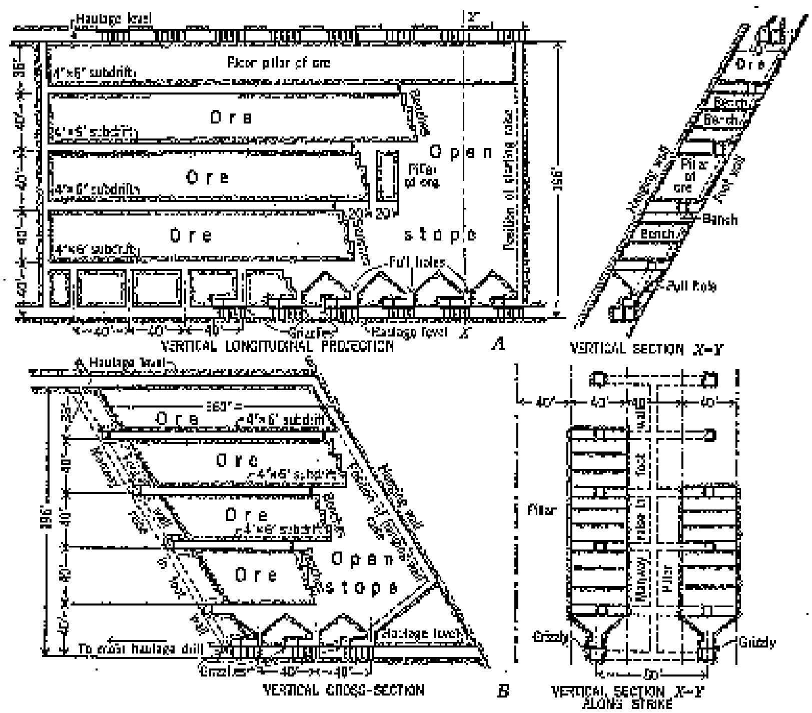


FIGURE 26.—Sublevel stoping in hard ore, Burra-Burra mine, Ducktown, Tenn.: A, Longitudinal stope in narrow ore; B, transverse stopes in wide ore.

that year 23.3 percent of the ore came from mining pillars under old shrinkage stopes, 9 percent from shrinkage stopes, 17.6 percent from floors and pillars on the upper levels, 12.4 percent from flat stopes and pulling caves, and 5.1 percent from development in ore.

Where the ore body is less than 40 feet wide sublevel stopes are run longitudinally. (See fig. 26, *A*.) The block to be stoped is developed by long raises driven on the footwall to the level above (196 feet vertically between levels). In the block shown in figure 26, *A*, two of these raises were driven, one at each end of the block. At 40-foot intervals along the haulage drift 4- by 5-foot raises called "pullholes" are driven to the first sublevel. Sublevel drifts are 40 feet apart vertically. Before stoping is begun chambers are opened over the haulage drift at one end of the block, and grizzlies are installed immediately over the drift timbers. The total stope development required for preparing a stope 40 feet wide and 320 feet long is as follows:

	Feet
Long raises (2)	450
Sublevels (4)	1,360
Pullholes (8)	400
Total	2,210

The tonnage developed, exclusive of ore in pillars over the haulage drift and in floor pillars, is 206,000 tons; thus 1 foot of development is required for every 93 tons of ore developed. This development assists later in extracting the pillars, which contain an additional 103,000 tons, 60 percent of which (62,000 tons) can be recovered later. Thus a total of 120 tons of recoverable ore is developed per foot of stope development.

Stoping is begun at one end of the block, after the tops of the pullholes have been funneled by drilling downward from the bottom sublevel; at the same time the hanging-wall side of the subdrift is slabbed off, thus exposing both walls. Beginning at the long raise on the second sublevel the drift is slabbed out to the walls, and the raise is benched out the full width of the ore. A slice about 5 feet thick and 6 feet high, extending from wall to wall, then is drilled and blasted below the sublevel, providing a second bench 6 feet lower from which downholes can be drilled to form another bench. Another row of holes is drilled from the sublevel, widening the second bench, and downholes are drilled therefrom to a third bench, forming a stepped face as shown in the drawing. The stope then is continued in this form by slabbing and benching, retreating from the raise. The two bottom benches are drilled with holes 10 feet deep. The broken ore falls into the funneled tops of the pullholes and is put through the grizzlies into cars on the level. When benching on the second sublevel has retreated far enough, slabbing and benching are started on the third sublevel and, similarly, on the fourth sublevel when the benches on the lower sublevels are back far enough. Stoping may be done simultaneously from all sublevels. The bench offsets are so proportioned that men working on the benches between sublevels are never under ground which cannot be tested from the sublevel above with a 15-foot bar. At the Burra-Burra mine benching is done entirely with downholes; in this respect practice at the mine differs from that at many other mines employing the sublevel-stoping system, where the stope face is broken back by both up- and down-holes from the sublevel bench and the men do not work on benches between the sublevels.

Bench holes are 4 to 5 feet apart along the face, with a burden of 2½ feet. The slabbing rounds for widening the upper bench are drilled 10 feet deep with three holes in a vertical row and 2 to 2½ feet of burden on each row of holes. Usually 9 holes can be drilled in a shift.

Where the ore body is more than 40 feet wide the sublevel stopes are opened up 40 feet wide across the strike of the ore body with 40-foot pillars between the stopes. (See fig. 26, *B*.) These stopes are developed from crosscuts on the haulage level in the same manner that the narrow longitudinal stopes are opened from the longitudinal drift. Up to 1929 no pillars had been recovered between transverse stopes, but the method of recovery will be the same as that employed in mining the pillars between the old shrinkage stopes. This is done by partly undercutting the pillar 50 feet above the haulage level and blasting the broken ore into pullholes of the old stope. Then, with deep-hole hammer drills set in stations cut from the footwall raise opposite the middle of the pillar, long holes are fanned out into the pillar from a few feet to a maximum of 120 feet in depth. The holes are loaded with 60-percent gelatin dynamite and detonated with cordeau, starting with the bottom row of holes. The charge is 0.1 pound of explosive per ton of ore.

Drilling in the stopes cost $0.07 per linear foot of hole drilled (1928–29), and an average of 48 tons per drill-shift was broken in the stopes, the powder consumption being 0.33 pound of 35-percent bulk powder per ton of ore broken. These figures include opening the stope, cutting grizzly chambers, and funneling the pullholes. An average of 72.5 tons per man-shift was put through the grizzlies, with a powder consumption of 0.167 pound per ton of ore. Labor for blocking and loading through grizzlies was paid at 4 to 7 cents per ton.

In 1928 direct stoping costs, including blocking through grizzlies and covering all stoping—sublevel stoping, mining pillars, shrinkage, and pulling caves—were as follows:

Direct stoping costs, Burra-Burra mine, 1928

	Cost per ton
Labor	$0.196
Compressed air, drills, and steel	.051
Explosives	.083
Other supplies	.002
Total	.332

In 1932, when a greater proportion of the ore came from sublevel stopes, stoping costs were as follows:

	Cost per ton
Labor	$0.100
Compressed air, drills, and steel	[21].081
Explosives	.078
Total	.259

Stoping and blocking costs during the first 4 months of 1933 totaled about 21 cents per ton.

Productivity of all stope labor in 1928, when only 32.6 percent of the total output of the mine was from sublevel stopes, was as follows:

Productivity of labor, Burra-Burra mine, 1928

Occupation:	Man-hour per ton	Tons per man-shift
Drilling and blasting	0.216	37.0
Blocking through grizzlies	.132	60.6
Subtotal	.348	23.0
Add shoveling (mostly in other than sublevel stopes)	.258	31.0
Total	.606	13.2

[21] Small tonnage being mined increased pro-rata power cost.

MENOMINEE RANGE, MICHIGAN

The sublevel system of open stoping is used extensively on the Menominee range in Michigan. The ore bodies occur in iron formations consisting of ferruginous cherts interbedded with slates and unaltered iron-bearing rocks. The structure of the beds is characterized by close and complex folding and some faulting. The ore bodies usually dip at steep angles and occur on one limb of a major fold, but frequently they occupy the entire trough formed by a minor plunging fold and are very irregular in outline. The ore is a firm, but not very hard, hydrated hematite which stands well in large stopes and is not affected by exposure to the air. The ferruginous cherts are harder than the ore and stand well. The slates forming the walls swell on exposure to the atmosphere and, as they contain appreciable pyrite and carbonaceous material, are apt to catch fire if exposed long in dry places.

Usually the sublevel stopes retreat from the starting raise with an overhanging face, as in figure 2, holes being drilled upward and downward from the sublevel benches without intermediate benches such as are used at the Burra-Burra mine. The sublevel interval usually is 20 to 25 feet, and with this spacing the entire bench between sublevels can be removed by rounds drilled downward from one sublevel and upward from the next sublevel below. In the system shown in figure 2 the ore is blasted into millholes funneled out above a grizzly level and is put through grizzlies into chutes, whence it is drawn into cars on the haulage level. In some mines where the ore breaks smaller no grizzlies are used. It is advisable to keep the face of the stope as nearly vertical as possible and at the same time to maintain the bench on each sublevel far enough back of that on the sublevel above so that the miners are protected from ore falling from the level above. An overhang of about 8 feet between sublevels is considered ideal.

Stopes are carried either longitudinally the full width of the ore body when the ore is narrow, or transversely with pillars between stopes in wide ore. The pillars may be recovered later. In the former case the ore body may be sectionalized to provide more stopes, and several longitudinal stopes may be worked at one time, the stopes being separated by small pillars as shown in figure 2. The pillars between stopes also reduce the unsupported area of backs in the stopes. The occasional collapse of stope backs has not been attended by serious consequences. When this occurs it is only necessary to step back a short distance, leaving a thin pillar of ore against the cave, and to resume stoping from sublevels already established.

At one mine near Iron River the stopes average 50 feet in width and are 90 to 125 feet in length. The ore is firm, strong, and moderately hard. Main levels are 160 feet apart vertically, and the sublevel interval is 25 feet. The stopes are carried longitudinally and separated by pillars 25 to 40 feet thick. Manway raises for entrance to the stopes are carried from level to level in alternate pillars, the other pillars being left blank. The ore is blasted into millholes, from which it is put through grizzlies into chutes. The amount of stope development per ton of ore stoped is somewhat higher than at some mines where no grizzly chambers and grizzly levels are used. About 45 tons of ore is developed per linear foot of stope development, which costs about 8 cents per ton of ore made available for stoping. Light, mounted hammer drills are employed, and 50 to 60 feet of hole is drilled per man-shift. Breakage per man-shift for stoping only is 40 to 80 tons of ore and for stoping and development combined 24 to 40 tons. Explosive consumption averages 1.15 pounds of 40-percent gelatin dynamite per ton broken in stoping and development combined. In 1929 four active stopes were producing 700 long tons per day and could have produced 1,000 tons per day if forced.

At another mine main levels are 120 feet apart and sublevels 25 feet apart vertically. The ore runs to a maximum of 220 feet in width. In wide ore, longitudinal haulage drifts are driven in the ore body on 45-foot centers. Raises are put up on 15-foot centers on alternate sides of the drifts to the first sublevel, where they are belled out to receive the ore to be blasted from the stope faces above. Stope grizzlies are not employed at this mine. The ore is quite hard and dry and breaks with a large percentage of fines. It is drilled with light, mounted hammer drills, two men working together on each machine. An average of 75 to 100 feet of hole is drilled per machine-shift, but as much as 150 feet can be drilled under favorable conditions and when it is not necessary to rig up and tear down each shift. Holes are up to 14 feet deep. The backholes are the easiest and cheapest to drill and often are drilled with a heavy burden of 6 to 8 feet. Some stopes have been 200 feet long and 150 feet wide, with long bench faces across the stope. These often are curved in plan, giving benches considerably longer than the width of the ore body. In these wide stopes the capping eventually caves. A thin pillar of ore then is left to hold back the cave, and the stope is again opened up. The stope faces are carried with only a slight overhang, so that if a cave occurs a vertical pillar can be left, thus tying up a minimum of ore.

During the first half of 1929, 44 tons were produced per miner per shift, including miners on development. Since there were as many miners on development as on stoping the production per miner per shift in stopes was about 80 tons. In 3 years and 9 months 976,896 tons were produced from an average of 6 stopes, or about 150 tons per stope per day. In May 1929, 24,535 tons were produced from 3 stopes, working 5 days per week and only 1 shift per day. This is equivalent to 350 tons per stope per day. Consumption of explosives averaged 0.569 pound per ton, including development. Over a period of years 55 tons of ore were made available for stoping per linear foot of stope development.

At some mines where sublevel stoping is used the ore is soft enough so that it can be drilled with light machines using auger steel and is too weak to hold an overhanging bench. Indeed, sometimes it is necessary to slope the stope face slightly outward toward the bottom. Each sublevel is thus stoped back farther than the one below. In such cases each sublevel must be developed with a series of longitudinal drifts if the ore body is wide, the drifts being separated by pillars of such thickness (about 20 feet) that they can be drilled and blasted out by long holes drilled into the sides of the drifts. The subdrifts are driven small in cross section. At the face of the open stope they are widened, and the top is taken down to make room for long steels employed for drilling upward and downward into the benches and laterally into the pillars between the drifts. A round of holes is drilled up and down and laterally into the pillars, and all are blasted together, the broken ore falling down the stope face into millholes below. As the miners work back of the face in the subdrifts they are protected by solid ground above them. In wide ore more sublevel development is required than where benches are carried, because of the greater number of drifts. Where the ground will not hold a bench, however, it usually is much softer, the cost per foot of driving the subdrifts is less, and the development cost per ton of ore stoped is about the same as in the overhanging bench system used in harder ore.

At one mine using the vertical or outward-sloping stope face the ore body was very irregular in outline and extended 350 feet above the haulage level. It was worked as one high stope with sublevels 25 feet apart vertically. When the mine was visited in 1929, 3 gangs of 2 men each were working in the stope (12 men on the 2 shifts), and the stope was producing 600 tons of ore per day or 50 tons per man-shift. By working all the sublevels at the same time the output could have been increased considerably. The stope face was curved in plan and on some sublevels was 200 feet long. During 1928, 200,000 long tons of ore were mined, and the direct stoping costs were as follows:[25]

Direct stoping costs, mine no. 1, Menominee range

	Cost per long ton		Cost per long ton
Labor	$0.246	Timber	$0.009
Supervision	.031	Other supplies	.011
Compressed air, drills, and steel	.080	Total	.467
Explosives	.090		

Productivity of stope labor (including stope development), as measured in man-hours per ton, was as follows:

Occupation:	Man-hour per ton	Tons per man-shift
Breaking	0.353	22.7
Timbering	.038	210.5
Shoveling	.011	727.2
Total	.402	19.9

Consumption of explosives in stopes was 0.619 pound of 40-percent gelatin dynamite per ton of ore broken.

At another mine in which most of the ore bodies were small the largest stope was about 100 by 100 feet in plan and 150 feet high. Sublevels were 25 feet apart vertically, and the drifts on each sublevel were 25 feet apart center to center. The ore was soft and was drilled with jackhammer augers. Raises were put up alternately on the right and left sides 15 feet apart along the haulage drift.

The stope face sloped outward toward the bottom, as the ore was too soft and weak to hold a bench. The ore broke fine, and it was

[25] Eaton, Lucien, Mining Soft Hematite by Open Stopes at Mine No. 1, Menominee Range, Michigan: Inf. Circ. 6180, Bureau of Mines, 1929, 10 pp.

unnecessary to use grizzlies. In stoping, 80 to 100 tons of ore were broken per man-shift. In one big stope two men regularly broke 100 cars of 2.3 long tons each per shift over a considerable period of time.

A sublevel stoping method was employed under rather unusual conditions at the Carpenter mine, Crystal Falls, Mich.[26] The ore body filled a pitching trough averaging 500 feet in length and about 300 feet in width. The surface material, consisting of fine, wet sand, gravel, and a little hardpan, was about 175 feet thick. Haulage levels were 150 feet apart vertically and were driven in ore near the walls of the trough. Thus there were two longitudinal drifts on each level. These were connected by crosscuts spaced 80 feet apart on the center lines of a series of 40-foot sublevel stopes separated by 40-foot pillars. Later crosscuts were driven under the center of the pillars. The stopes were developed by raises from level to level on the center line of the stope along one wall, one to serve as a starting raise for the stope and the other as a manway raise from the drift near the other wall. These raises were connected with a crosscut on each sublevel. The first sublevel was 30 feet above the haulage level, and succeeding sublevels were 25 feet apart. The top sublevel was 20 feet below the upper worked-out haulage level, leaving a 15-foot pillar under the level. Chute raises were staggered along the main level crosscuts at 12- to 15-foot intervals and were put up to the first sublevel; later they were belled out as stoping retreated toward the haulage drift. Stoping is begun on the lowest sublevel at the starting raise, which is that farthest from the manway raise and the haulage drift. A bench is cut right and left from the sublevel crosscut to the pillar lines, the broken ore falling into the raise, and as the stope face retreats each sublevel bench is farther back from the raise than the bench above, thus giving an overhanging stope face. Upper holes were drilled 8 to 10 feet deep, and the back was blasted down; then, starting from the pillar end of the bench, downholes were drilled and blasted, knocking down the bench into the raise. The stope was continued backward from the raise until the back showed signs of weakening, then bulkheads were put in each sublevel crosscut, and a mat of ore was left in the bottom of the stope. The bulkheads were to keep sand and surface material from filling up the subdrifts, while the mat of ore protected the main haulage drift. They were of timber and had a small door through which entrance to the stope could be gained. They were placed directly above each other on the several sublevels at a point where it was thought stoping operations could be carried before the stope would cave through. If the stope did not cave after 2 or 3 days it was entered through the bulkheads, and another bench was shot off each sublevel to within 5 feet of the line of bulkheads. Then, if the stope did not cave, long holes were drilled in the 15-foot back pillar, and after the bulkheads were closed these holes were blasted electrically. After the stopes had caved and the water had drained through the chutes and bulkheads the broken ore mat and the ore from the caved back pillar were drawn through the chutes until dilution with sand prohibited further drawing. Then a new start-

[26] Wortley, R. R., Novel Method of Sublevel Stoping: Eng. and Min. Jour., vol. 126, no. 22, 1928, pp. 867–868.

ing raise was put up just back of the bulkheads, and stoping was continued as before.

After the stopes on each side of a 40-foot pillar were mined and caved the pillar was developed in the same manner as the stopes, except that the middle or 80-foot sublevel was driven large enough to accommodate chutes and sublevel transfer cars. Starting at this sublevel the upper half of the pillar was mined by sublevel stoping, dropping the ore into chutes in the subdrift. As the stope retreated test holes were drilled into the walls to determine the position of the sand in the old stopes. When a pillar was mined between stopes in which sandy material containing some clay had remained for a year or more and had dried out the fill would stand without support from the ore, and the pillar could be mined clean. When the fill was not firm it was necessary to leave a shell of ore 3 to 5 feet thick to hold it back. After the part of the pillar above the 80-foot sublevel was mined the lower part was mined in a similar manner, leaving a 10-foot floor pillar below the 80-foot level.

MARQUETTE RANGE, MICHIGAN

Sublevel stoping is employed in part of mine no. 2 on the Marquette range.[37]

The ore occurs in troughs and folds as lenticular masses, sheets, and chimneys. The dip of the formation is fairly steep at the surface and becomes flatter at depth. The ore bodies are small and scattered. The ore itself is a soft, hydrated hematite which can be drilled with auger steel and generally requires timber support, except in the smaller openings. The overlying jasper is fairly hard and generally stands well. The slate underlying the ore is somewhat friable and drills and breaks readily, but as it stands at a steep angle little timber is required to support drifts through it. Much of the ore is mined by top slicing, but in some of the narrow, steeply dipping ore bodies, sublevel stoping can be employed, and this method is used whenever possible. If the ore extends upward from a main level a drift is driven lengthwise of the ore body, and raises are put up at 25-foot intervals on the footwall side to the first sublevel 25 feet above the floor of the main level. The main level is timbered, and a chute is built in every raise but one, which is used as a manway. The raises are continued to the top of the ore and connected on successive sublevels, which are 20 feet apart vertically. Stoping is begun on the first sublevel in the raise farthest from the manway. The raise is opened out into an inverted cone the full width of the ore body, usually 14 to 25 feet, and uppers are drilled in the back around the raise and are blasted. This is repeated on successive sublevels until the stope has been opened through to the top of the ore. Starting on the top sublevel two vertical rows of horizontal holes then are drilled in the side of the sublevel drift at right angles to its course and blasted. This leaves room enough on the bench thus formed to drill long holes both ways from the drift to reach the walls of the ore body. When these are blasted a cut 7 feet high is left across the ore body. Then a double row of upper holes is drilled

[37] Caton, Lucien, Mining Soft Hematite at Mine No. 2 of the Marquette Range, Michigan: Inf. Circ. 6179, Bureau of Mines, 1929, 14 pp.

and blasted in the brow over this cut, leaving room to drill 10-foot holes into the remainder of the brow above. This process is repeated on successively lower sublevels, and the stope face retreats from the end of the ore body, maintaining an 80° slope backward from the chute.

Direct stoping costs for 1928 in sublevel stopes at this mine were as follows:

Direct stoping costs (sublevel stoping), Mine No. 2, Marquette range, 1928

	Cost per long ton of ore mined		Cost per long ton of ore mined
Labor	$0.180	Explosives	$0.050
Supervision	.025	Timber	.010
Compressed air, drills, and steel	.030	Other supplies	.075
Power	.047	Total	.417

NORANDA, QUEBEC

At the Horne mine, Noranda, Quebec, a method is employed similar in principle to the sublevel-stoping method for mining large bodies of massive sulphide ore. The large H ore body at this mine has a maximum width of about 300 feet in places. The ore and wall rocks stand well over spans of 50 feet or more. Levels are 125 feet apart vertically, and large haulage drifts are driven along or near each wall or along the center of the ore body on each level. Stopes 46 feet wide are laid out between 35-foot pillars across the ore body and are developed by inclined raises from the haulage levels. A series of parallel branch raises is driven from the main raises in each stope. These parallel raises correspond to the sublevel drifts in sublevel stoping, and from them the ore is broken down into the open stopes, whence it is drawn off through chutes installed at the bottom of the main raises. Figure 27, *A*, is a vertical cross section through a fully developed stope area extending from the ninth to the fifth level, and figure 27, *B*, shows the method of stoping from inclined stope raises by drilling up- and down-holes. The ore is a high-grade copper ore carrying appreciable gold values, and after the stopes have been filled it will be necessary to mine the pillars between them. The sulphide ore takes on a partly oxidized film after exposure to the atmosphere for a few days; and milling ore, concentrated by flotation, must be removed as rapidly as it is broken and put through the mill immediately to obtain maximum recovery. The stoping method employed permits immediate drawing of the ore.

In stoping, the raises first are slabbed out to the pillar lines, making them 46 feet wide. Then, beginning at the lower series of parallel branches, the ore is blasted off by up- and down-holes. Similarly, benches are blasted from the other branches as soon as the lower one is far enough back so that ore falling from above will not endanger the workmen below. Entrance to the stopes is through manway raises in the pillars, from which small drifts are driven to the stopes. By the use of inclined raises instead of sublevels no mucking is required, either in development of the stopes or in slabbing operations. The disadvantages are that the men are work-

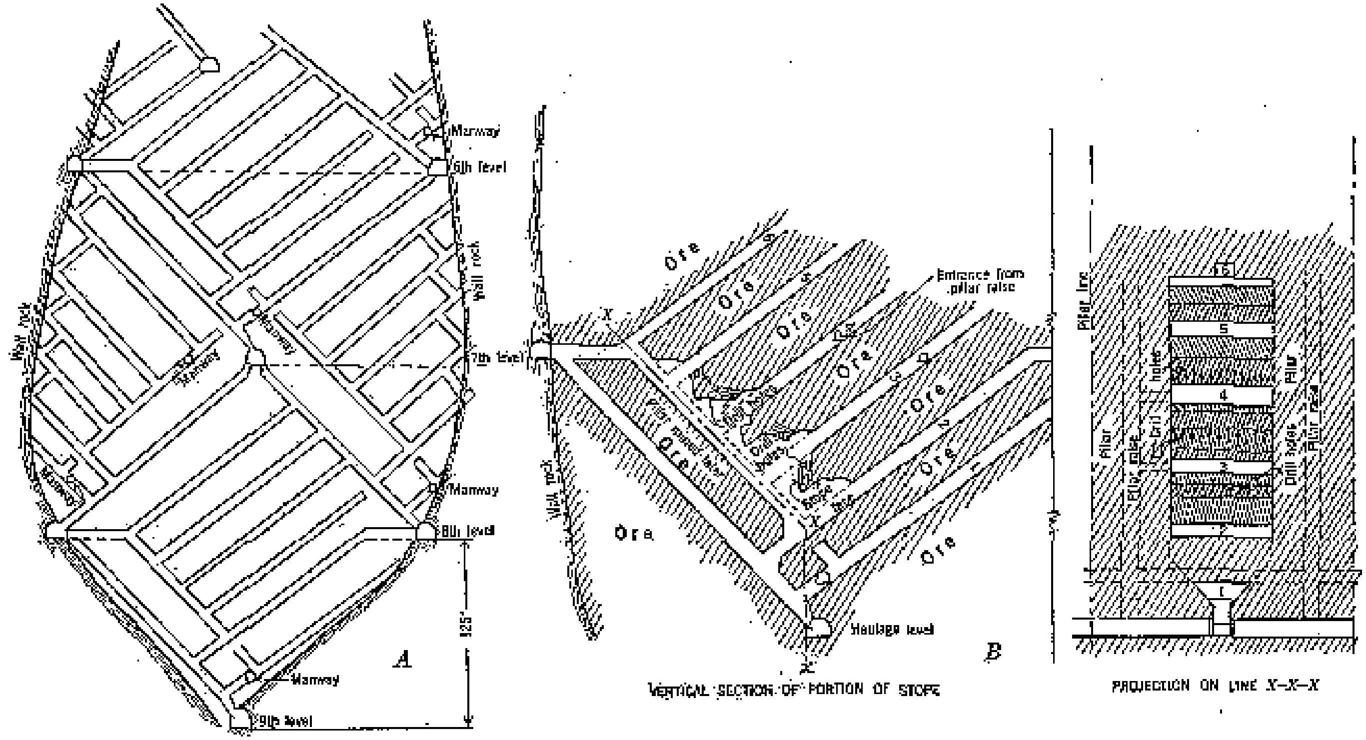


FIGURE 27.—Open stoping at Horne mine, Noranda, Quebec: *A*, Vertical section through fully developed stope area; *B*, method of open stoping from inclined stope raises.

ing on an inclined bottom at the edge of a high open stope; it is difficult to reach and trim the back; and handling of steel, drills, and drilling gear is more difficult than on level benches.

SHERRITT GORDON MINE, MANITOBA

Brown[26] has described the stoping methods at the Sherritt Gordon mine. The ore occurs in two elongated, lenticular bodies. The east ore body is 4,200 feet long and averages 15.2 feet in width. It dips 45° north at the west end, steepening gradually toward the east to no. 1 shaft, where it is vertical, and then changes to a southerly dip which flattens to 40° at the east end. The west ore body is 5,800 feet long, with an average width of 15.5 feet. The dip ranges from 30° to 65°. In both ore bodies the width varies greatly, ranging from a few inches to over 50 feet. The ore and walls are firm and stand well over wide spans.

An adaptation of the sublevel-stoping system employed at the Burra-Burra mine is employed for stoping ore dipping at angles of 45° to vertical. The levels are 150 feet apart measured on a 51° dip (the shaft inclination). Instead of sublevels being used, however, the ore is benched down in single slices or benches about 8 feet wide from one level to the next. Boxhole raises were put up on the footwall side of the haulage drift on the third or 500-foot level at intervals of 30 feet. These are funneled out above the level until the funnels connect, and the ore is slabbed off to expose both walls on the stope floor above the raises. This slabbing was contracted at $0.40 to $0.70 per ton for labor and explosives, the price varying with the width of the ore.

Starting raises are driven through from the third level to the second level, then to the first level, and finally to the sublevel below the surface pillar. These raises are 120 feet apart along the strike of the ore body. Stoping is begun at the second level by slabbing around a starting raise to expose both walls and cutting a bench about 8 feet wide on both sides of the raise. The bench on one side is then carried down until it breaks through to the stope floor below, the broken ore falling into the belled-out boxholes at the third level. Holes in the benches are drilled 10 feet deep, and each blast carries the bench down that distance. Succeeding benches are carried down in the same manner, retreating from the raise. After a stope has retreated 30 to 40 feet back from the raise a similar stope is begun on the next higher level, the ore falling down through the lower stope to the boxholes at the third level. Similarly, when this stope is back far enough the stope above is begun and mined back.

Where the ore was over 9 feet wide three men were employed on each bench; in narrower ore only two men were employed. All work in stopes was done on contract. Contract prices covering labor and explosives were as follows:

Width of ore:	Price per ton
Over 15 feet	$0.30
Over 12 feet, less than 15 feet	.35
Over 9 feet, less than 12 feet	.40
Over 6 feet, less than 9 feet	.45
Less than 6 feet	.50

[26] Brown, Eldon L., Mining Methods and Costs at the Sherritt Gordon Mine: Canadian Inst. Min. and Met. Bull. 257, September 1933, pp. 468–494.

Where the dip is less than 45° overhand open stopes are used. A stope floor is cut out in the same manner as in the steeper ore, except that regular pillars divide the ore body into 100-foot sections. A small heading is driven into the hanging wall for about 10 feet to serve as a scraper station. A bench is begun each way from the foot of the raise in the center of the stope and carried horizontally 45 to 50 feet and the full width of the ore, except where it exceeds 8 feet. In ore over 8 feet wide the bench is carried 6½ feet high along the hanging wall, and the ore left on the footwall is taken up by underhand working, using pluggers. After the first bench is finished a second is begun from the raise in the same manner. When the back of the stope gets so far above the boxholes that the muck starts to pile up on the footwall, a double-drum scraping outfit is put in operation for scraping the ore down to the boxholes. Ore is scraped successfully for distances of 200 feet on the incline.

Direct stoping costs during 1931 and 1932 were as follows:

Direct stoping costs, Sherritt Gordon mine, 1931–32

	Costs per ton of ore			Costs per ton of ore	
	Dip, 45° to 90° (1931–32)	Dip, less than 45° (1932)		Dip, 45° to 90° (1931–32)	Dip, less than 45° (1932)
Labor	$0.298	$0.302	Compressed air	$0.128	$0.061
Explosives	.210	.140	Scraping		.054
Drill repairs and drilling supplies	.056	.096	Mine, general	.059	.074
Steel and steel sharpening	.007	.007	Total	.747	.740

Costs include write-offs on all machines and equipment used. The average cost is slightly lower on dips of less than 45° because flat stopes were not begun until January 1932, when general efficiency in the mine was better than during the first 8 months of 1931.

SPRING HILL MINE, MONTANA [29]

The ore occurs in large, irregular, contact-metamorphic deposits containing gold as the only valuable mineral. The contacts are between the limestone and an intrusive body of fine-grained diorite. The average grade of the ore is about $6 per ton in gold. It is extremely hard and tough and will stand over spans of 60 feet or more, although it is cracked and seamed in many directions. The ore is quite irregular in outline. The dip is nearly vertical in most places but locally flattens to as low as 60°.

Levels are spaced at 100-foot vertical intervals, from which the ore was developed and mined for a time by shrinkage stoping. Owing to the width of the ore and the nature of the ground shrinkage stoping, entailing working under wide backs, was dangerous, and a modified sublevel-stoping system was inaugurated. An intermediate level or sublevel was driven halfway between the haulage levels, from which the ore is benched down into the partly completed shrinkage stopes below, as in regular sublevel stoping. Where

[29] Pierce, A. L., Mining Methods and Costs at the Spring Hill Mine, Montana Mines Corporation, Helena, Mont.: Inf. Circ. 6402, Bureau of Mines, 1931, 11 pp.

the ore body is narrow shrinkage stoping is still employed. It is estimated that 90 percent of the ore above the haulage level will be extracted by the present system. There is virtually no dilution of the ore with wall rock, and the latter stands well. Pierce compares the old (shrinkage) and new (sublevel-stoping) systems as follows:

In a wide ore body, sublevel stoping has the following advantages over shrinkage stoping: (1) Sublevel stoping is safer because men are working under a freshly broken and barred back which is readily accessible during the short time it serves for protection, and the men are not exposed to the danger of being drawn down with the ore if the trammers pull the wrong chute or if the ore hangs up and suddenly drops; (2) more ore is recovered, as irregularities are found and followed more easily; (3) the cost of mining is lower because boulders do not have to be dug out of the ore to be blockholed, and none escape the grizzly to slow up tramming; (4) it eliminates the investment in broken ore, which becomes excessive with a large ore body; (5) tramming from any given pullhole does not have to stop because men are working above it.

The direct stoping cost, August 1, 1929, to April 30, 1930, was as follows:

Direct stoping cost, Spring Hill mine, Aug. 1, 1929, to Apr. 30, 1930

	Cost per ton	
Labor,[30] mining	$0.464	
Labor, blockholing	.274	
		$0.738
Compressed air, drills, and steel		.124
Explosives		.178
Other supplies		.032
Total		1.072

Productivity of stoping labor in man-hours per ton was as follows:

Productivity of stoping labor, Spring Hill mine

	Man-hours per ton	Tons per man-shift
Breaking (drilling and blasting)	0.675	11.8
Blockholing	.406	19.7
Total	1.081	7.4

The above costs were obtained during the transition from shrinkage stoping to sublevel stoping and include shrinkage-stoping costs in part of the mine where the ore body was narrow; therefore they do not indicate the full benefit derived from the change in the stoping system.

HANOVER BESSEMER IRON & COPPER CO., FIERRO, N. MEX.[31]

The ore occurs in lenticular masses dipping about 50°. As it is very irregular in iron content and distribution of impurities—chiefly sulphur, silica, and magnesia—a stoping method must be used that will permit selective mining and allow ore and waste to be taken out

[30] Labor includes surface labor in steel shop and compressor room.
[31] Kniffen, Lloyd M., Mining and Engineering Methods and Costs of the Hanover Bessemer Iron & Copper Co., Fierro, N. Mex.: Inf. Circ. 6361, Bureau of Mines, 1930, 20 pp.

separately as conditions change. The ore occurs as replacements in limestone beds at and near their contact with a granodiorite batholith. The magnetite ore bodies are very irregular in size, run more or less parallel to the granodiorite contact, and usually are lenticular. Occasionally a thickness of 200 feet is attained, but the average is about 40 feet. In many instances alternate beds, one above the other, have been replaced, leaving beds of altered rock and low-grade iron between, and four parallel beds often occur. The ore and wall rocks are strong and self-supporting over considerable areas. Levels are driven in the lower ore bed at 125- to 175-foot intervals, and from these chute raises are run at 150-foot intervals along the drifts to grizzly chambers 25 feet above the level. The grizzly chambers are connected by a manway drift, to which access is afforded by manway raises halfway between the chute raises. From the grizzly chambers inclined stope raises are opened the full width of the ore, pitching up along the ore body about 45° from the horizontal. A pilot raise also is driven straight up the dip from the grizzly chamber to the next level. A small pilot drift is driven along the ore under the 25-foot floor pillar of the level above. Starting on the pilot-drift level at the pilot raise and at the intersections of the pilot-drift level with the inclined stope raises the ore is benched down by underhand mining, the broken ore running by gravity to the grizzlies below. Occasional connections from the stope raises in the lower bed allows the ore from the stopes in the upper beds (which are mined in the same manner) to run by gravity to the grizzly chambers in the lower bed. Irregular pillars are left as required to support the hanging wall, and wherever possible the pillars are left in lean, hard, siliceous material. When stoping is nearly completed in one section of an ore body, pillars of ore can be trimmed considerably and in many cases taken out entirely. Caving eventually occurs, and the limestone between the ore beds and horses of waste fall and mix with the remains of the ore pillars. Often it is possible to draw this mixed material and recover the magnetic ore in the cobbing plant.

Costs of stoping per long ton of ore mined in 1930 were as follows:

Stoping costs, Hanover Bessemer Iron & Copper Co., 1930

	Labor	Supplies	Compressed air	Total
1. Stoping (first breaking):				
Drilling	$0.111	$0.023	$0.040	$0.174
Explosives		.064		.064
Mucking	.001			.001
Timbering	.003	.003		.006
Piping	.004	.002		.006
General	.007			.007
Total stoping (first breaking)	.126	.092	.040	.258
2. Secondary breaking:				
Blockholing	.046	.004	.006	.056
Explosives		.022		.022
Grizzly maintenance	.002	.002		.004
General	.003			.003
Total secondary breaking	.051	.028	.006	.085
Total stoping cost	.177	.120	.046	.343

Consumption of explosives per long ton in stoping is 0.355 pound for first breaking and 0.096 pound for secondary breaking—a total of 0.451 pound per ton.

SUMMARY OF OPEN STOPING

1. *Applicability.*—Open-stoping methods are applicable to the mining of small and large deposits dipping at angles from horizontal to vertical and having firm, strong ore and wall rocks that will support themselves over considerable areas while the ore is being taken out. The sublevel-stoping system is adaptable to somewhat weaker ores and wall rocks than other open-stoping systems because the men always are working under solid backs which can be readily inspected, trimmed, and kept safe.

2. *Flexibility.*—Considerable flexibility can be attained in open stoping; irregularities in the ore body can be followed, lean ore and waste inclusions can be left as pillars, and waste can be sorted and rejected from the broken ore. The ore can be removed from the stopes as it is broken; or, if desired, considerable ore often can be accumulated in the stopes to provide a reserve of broken ore.

3. *Recovery.*—Except for ore tied up in pillars complete extraction of the ore is obtained readily. Since the stope walls are exposed at all times ore is not apt to be overlooked, and irregular tongues and offshoots of ore may be taken out. The amount of ore left in pillars varies considerably with the strength of the ore and wall rocks and their action on prolonged exposure to the air and with the need for permanent support of the back and prevention of ultimate caving and subsidence. From 10 to 40 percent of the ore may be left in pillars, the higher percentages being left where it is imperative that breaks in the capping be prevented because of overlaying water-bearing strata, streams or lakes, or surface improvements. Often a large percentage of the pillar ore may be extracted before the mine is abandoned so that the final loss of ore in pillars may range from zero (in certain small ore bodies) to 10 or 15 percent. Except under special conditions, usually 80 to 95 percent of the ore ultimately can be recovered.

Development.—Development is largely in ore once the ore body has been penetrated, and large development openings can be employed without much additional cost, since with large headings the cost per ton is less than with small headings although the cost per linear foot driven may be greater. Once the ore body has been penetrated stoping can begin almost immediately without much preliminary stope preparation. (The sublevel stoping system is an exception, but the preliminary work serves to block out the ore thoroughly ahead of stoping and assists in obtaining low over-all costs.) In some mines using the open-stoping method the ore is developed largely by advancing the stope faces. If the ore is high grade some means of extracting the pillars must be employed, and this may necessitate filling the stopes and using a supported-stope method for pillar recovery.

Table 2 gives the development cost in feet of development per ton of ore developed, as well as the tons of ore developed per linear foot of stope development, at a number of mines employing open stoping.

TABLE 2.—*Development cost in feet of development per ton of ore developed, open-stope mines*

Mine	Variation of open-stope method	Stope development per ton of ore developed, feet	Ore developed per foot of stope development, tons
Burra-Burra, Ducktown, Tenn.[1]	Sublevel stoping	0.0091	110
Isabella, Isabella, Tenn.[2]	Room and pillar; underhand mining	.0050	200
Menominee range, mine A.[3]	Sublevel stoping	.030	33
Menominee range, mine B.[4]	do	.018	55
Menominee range, mine C[5]	do	.022	46
Menominee range, mine D[6]	do	.050	20
Marquette range, mine E[7]	Overhand working, regular pillar support; stopes 22 feet wide.	.0167	60
Gold mine, Ontario[8]	Sublevel stoping	.008	125
Horne, Quebec[9]	Inclined stopes benched as in sublevel stoping	.013	75

[1] Hard, firm ore and walls; sublevel interval, 40 feet.
[2] Hard, firm ore and walls; pillars left for permanent support.
[3] Firm, strong, and moderately hard ore; walls moderately strong; sublevel interval, 25 feet.
[4] Ore and walls moderately strong; sublevel interval, 25 feet.
[5] Ore moderately strong and hard, walls strong but spall somewhat upon exposure; sublevel interval, 25 feet.
[6] Ore soft and not very strong; stope face slopes outward; sublevel interval, 25 feet.
[7] Ore and walls firm, ore moderately hard.
[8] Ore and walls hard and strong; sublevel interval, 33 feet.
[9] Ore and walls hard and strong; interval between inclines, 30 feet center to center.

5. *Cost of open stoping.*—The cost of stoping generally is comparatively low in open-stope mining. Although the ore often is hard holes usually can be drilled to break to 2 and sometimes 3 free faces for blasting much of the ore. In deposits dipping at angles steeper than the angle of repose the ore runs by gravity to the haulage-level chutes, and in flatter deposits power scrapers usually can be employed to good advantage, so that little hand shoveling is required.

Direct stoping costs per ton mined are given in table 3 for a number of mines using open-stoping methods.

Table 4 shows the productivity of stope labor in man-hours per ton. Table 5 shows consumption of explosives in open-stope mines;

TABLE 3.—*Typical stoping costs at open-stope mines*

Mine	Year	Variation of open-stope method	Direct stoping costs per ton of ore hoisted: Labor	Supervision	Air compression, drills, and steel	Power	Explosives	Timber	Other supplies	Total
Average of 7 mines, Tri-State district.	1927–30	Irregular pillars; flat beds	$0.273	$0.020	$0.102	$0.005	$0.104			$0.604
No. 8, Southeast Missouri	1928	do	.383	.030	.080	.010	.072		$0.033	.588
Mascot, Tenn	1920	Mill-hole system; irregular pillars	.130		.042		.045		.005	.232
Marquette range, no. 1 hard-ore mine.[1]	1928	Regular room-and-pillar	.614	.028	.104	.006	.105	$0.002	.011	.770
Mineville, N. Y.[1]	1927	Irregular pillars	.181	.024	.113		.122		.003	.416
Osceola, Mich	1931	Wide stopes; irregular pillars								[2] .450
Ahmeek	1931	Long narrow stopes; regular narrow pillars.								[2] .690
Vanadium, Rifle, Colo	1931	Room-and-pillar; pillars extracted on retreat.	.740	.050	.090	.130	.200	.040	.100	1.350
Granada, Quebec	July 1932	Overhand mining; no pillars except in lean ore and waste.	1.550	.057	0.476		.606	.040		[3] 2.766
Mary, Isabella, Tenn	1928	Underhand mining; regular stope and pillar.	.509	.022	.177		.076			.784
Burra-Burra, Ducktown, Tenn	1928	Sublevel stoping; pillar mining, and some shrinkage.	.196		.051		.083		.002	.332
Do	1932	Sublevel stoping	.100		.081		.078			.260
No. 1, Menominee range,[1] Mich	1929	Sublevel stoping, vertical stope face	.246	.051	.060		.090	.009	.011	.467
No. 2, Marquette range[1]	1928	Sublevel stoping; outward sloping stope face.	.160	.026	.030	.047	.050	.010	.075	.417
Spring Hill, Mont	Aug. 1, 1929, to Apr. 30, 1930.	Sublevel stoping in wide ore; shrinkage stoping in narrow ore.	.718		.124		.178		.032	1.072
Sherritt Gordon[4]	1931–32	Underhand benching similar to sublevel stoping on dips above 46°, but benches 150 feet high measured on dip.	.298		.150		.210		.089	.747
Do	1932	Overhand, using scrapers on dips under 46°.	.302		.161 [5] .054		.146		.074	.740
Hanover Bessemer, Fierro, N. Mex	1930	Underhand benching to inclined stope raises.	.167	.010	.075		.080	.003	.002	.343

[1] Costs per long ton of 2,240 pounds.
[2] Breaking cost.
[3] One-third of broken rock sorted and rejected as waste in mine. Total direct stoping cost per ton broken was $1.943.
[4] Brown, Eldon L., Mining Methods and Costs at the Sherritt Gordon Mine: Canadian Inst. Min. and Met. Bull. 257, September 1933, pp. 468–494.
[5] Scraping.

TABLE 4.—*Productivity of stope labor, open-stope mines*

Mine	Year	Variation of open-stoping method	Man-hours per ton of ore hoisted				Tons per man-shift
			Breaking	Hand and power loading in stopes	Other stope labor	Total	
Average of 7 mines, Tri-State district	1927–30	Irregular pillars; flat beds	0.252	0.226	Hand tramming, 0.076	0.554	14.4
No. 8, Southeast Missouri	1928	do	.122	.319		.441	18.1
Another mine, Southeast Missouri	1929	do	.162	.233	Roof trimmers, 0.034	.429	18.6
Mascot	1928	Mill-hole system; irregular pillars	.214	.066	Timbering, 0.036	.316	28.5
Marquette range, no. 1 [1]	1929	Regular room-and-pillar	.342	.310	Timbering and filling, 0.051	.703	11.4
Mineville, N. Y. [1]	1927	Irregular pillars	.230	.480		.710	11.2
Edwards	1930	Irregular pillars in stronger ground; regular pillars in weaker ground.	.456	.304	Timbering, 0.072	.832	9.6
Vanadium, Rifle, Colo.	3 months, 1931	Room-and-pillar; pillars extracted on retreat	.300	.900	Timbering and filling, 0.240	1.440	5.6
Granada	July 1932	Overhand mining; no pillars except in lean ore and waste.	.809	.249		1.058	7.6
Mary	1928	Regular stope-and-pillar; underhand mining	.556	.467		1.023	7.8
Burra-Burra	1928	Sublevel stoping, pillar mining, and some shrinkage stoping.	.349	.258		.609	13.2
No. 1, Menominee range [1]	1928	Sublevel stoping; vertical stope face	.353	.011	Timbering, 0.038	.402	19.9
Spring Hill	Aug. 1, 1929 to Apr. 30, 1930.	Sublevel stoping in wide ore; shrinkage stoping in narrow ore.				1.081	7.4
Iron mine A [1]	1928	Sublevel stoping	.200			.200	40.0
Iron mine B [1]	1928	do	.180			.180	44.4
Iron mine C [1]	1928	do	.140			.140	57.1
Iron mine D [1]	1928	do	.140			.140	57.1
Montreal [1]	1928	do	.192	.024		.216	37.0

[1] Based on long ton of 2,240 pounds.

TABLE 5.—*Consumption of explosives in open stopes*

Mine	Variation of open-stoping method	Kind and grade of explosive used	Consumption of explosives per ton of ore broken, pounds
Tri-State No. 1	Irregular pillars	(?)	0.750
Tri-State No. 2	do	Ammonia, 33 percent	1.265
Tri-State No. 3	do	Gelatin dynamite, 20 percent	.875
Tri-State No. 4	do	Gelatin dynamite, 30 percent	.594
Tri-State No. 5	do	Ammonia, 40 percent	.805
Tri-State No. 6	do	Gelatin dynamite, 20 percent	.741
Tri-State No. 7	do	Ammonia, 40 percent	1.120
No. 8, Southeast Missouri.	do	75 percent gelatin, 35-percent strength, 25 percent bulk dynamite.	.565
Another mine, Southeast Missouri.	do	Ammonia, 40 percent	.459
Mascot	Mill-hole system with irregular pillars.	Gelatin dynamite, 30 percent	.602
Marquette No. 1	Regular stope-and-pillar	Ammonia, 50 percent	1.705
Marquette No. 2	Sublevel stoping	Ammonia, 60 percent	1.244
Menominee No. 1	do	Ammonia, 40 percent	1.619
Iron mine A	do	Gelatin dynamite, 40 percent	1.640
Iron mine B	do	do	1.074
Iron mine D	do	do	1.600
Mineville	Irregular pillars	75 percent gelatin dynamite, 40-percent strength; 25 percent gelatin dynamite, 25-percent strength.	1.790
Vanadium	Room-and-pillar; pillars mined on retreat.	Special bulk dynamite	1.050
Granada	Overhand mining; no pillars except in lean ore or waste.	Gelatin dynamite, 40 percent	3.930
Mary	Underhand mining; regular stope-and-pillar.	do	.420
Burra-Burra	Sublevel stoping	Ammonia, 40 percent	1.755
Hanover Bessemer	Underhand benching to inclined stope raises.	Gelatin dynamite, 30 percent	1.461
Montreal	Sublevel stoping	Ammonia, 35 and 60 percent	1.490

[1] Per long ton of 2,240 pounds.
[2] Includes some development, shrinkage stoping, and pillar robbing.

SHRINKAGE STOPING

Shrinkage stoping in its simplest form is employed for mining tabular deposits of firm, strong ore ranging from 3 or 4 feet in thickness to the maximum thickness over which the back of the stopes will stand unsupported, enclosed between strong walls and dipping at angles of 50° to 90°. It is also used for mining deposits too wide to stand unsupported, in which case the stopes are run at right angles to the strike, are of a width which will stand safely, and are separated by transverse pillars of ore. These pillars often can be mined economically later, but usually the stopes must be filled first, and a different method of stoping may have to be applied for mining the pillars. Modifications of shrinkage stoping are used for mining large masses of low-grade ore. In modifications such as those employed at the Alaska Juneau mine [32] and the Beatson mine [33] the stope backs are broken down by large blasts. These methods are sometimes referred to as caving methods, but, as the stopes are kept

[32] Bradley, P. R., Mining Methods and Costs, Alaska Juneau Gold Mining Co., Juneau, Alaska: Inf. Circ. 6186, Bureau of Mines, 1929, 18 pp.
[33] Presley, Bevan, The Latouche System of Mining as Developed at the Beatson Mine, Kennecott Copper Corporation, Latouche, Alaska: Am. Inst. Min. and Met. Eng., Pub. 20, 1927, 42 pp.

nearly full of broken ore and such caving as occurs is forced by blasting, the authors have classed them as variations of the shrinkage method. Shrinkage stoping also is employed in combination with other methods. Thus boundary shrinkage stopes often are used to cut off caving blocks in block caving. In some instances first mining is done in shrinkage stopes between pillars, whereas the pillars may be extracted by caving during final drawing of the shrinkage stopes, by square-setting, or by top-slicing.

ORDINARY SHRINKAGE STOPING IN NARROW TO MODERATELY WIDE DEPOSITS

Narrow to moderately wide, tabular, steeply dipping deposits afford the simplest conditions for shrinkage stoping. Development consists of a single drift in ore on each level, usually along the foot-

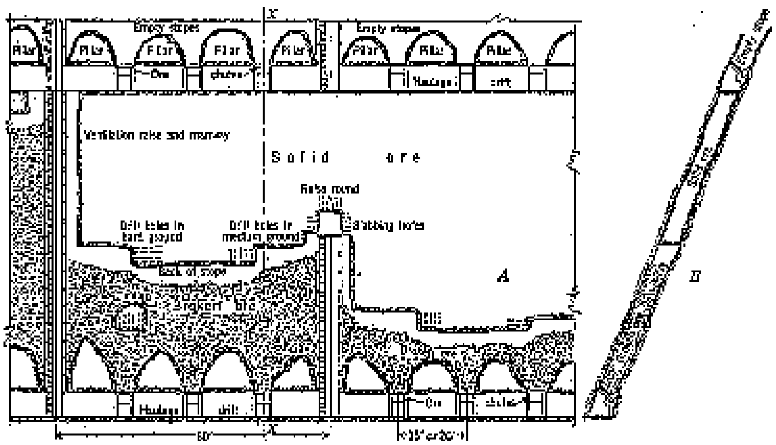


FIGURE 28.—Shrinkage stoping on drift pillars, Nevada-Massachusetts mine, Nevada: *A*, Vertical longitudinal projection; *B*, vertical cross section *X–X*.

wall of the ore body, with chutes at intervals of 10 to 20 feet for drawing off and loading the broken ore. (In fig. 5 a drift is shown timbered with regular sets, which are lagged over to hold the broken ore in the stope.) In narrow veins stull timbers may be used instead of regular drift sets. Sometimes the drift is untimbered, and the stope is begun 10 to 20 or more feet above the back of the drift, in which case chute raises are put up at regular intervals along the drift and are connected by a stope drift, the bottom of which forms the bottom of the stope, leaving arch pillars over the drift for supporting the broken ore in the stope. (See fig. 28.) If the walls are good these pillars can be recovered later. In any case the stopes are advanced upward by horizontal or inclined slices the full width of the vein, which may be drilled with the flat breast holes as shown in figure 5 or with holes drilled upward in the back by means of stopers. The miners work on top of the ore, and as the stope advances enough of the broken ore is drawn off after each blast to provide working room between the top of the pile and the back of the stope.

NEVADA-MASSACHUSETTS MINE, MILL CITY, NEV.[34]

The ore is a tungsten ore carrying 0.5 percent of tungsten trioxide; it occurs in limestone beds at their contact with granite, also at a considerable distance from the granite. The ore is mined from two limestone beds, the most productive of which averages 4½ feet in width and dips at an angle of 75°. The bed has been displaced 3 to 60 feet in places by postmineral faults. The walls stand well with a moderate amount of support, and the ore breaks easily. It is stoped in sections 75 or 80 feet long by longitudinal shrinkage stopes carried on arch pillars over the backs of the haulage-level drifts. (See fig. 28.)

To prepare a block of ore for mining a ventilation raise is put through to the level above at a point as far from the shaft as possible. While this is being driven chute raises are put up at 15- to 20-foot intervals, and chutes are installed. A manway for access to the stope is begun 75 or 80 feet from the ventilation raise. If the ore is damp and apt to hang up in the stope, chute raises are 15 feet apart; if the ore is dry and free-running, 20-foot spacing is employed. After the ventilation raise is completed, the chutes installed, the tops of the chute raises belled out to connect with each other, and the manway started, stoping operations are begun.

The stope is advanced as a single face by horizontal slices drilled with Leyner machines if the ground is hard, or by vertical slices with hand-rotated stopers if the ground is comparatively soft. Two miners are employed in each 75-foot block; they advance toward each other, one starting at the ventilation raise and the other at the manway raise at the opposite end of the block. Enough broken ore is drawn during stoping to leave 6 feet of headroom between the broken ore and the back. The stope is carried within 6 feet of the level above, leaving a floor pillar this thick under the level, which is blasted down into the stope before the level is abandoned. When loose spots of hanging wall are encountered 8-inch stulls with 3- by 12- by 18-inch headboards are installed to prevent dilution of the ore. In final drawing of the stope the top of the broken ore is followed down by timbermen who catch up slabs of loose hanging wall and trim down any ore left on the walls.

Direct stoping costs for 1928, when 40,924 tons of ore were hoisted, were as follows:

Direct stoping costs, Nevada-Massachusetts mine, 1928

	Cost per ton		Cost per ton
Labor	$1.662	Timber	$0.146
Supervision	.064	Other supplies	.207
Drills and steel	.100		
Power (air compression)	.209	Total	2.743
Explosives	.355		

Miners and timbermen were paid $5.25, muckers $4.75, blacksmiths $6.50, and hoistmen $6.00 and $5.75 per shift.

Productivity of stope labor, as measured in man-hours per ton, was as follows:

[34] Heizer, Ott F., Method and Cost of Mining Tungsten Ore at the Nevada-Massachusetts Co. Mines, Mill City, Nev.: Inf. Circ. 6284, Bureau of Mines, 1930, 13 pp.

Productivity of stope labor

Occupation:	Man-hours per ton	Tons per man-shift
Breaking	0.700	11.4
Timbering	.376	21.3
Subtotal	1.076	7.4
Shoveling and tramming	.383	21.3
Total	1.459	5.5

Consumption of explosives was 1.650 pounds of 35-percent gelatin dynamite per ton mined.

HARMONY MINE, BAKER, IDAHO[35]

The ore occurs in sheeted zones, in thick beds of highly siliceous shale, as regular shoots or pipes of primary chalcopyrite which is partly altered to bornite and chalcocite in the upper levels. The ore shoots are included within the fractured zones and are uniform in their principal characteristics. The main ore shoots average 5 feet in thickness and 80 feet in length and dip 60° to 65°. The vein

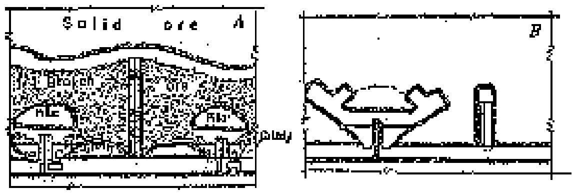


FIGURE 29.—Shrinkage stoping, Harmony mine, Baker, Idaho: *A*, General vertical longitudinal projection; *B*, method of cutting around grizzly pillars.

filling and ore are difficult to break, but planes of weakness in the vein parallel to the walls assist in breaking. Both walls are hard shale, which stands well and shows no appreciable alteration after long exposure. In 1929 about 150 tons of ore containing 4 percent of copper were being mined per day by shrinkage stoping.

After a haulage drift has been driven to the limits of an ore body a stope is developed by installing double chutes on 40-foot centers and manways at 120-foot intervals. If the stope is less than 80 feet long only one manway is necessary. Raises for the chutes are 5 by 12 feet in section and are driven 18 feet above the back of the drift; chutes are then put in, the chute raises belled out as shown in figure 29, and grizzlies installed. A 4- by 5-foot raise is driven to the level above at the far end of the ore body for ventilation and as a starting point for stoping rounds. From the ends of the belled-out chutes raises are driven at an outward inclination of 40°, and about 12 feet above the grizzlies flat raises are driven from the backs of the 40° raises to meet each other, thus cutting over a pillar which is left above the grizzlies to protect the grizzly men. (See fig. 29, *B*.) Flat raises also are driven from adjoining chutes to meet each other.

Stoping proceeds along the entire stope as soon as the work around the pillars has been completed. The back of the stope is drilled with automatically rotated wet stopers. As breaking pro-

[35] Gardner, E. Duncan, Mining Practice at Harmony Mines Co., Baker, Idaho: Inf. Circ. 6240, Bureau of Mines, 1930, 8 pp.

ceeds enough ore is pulled through the grizzlies to keep the top of the broken-ore pile a convenient distance below the back of the stope. About 40 percent of the ore is drawn off during stoping. It is continued beyond the level above until all pillars on that level are taken out.

When a chute is being pulled one man is required on the grizzlies to work the ore through and bulldoze any boulders that cannot be broken with a hammer.

Direct stoping costs from January 5 to June 5, 1929, when 19,000 tons were extracted, were as follows:

Direct stoping costs, Harmony mine

	Cost per ton		Cost per ton
Labor	$0.65	Explosives	$0.27
Supervision	.30	Other supplies	.22
Air, drills, and steel	.27		
Power	.16	Total	2.05
Timber	.18		

Productivity of stope labor, as measured in man-hours per ton, was as follows:

Productivity of stope labor, Harmony mine

Occupation:	Man-hours per ton	Tons per man-shift
Breaking (drilling and blasting)	1.19	6.7
Shoveling	.06	133.4
Total	1.25	6.4

Consumption of explosives was 2.05 pounds per ton of ore.

ROSICLARE, ILL.

Shrinkage stoping is employed for mining fluorspar at mines near Rosiclare, Ill. The fluorspar is found principally in fault fissures, wholly or partly replacing calcite.

At one mine[59] the vein attains a maximum width of 34 feet; widths of 18 to 20 feet are common, and the average width is about 12 feet. The length of the ore shoot is 1,900 feet, the ore being virtually continuous for this distance. The dip is almost vertical. Both walls are limestone, except near the surface, where to a depth of 170 feet the hanging wall is a sandstone underlain by a bed of shale. This upper section was mined by square-set stoping and the rest by shrinkage stoping. The limestone walls are firm and will stand for a long time after the stopes have been mined out and drawn empty.

The vein is developed by haulage drifts spaced 100 feet apart vertically. In wide ore raises are put up on 25-foot centers along the drift and connected by a stope drift above the back of the level so as to leave an arch pillar as thick as the width of the vein. In narrow ore the drift is timbered with stull sets upon which the broken ore in the stopes is supported, and chutes are installed in every other set, the sets being on 6-foot centers.

[59] Reeder, Edwin C., Methods and Cost of Mining Fluorspar at Rosiclare, Ill.: Inf. Circ. 6294, Bureau of Mines, 1930, 10 pp.

Individual stopes range in length from 100 to several hundred feet. The back of the stope is carried as an inclined stepped face; thus the top of the broken-ore pile is inclined and slopes from the bottom of the stope at one end to the highest face at the other end. The top of the ore pile is kept 6 or 7 feet below the back of the stope. The backs are drilled either with uppers, using stoper drills, or with flat holes, using light hammer drills. Flat holes are preferred and used wherever possible. Frequently they can be drilled 12 or 15 feet deep and in wide ore are especially effective, producing less fines than the vertical upper holes and breaking more ore per foot of hole drilled. Stopes are carried through from level to level without permanent level pillars being left.

Direct stoping costs during 1929, when 51,208 tons were mined (of which 25 percent was stoped by square-setting and the balance by shrinkage stoping), were as follows:

Direct stoping costs, Hillside Fluorspar mine, 1929

	Cost per ton		Cost per ton
Labor	$0.700	Explosives	$0.219
Supervision	.020	Timber	.082
Compressed air, drills, and steel	.115	Other supplies	.053
Power	.087	Total	1.276

At the Daisy mine of the Rosiclare Lead & Fluorspar Mining Co.[57] there are two veins, known respectively as the "Daisy" and the "Blue Diggings" veins. The Daisy vein dips west about 70°, and the Blue Diggings vein dips east about 70° near the surface, flattening to 35° at a depth of 640 feet. The two veins intersect a short distance below the 640 level. The ore occurs in lenses or pockets which vary in length and are 2 to 30 feet wide. They are separated laterally and vertically by pinches where there is little mineralization. The hanging wall usually is heavy, and large slabs of calcite, limestone, or shale slough away unless props are used. Most of the upper portions of the vein are badly weathered, making timber support necessary in stoping. The shrinkage-stoping method is employed below the weathered parts of the veins.

The veins are developed by levels at depths of 180, 300, 412, 537, and 640 feet. The drifts are driven the full width of the ore, and stopes are begun by taking a stope slice out of the back, bringing the back about 12 feet above the floor of the drift. From the top of the muck pile a second round is drilled in the back but is not blasted until later. Stull timbers are placed every 5 feet at drift height and lagged over with poles. Chute openings are left every 10 feet, and chutes are installed therein. Raises are driven to the level above near each end of the ore shoots for ventilation. Starting at one end of a stope the ore is shot down on the poles and into the chutes, and regular shrinkage stoping then is carried to the level above. The stope back is carried on a slope, step fashion, giving several working faces. Light, hand-rotated stopers are used, and for average stope widths three holes are drilled across the back to a depth of 5 feet in rows 30 inches

[57] Cronk, A. H., Mining Methods of the Rosiclare Lead & Fluorspar Mining Co., Rosiclare, Ill.: Inf. Circ. 6381, Bureau of Mines, 1930, 13 pp.

apart. Loose slabs of wall rock are either shot down or held by stulls, depending upon the character of the wall.

During 1929, when 42,000 tons of ore were mined, direct stoping costs were as follows:

Direct stoping costs, Daisy mine, 1929

	Cost per ton		Cost per ton
Labor	$0.710	Explosives	$0.160
Supervision	.065	Timber	.120
Compressed air (including power), drills, and steel	.190	Other supplies	.070
		Total	1.315

CONSOLIDATED CORTEZ SILVER MINE, CORTEZ, NEV.

Three types of ore deposits occur at this mine: (1) Dike ore, in and immediately adjoining dikes; (2) bedding-plane ore, along bedding planes in limestone; and (3) fissure ore, in fissures approximately parallel to the dikes.[36] The third type has furnished most of the ore. In general, the formation is a series of limestone and quartzite beds, uplifted by an intrusion of granodiorite and later cut by a series of dikes. The fissure ore bodies range from 1 to 20 feet in width, averaging about 7 feet. The maximum length of the ore bodies is 600 feet, and the dip ranges from 45° to 80°. The walls are poorly defined, being economic rather than structural. The walls of the fissures are solid, and very little support is required in the stopes, so that the ore is well-adapted to shrinkage stoping. In places the ore breaks in large boulders which are broken in the stopes with hammers or drilled with jackhammers and blasted. The valuable minerals in the unoxidized zone are silver-bearing quartz, galena, pyrite, tetrahedrite, sphalerite, and chalcopyrite.

Levels are about 100 feet apart vertically, and the drifts are driven 6 by 7½ feet in cross section. At this mine the practice is to stope on arch pillars over the drifts. Before a stope is begun the length of the ore shoot is determined by drifting, and the lay-out for the stope is planned accordingly. Where the ore shoot is long enough, the stope is laid out in sections with manway raises at intervals of 100 or 125 feet. Manway raises first are put up to a height of 25 feet above the track and are timbered with stulls. While these raises are being timbered chute raises are installed on 24-foot centers between the manway raises to a height of 25 feet, and chutes are installed. The raises are connected by a subdrift, leaving an 8-foot pillar over the haulage drift, and the manway raises are advanced another 25 feet, completing the stope preparation. In stoping, horizontal rounds or slices are blasted; starting at each manway raise the slices are advanced toward each other to meet in the center of the stope. This procedure is repeated, enough ore being drawn off after each cut to maintain working room between the broken ore and the back of the stope. Stoping is carried within 15 feet of the level above, a pillar this thick being left to support the level. The manways are advanced from time to time to keep them ahead of the stoping. Occasional pillars of low-grade ore are left to support

[36] Hezzelwood, George W., Mining Methods and Costs at the Consolidated Cortez Silver Mine, Cortez, Nev.: Inf. Circ. 6327, Bureau of Mines, 1930, 15 pp.

the hanging wall. Pillars of good ore sometimes are left for temporary support, but these are blasted when the ore is being drawn from the completed stope. Drilling is done with Leyner machines mounted on a crossbar where the width permits, or on an arm attached to a column. Ninety-five percent of the ore is recovered by shrinkage stoping; the balance is left as arch and floor pillars. These may be recovered eventually when the levels are abandoned.

During 1929, 43,806 tons of ore were mined, and direct stoping costs were as follows:

Direct stoping costs, Cortez mine, 1929

	Cost per ton		Cost per ton
Labor	$1.082	Timber	$0.054
Supervision	.086	Other supplies	.126
Compressed air, drills, and steel	.250	Total	1.902
Explosives	.304		

Wage rates were: Machinemen $5.75, muckers $5.25, pipe and trackmen $5.75, blacksmiths $7, blacksmith helpers $5.75, and timbermen $5.75.

Stope-labor performance for the same year was as follows:

Performance of stope labor, Cortez mine, 1929

Occupation:	Man-hours per ton	Tons per man-shift
Breaking	0.901	8.9
Timbering	.209	38.3
Shoveling	.475	16.8
Total	1.585	5.0

Consumption of explosives in stopes was 2.03 pounds of 40-percent gelatin dynamite per ton of ore mined.

EIGHTY-FIVE MINE, VALEDON, N. MEX.

The stoping methods at this mine have been described by Youtz.[89] The ore occurs in veins in igneous rocks, the most important being the Esmeralda, which is in a stock of monozonite. This vein dips an average of 80° to the southeast and can be traced on the surface for 5,000 feet. Ore has been mined for 2,300 feet along the strike and, on the 1,500-foot level, ore occurs 2,600 feet along the vein. The vein is very regular in its course, ranging from 2 to 10 feet in width and averaging about 5 feet. In places, especially in the upper levels, it attains a width of 30 feet. The vein filling is massive quartz, country rock replaced by sulphides, and silicified, altered, quartz-seamed rock containing stringers of chalcopyrite and pyrite. The vein material is hard and stands without support over wide spans. On rare occasions the hanging wall slacks and sloughs off to such an extent as to weaken the stope back, which then may break down in large blocks. The ore is hard to drill but breaks fine so that secondary blasting seldom is necessary. Below the 1,350-foot level, the walls are mineralized to a greater extent than above and frequently they are thoroughly

[89] Youtz, Ralph B., Mining Methods at the Eighty-Five Mine, Calumet & Arizona Mining Co., Valedon, N. Mex.: Inf. Circ. 6415, Bureau of Mines, 1931, 20 pp.

kaolinized. In some places this makes it necessary to employ cut-and-fill instead of shrinkage stoping.

The upper levels are 100 feet apart and the lower levels 150 feet apart vertically. From 1920 to 1930 the following stoping methods have been used: Shrinkage stoping, 72 percent of the total; cut-and-fill stoping, 28 percent. Of the stopes mined by cut-and-fill stoping, 64 percent were mined in the period 1920–23. In 1931 both shrinkage and cut-and-fill stoping were employed. Shrinkage stoping always was used where the walls were firm and the vein was more than 3 feet wide, as well as in marginal ore of high silica content. Cut-and-fill stoping sometimes was used, however, where (1) the walls were weak owing to a system of fractures closely paralleling either wall or to kaolinization of the walls, (2) the vein was wide but unmineralized country rock occurred as a wide band within the enclosing walls or splits occurred in the vein, (3) high-grade ore occurred in a narrow (2 or 3 feet) vein, and (4) a cleaner product was desired.

The advancing system of shrinkage stoping (beginning at the end of the ore body nearest the shaft) is employed at the Eighty-Five mine. Usually the stopes are carried on stull timbers over the back of the level drift, but about 15 percent of the stopes have been carried on arch pillars. Arch pillars are used only if the walls are weak and the vein is unusually wide, or if the back of the drift is in waste or lean ore but development has shown ore to be in the block higher up. Where pillars have been left over the drifts 60 percent were in waste or lean material and 40 percent in ore. Of the 40 percent in ore 20 percent had already been mined in 1931, and 50 percent was still recoverable.

When a stope is begun a raise is put through first to the level above, and, beginning at the raise, the back of the drift is drilled and shot down on the tracks. With the miners standing on top of the muck, a second line of holes is drilled and blasted, and if the ore is wider than drift width the drift is slabbed off the full width of the vein. Stull timbers are placed across the drift at 5-foot intervals 8 feet above the track. Where the vein is 4 to 8 feet wide no posts are used under the stulls other than chute posts. Chutes are built on the footwall side at 10-foot intervals. Back stoping is begun by a crew of 1 to 4 miners, drilling with hand-rotated stopers. The back of the stope is carried at a slight inclination upward toward the through-raise, the angle ranging from 10° to 15° from the horizontal. At 100-foot intervals cribbed manways are carried up with the stope. These manways are protected on the stope side by stulls set in hitches in the walls at 5- to 8- or 10-foot intervals. Usually the stopes are mined to within 5 to 15 feet of the level above, leaving a floor pillar to support the level. When the shrinkage stopes have been completed and are being drawn empty, the hanging wall begins to slough off; consequently, by the time the stope is nearly empty considerable waste has fallen on top of the broken ore. Care therefore must be taken to draw the stopes down evenly to prevent piping through of waste.

The cost of shrinkage stoping by this method on the 1,350-foot level was as follows during 1925:

Cost of shrinkage stoping, 1,350-foot level, Eighty-Five mine, 1925

Stope no.	Labor (including bonus)	Air drills	Explosives	Timber	Other supplies	Total cost per ton
8	$0.59	$0.39	$0.23	$0.14	$0.02	$1.37
6	.75	.46	.27	.20	.02	1.70

The average width of no. 8 stope was 8.5 feet and that of no. 6 stope 4.0 feet.

Direct stoping costs in typical shrinkage stopes from 1927 to 1929 were as follows:

Direct stoping costs in typical shrinkage stopes, Eighty-Five mine

Stope no.	Year	Costs per ton mined					
		Labor (including bonus)	Air drills	Explosives	Timber	Other supplies	Total
26	1928–29	$0.72	$0.64	$0.31	$0.07	$0.01	$1.75
5	1927	.49	.39	.24	.10	.01	1.23

Wage rates during these years were as follows: Miners using Leyner machines, $4.02 to $4.60; miners using stopers, $3.30 to $3.80; timbermen, $3.52 to $4.05; and muckers, $2.86 to $3.30.

VERDE CENTRAL MINE, JEROME, ARIZ.[40]

Almost all the workings at the Verde Central are in shear zones in the greenstone on the contact of a quartz-porphyry batholith with greenstone. The most important ore zones are the Rock Butte fracture and the Silver Cliff contact vein. The Rock Butte fracture zone is a mineralized shear zone more than 1,000 feet long, ranging from 5 to more than 100 feet in width and extending from the 600- to below the 1,900-foot level. Ore bodies in which chalcopyrite is present in commercial quantities occur as lenses in the larger mineralized zone. They range from 5 to 40 feet in width but average about 15 feet and are 50 to 300 feet long. The ore bodies are nearly vertical.

Both the greenstone and the quartz porphyry are highly siliceous, hard, and firm and permit openings to be driven without timbering. The walls of the ore bodies stand well except near faults, and the ore is fractured little. At several places faulting cuts the ore lenses at oblique angles, making the walls heavy and creating a tendency for blocks to slide on the fault planes as the ore is mined from the vein. Vertical andesite dikes 1 to 6 feet wide cut across the ore bodies almost at right angles. The dike rock is very soft and crumbles, but the only trouble it causes is dilution of the ore. The

[40] Dickson, Robert H., Mining Methods and Costs of Mining Copper Ore at The Verde Central Mines, Inc., Jerome, Ariz.: Inf. Circ. 6464, Bureau of Mines, 1931, 13 pp.

ore is not always as wide as the mineralization. In places the ore is on one side of the mineralized fracture, in other places it covers the entire width, and in still other places it occurs as two lenses with a horse of waste between. The grade of the ore mined has averaged 2.7 percent of copper and 0.4 ounce of silver per ton.

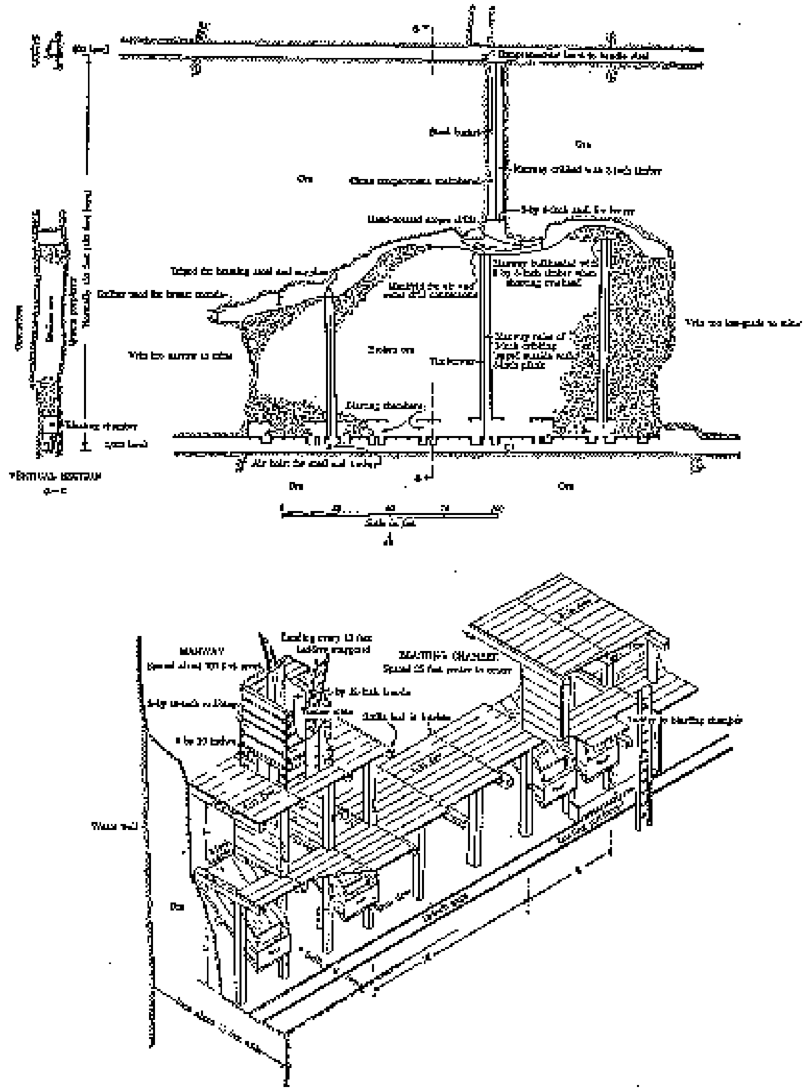


FIGURE 30.—Shrinkage stoping at Verde Central mine, Arizona; *A*, Longitudinal section through 10-4 stope; *B*, timbering at bottom of shrinkage stopes.

Levels are about 150 feet apart, and before stoping is begun they are fully developed and raises run to determine the vertical extent of the ore shoots. Double chutes are installed at 25-foot intervals with a blasting chamber above each chute which is constructed by putting in three 10- by 10-inch stulls 7 feet above the chutes. (See fig. 30, *A*.) These stulls usually are less than 10 feet in length and are lagged over and supported in the middle by 8- by 8-inch posts which limit the width of opening through which the ore is drawn from the shrinkage stope to 4 feet. (See fig. 30, *B*.) The stopes are carried on lagged 10- by 10-inch stull timbers placed over the back of the drift with 8- by 8-inch posts under them to limit the span to 5 feet. To make room for chutes and stull timbers the back of the drift is shot down the full length of the stope. Cribbed manways are carried up through the broken ore at 100-foot intervals as the stope advances upward.

Stoping is begun at one end of the stope and progresses in benches over the whole length. The intention is to keep the back of the stope nearly level until it nears the top, when it is inclined to improve ventilation. Lean spots in the vein are left as pillars.

The ore is hard, and 145-pound drifters are used for breastwork and heavy hand-rotated stopers for drilling the backs. The drifters have been used for drilling the back, but the tendency is for flat holes to break the ore in large chunks. Stoper machines break the ore finer and give more time for drilling, as they do not have to be set up.

A floor pillar a little thicker than the width of the stope is left to support the walls and upper level during final drawing of the stope. If the grade of ore is high enough the stope is filled with development waste, and the pillar is then mined out. When a stope has been completed the broken ore is drawn off as rapidly as it can be trammed. The top of the broken-ore pile is kept as nearly level as possible during drawing to prevent any waste that may fall from the walls from mixing with the ore. Broken ore often hangs up on pillars, and when this occurs it is dislodged by bombs consisting of several sticks of powder tied to a pole.

The dilution from wall rock, barren areas, and andesite dikes is estimated to be about 10 percent.

Direct stoping costs, including stope development after stoping is begun, such as raising from stopes to levels above, from June 1, 1929 to August 1, 1930, were as follows:

Direct stoping cost, Verde Central mine, June 1, 1929 to Aug. 1, 1930

[Tons hoisted, 130,203]

	Cost per ton
Labor	$0.734
Supervision	.073
Compressed air, drills, and steel	.485
Explosives	.234
Timber	.135
Total	1.661

Wages were as follows: Timbermen $6.27, miners $5.69, muckers $5.06, blacksmiths $6.27, and steel sharpeners $6.27.

Labor performance in stopes was as follows:

Labor performance in stopes, Verde Central mine

Occupation:	*Man-hours per ton*	*Tons per man-shift*
Breaking	0.72	11.1
Timbering	.33	24.0
Total	1.05	7.6

Consumption of explosives was 1.16 pounds of 40-percent gelatin dynamite per ton of ore.

TECK-HUGHES MINE, KIRKLAND LAKE, ONTARIO[a]

The ore bodies at the Teck-Hughes mine occur as irregular, lenticular veins in a steeply dipping fault zone which reaches 80 feet in width. The rocks are syenite, syenite porphyry, and lamprophyre cut by diabase dikes. Most of the ore bodies are elliptical, and the vertical dimension usually exceeds the horizontal length. Ore bodies large enough to mine range from 30 to 800 feet in length and from 100 to 1,000 feet or more in height. The width ranges from a few inches to 60 feet. Some narrow veins lie as flat as 45°, but the main ore bodies dip from 75° to vertical. The fault zone pinches at regular vertical intervals of 1,000 feet, allowing the hanging wall a bearing on the foot wall. Wall rocks show evidences of being under strain in some parts of the mine, but this condition is not general. All rocks in the faulted zone are hard, badly fractured, and cut by fault planes or "mud slips." Where the limits of an ore body do not reach the boundaries of the fault zone the wall rocks, which are no stronger than the ore, slough considerably. If a wall of the fault zone forms a wall of the ore body, it stands fairly well. The limits of the ore usually are commercial and have to be determined by careful sampling. The ore is high grade and averages about $20 per ton, but the ore as mined often is considerably lower grade owing to dilution in stoping and drawing. In 1930 all the ore was being mined by shrinkage stoping, 90 percent being mined with chute raises or boxhole pillars (that is, on arch pillars over the drifts) and 10 percent with timbered drift backs.

Chute raises and boxhole pillars are employed where the ore is wide and dips 60° or more and a stope from below is sure to break through underneath the pillar. The back of the drift is timbered, and the broken ore is held on lagging where the vein is less than 6 feet in width or the dip is too flat for the ore to run well, or at the bottom of an ore body where boxhole pillars could not be caved into a stope below.

After a level is developed the drift is slashed to a width of 8½ feet and cleaned out. Boxhole raises are then carried up three rounds, a loading and bulldozing chamber is cut (fig. 31, *A*), and a platform or "Chinaman" chute built. At intervals of about 120 feet a small, timbered drift stope is begun by taking down back for 30 feet along the vein and is timbered with caps and posts and lagged. It has a manway opening at the center on the footwall side and a chute on each side of the manway. After the timbering is completed the chute raises are driven to a height of 30 feet above the track, and subdrift connections are made between the chutes. The subdrift then is widened to the full width of the ore, and an inclined manway raise is driven to the level above for ventilation, pipe lines, and travel.

Shrinkage stoping proceeds to within 20 feet of the level above in horizontal lifts about 8 feet high and 60 feet long. In wide stopes

[a] Henry, R. J., Mining Methods and Costs at the Teck-Hughes Gold Mines, Ltd., Kirkland Lake, Ontario: Inf. Circ. 6322, Bureau of Mines, 1930, 12 pp.

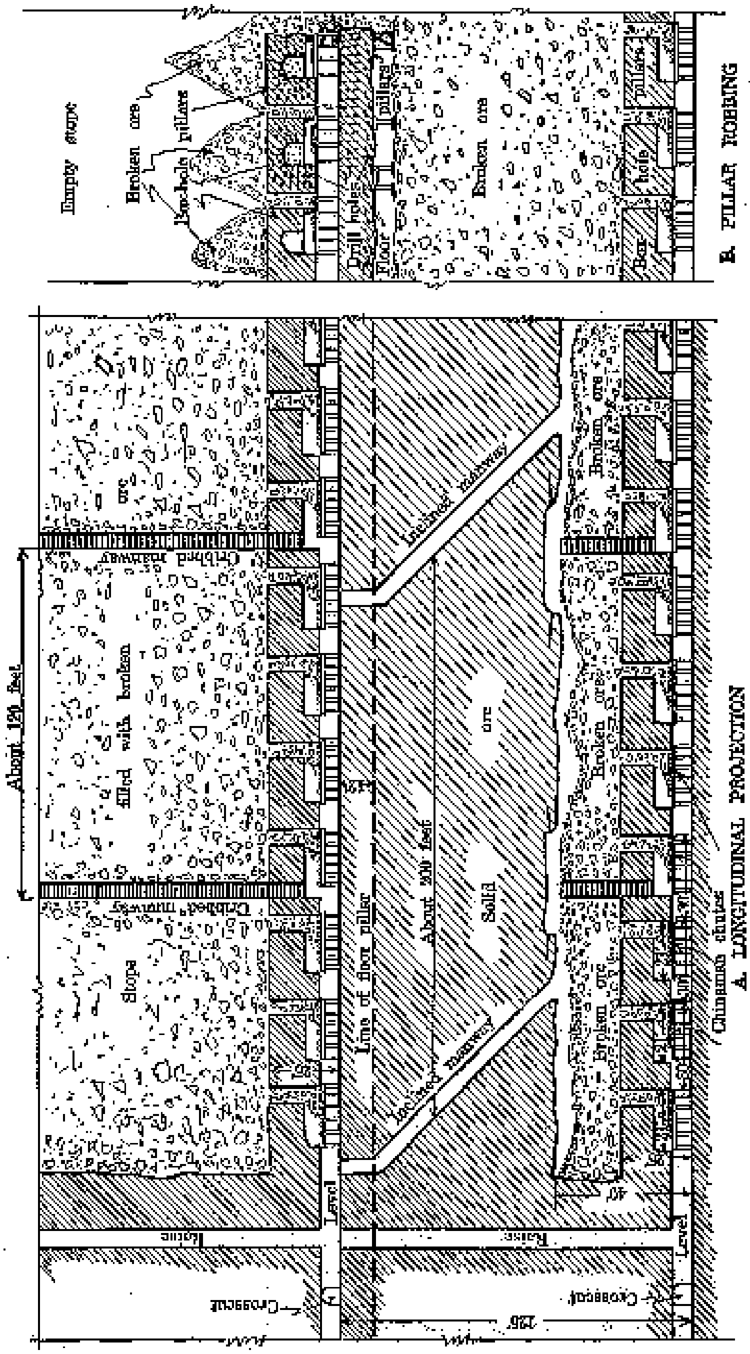


FIGURE 31.—Shrinkage stoping over boxhole pillars, using chinaman chutes, Teck-Hughes mine, Kirkland Lake, Ontario.

loose slabs in the back are caught up on posts. The stopes have to be scaled after each blast. Vertical raises are driven through the 20-foot back pillar at 60-foot intervals and recovery of the pillars is begun when all the ore that can be pulled from the stopes on the level above has been drawn. The boxhole pillars of the upper level and the back pillar of the stope below are pulled back together, usually in sections about 60 feet long. A small raise 2 short rounds high is put up in the center of each boxhole pillar, the pillar below is thinned to 12 feet, holes are fanned out in the boxhole pillars around the small raises, and holes are drilled upward and downward into the back pillar. (See fig. 31, *B*.) These holes are loaded and blasted electrically in one blast with instantaneous detonators from a 110-volt lighting circuit. As many as 3,000 holes have been fired simultaneously. Most of the timber is taken out, and the remainder is bored and blasted with the big blast so that it will not block the chutes below.

Direct stoping costs during a typical month in 1928 were as follows:

Direct stoping cost, Teck-Hughes mine, typical month, 1928

	Cost per ton of ore hoisted		Cost per ton of ore hoisted
Labor	$0.761	Timber	$0.042
Supervision	.096	Other supplies	.033
Compressed air, drills, and steel	.309	Total	1.557
Explosives	.316		

The scale of wages was as follows: Shift bosses, $7; mucker bosses and chute blasters, $5; drill runners, timbermen, and scalers, $4.75; muckers, trammers, and drill helpers, $4.25.

Productivity of stope labor in man-hours per ton during October 1928 was as follows:

Productivity of stope labor, Teck-Hughes mine, October 1928

Occupation:	Man-hours per ton	Tons per man-shift
Breaking	1.400	5.71
Timbering	.117	67.38
Total	1.517	5.27

KIRKLAND LAKE GOLD MINING CO., KIRKLAND LAKE, ONTARIO [43]

The ore occurs in fault zones in igneous rocks, lamprophyre, syenite, and syenite porphyry. The ore bodies are roughly lenticular and longer vertically than horizontally. Virtually all ore lenses rake west, going down in echelon formation. They range from 30 to 300 feet in horizontal length and 100 to 500 feet along their longer axis, are from a few feet to 30 feet thick, and have a steep dip. Most of the lenses worked have been on the hanging or foot wall of a nearly vertical fault zone. A few high-grade veins dip at a much flatter angle. The ore consists of country rock impregnated with

[43] Dumbrille, J. C., Mining Methods of Kirkland Lake Gold Mining Co., Ltd., Kirkland Lake, Ontario: Inf. Circ. 6490, Bureau of Mines, 1931, 12 pp.

and replaced by highly fractured quartz. The rocks in the ore zone are fractured and cut into blocks by intersecting slips or mud seams.

Shrinkage stoping has been used on timbered-drift backs. Chutes are spaced at 10-foot intervals, and 4- by 4-foot manways are carried to 50 feet above the level at convenient intervals. Raises are driven through from this point to the level above, the level interval being 125 feet. Entrance to and exit from the stopes is through these raises. In shrinkage stoping about 2 feet of ore is left on the walls. This will eventually slough off, and the walls will remain in better shape than if the ore was blasted. Stopes are drilled with 3¼-inch drifter machines. Flat breast holes up to 10 feet deep are drilled and blasted with 40-percent gelatin dynamite.

Sills are recovered by mining directly under the old back timbers of the level, starting at the inside end and retreating toward the shaft. The empty stope above usually will stand long enough to permit this procedure. Waste which sloughs from the walls is mostly in the form of very large blocks, which hang up in the stope and allow virtually all of the broken ore to be removed. In shrinkage stoping at this mine the dilution of ore with waste is high—at least 30 percent—due to the character of the wall rock.

Direct stoping costs for a typical month in 1930 were as follows:

Direct stoping costs, Kirkland Lake Gold Mining Co., 1930

	Cost per ton of ore hoisted	Cost per ton of ore broken	Cost per ton of ore and waste hoisted
Labor	$0.77	$0.671	$0.485
Supervision	.07	.061	.044
Compressed air, drills, and steel	.23	.200	.146
Explosives	.46	.400	.290
Timber	.03	.027	.019
Other supplies	.06	.053	.037
Total	1.62	1.412	1.020

The wage scale was as follows: Machinemen and timbermen, $4.75, and machine helpers and muckers, $4.25.

Labor performance in stoping for a typical month was as follows:

Labor performance in stoping, Kirkland Lake Gold Mining Co.

Occupation:	Man-hours per ton of ore hoisted	Tons per man-shift
Breaking	0.90	8.9
Timbering	.07	114.4
Shoveling	.48	16.7
Total	1.45	5.5

VIPOND MINE, TIMMINS, ONTARIO [47]

This mine is located at the nose and on one leg of a synclinal fold in an area of unusually contorted, sheared, and schisted basaltic

[47] Dye, Robert E., Mining Practices and Costs at the Vipond Mine, Timmins, Ontario, Canada: Inf. Circ. 6525, Bureau of Mines, 1931, 11 pp.

rocks intruded by quartz porphyry. The ore bodies occur erratically and vary greatly in attitude and form. The ore consists of quartz veins and irregular masses of mineralized schist impregnated with quartz. Individual masses of ore range from 10 or 12 feet to 400 feet in length and from the narrowest workable to 60 feet in width and dip at varying angles. The dips generally are more than 60°, although a few flat veins dipping about 30° have been mined. The ore is quite hard and stands well. The walls generally are well-defined, but part are "assay walls." The wall rocks are firm and stand well, except where they are badly sheared or consist of graphitic material.

Levels are established at depths of 100, 200, 300, 400, 500, 600, 733, 866, 1,000, and 1,200 feet. Shrinkage stoping is employed exclusively for mining the ore. The choice between stoping directly on timbered-drift backs or on arch pillars (termed "boxhole pillars" in Ontario) depends upon the width and grade of the ore. Narrow veins always are stulled and lagged if the back of the level drift is of workable grade. If the back of the drift is not of workable grade, exploration raises are driven at favorable points. If they encounter ore a stope is begun, and the ore is drawn off through them and through intermediate raises put up to tap the bottom of the stope. Ore bodies up to 25 feet in width are stulled and lagged, the stulls being supported on posts. Ore bodies over 25 feet in width are opened by boxhole raises, and the arch and floor pillars are extracted after the stope has been drawn empty. Where timbering is employed the chutes are spaced 15 feet center to center. Where boxholes are put up they are rarely spaced closer than 25 feet, and if they have to be driven an appreciable distance to reach the bottom of the ore they are 40 feet or more apart. Cribbed manways are carried in each stope at 100-foot intervals. Stopes are carried about 60 feet above the level, a raise is then put through to the level above, and all manways but one are covered over. When the stopes are drawn the walls are trimmed, and bad ground is supported by sprags or stulls. The good walls have made it possible to scale all stopes and clean out all broken ore. Leyner machines are employed to drill horizontal breast holes in the back.

Direct stoping costs from August 1, 1929 to July 31, 1930, when 113,329 tons of ore were hoisted and 104,381 tons broken, were as follows:

Direct stoping costs, Vipond mine, Aug. 1, 1929 to July 31, 1930

	Cost per ton of ore hoisted	Cost per ton of ore broken
Labor	$0.626	$0.680
Supervision	.084	.091
Compressed air, drills, and steel	.250	.271
Explosives	.180	.195
Timber	.046	.050
Other supplies	.089	.097
Total	1.276	1.384

Labor in stopes for drilling, blasting, and timboring during the same period was 1.176 man-hours per ton of ore, or 6.8 tons of ore per man-shift. Consumption of explosives in stoping averaged 1.05 pounds of 40-percent gelatin dynamite per ton of ore.

MC INTYRE MINE, SCHUMACHER, ONTARIO[44]

The McIntyre mine is on the north limb of the Porcupine syncline within an area of Keewatin lavas, later intrusions of quartz porphyry masses and offshoots therefrom, and diabase and albatite dikes. The rocks in the productive area have been subjected to compressive forces at several stages in their history. The complex folding, the fractures now occupied by veins, the prevein and postvein faulting, and the broad belt of schist are evidences of these forces. The zone of schistosity extends northeast for several miles over a width of 1,600 feet, in which the most productive veins occur. The strike of the schist is about N. 65° to 80° E., with a steep dip south. Several planes of fracturing in the schisted zone cut across the schisting at a small angle, and it is in these that the productive veins occur. The productive veins are found in the lava schists and seldom are workable in the porphyries.

The ore is of two types: Quartz veins with included schist fragments and irregular replacement bodies. The ore bodies are nearly vertical as a rule, but some dip north about 60°. Their length varies up to 1,200 feet and their width to 100 feet. The average width is about 10 feet. The ore and walls generally stand well, except where amorphous carbon has developed in the schists and in certain badly faulted areas.

The ore is a gold ore which had averaged $8.95 per ton up to March 31, 1932, when a total of 6,015,718 tons had been mined and milled. High-grade pockets occur locally, usually in a quartz-vein filling.

Up to 1930 most of the ore had been mined by shrinkage stoping, although some wide ore bodies in heavy ground had been mined by square-setting. Since then cut-and-fill stoping has been employed more and more, and in 1932 about 50 percent of the production came from cut-and-fill stopes.

In shrinkage stoping raises sometimes are put through from level to level before stoping is begun, but often the stopes are mined to 60 feet above the level before a raise is put through. In preparing for stoping the drifts are slashed out the full width of the ore and the backs taken down to a height of 18 or 20 feet. The level then is timbered with posts and caps on about 4- to 5-foot centers, and chutes are installed at intervals of 15 to 30 feet along the drift. If the vein is wide and comparatively low grade the backs are not shot down, but a series of "boxhole" raises is put up on about 25-foot centers to a height of 30 feet and connected by a subdrift, from which regular shrinkage stoping is started. Breast stoping is used, and holes drilled with 3½-inch drifters are fanned out in the wider stopes. In some of the longer stopes pillars are left where areas of stress develop locally.

At present, shrinkage stoping is employed in isolated stopes, in stopes where it is anticipated that the walls will be strong and regular, and in low-grade stopes of considerable width.

[44] Skavlem, H. G., Mining Methods at the McIntyre Porcupine Mines, Ltd., Schumacher, Ontario: Inf. Circ. 6741, Bureau of Mines, 1933, 18 pp.

The tonnage broken per man-shift in shrinkage stoping is about 20, and the consumption of explosives is about 1 pound per ton of ore broken.

ELKORO MINES, JARBIDGE, NEV.[43]

The gold-silver ores of the Jarbidge district occur in quartz veins in a faulted series of Tertiary volcanic flows which have undergone extensive alteration. The veins dip at 60° to 80° and vary greatly in length and width. The width of the main vein averages about 6 feet and ranges from narrow stringers to 50 feet in rare instances. This vein is enriched by small leads which enter or cross the main vein at an angle of about 60°. These branching or intersecting veins do not carry values away from the main ore body. An operating difficulty is presented in mining the ore in splits extending 3 or 4 feet into the hanging wall. In stoping this ore a shelf is formed in the hanging wall which requires a large amount of timbering and blocking. Except at the crossings the walls of the veins are defined clearly. At these places some of the wall rock usually is broken to make sure that all commercially valuable material is recovered.

No very high grade ore is found, the average value of the mill feed over a period of years having been slightly over $10 per ton. Here and there along the vein lean spots are left as pillars in the stopes. The mine is developed by adit levels, which are approximately 150 feet apart vertically and connected by raises or winzes at convenient points to explore the ore bodies; these are placed so as to serve later for stope manways or ore chutes and to ventilate the stopes. During 1930 all production came from shrinkage stopes, although in earlier and later years cut-and-fill stoping was employed in some of the stopes.

Drifts are driven the full width of the ore, and a slice is shot out of the back in preparation for shrinkage stoping. Lagged stulls are placed and lagged over under the section to be stoped, and chutes are installed at 15-foot intervals. Manways generally are about 125 feet apart, which is the length of most stope sections. Usually each section is mined by horizontal slices, but adjacent sections are not necessarily kept abreast. Drilling is done with stopers. The vein material drills easily and is of medium hardness; a driller seldom uses more than three sets of steel a day. Only about 23 percent of the broken ore is drawn from the stopes during stoping to maintain the desired working space between the top of the broken ore and the back of the stope. (This compares with 30 to 40 percent in most other mines employing shrinkage stoping.) Stopes are carried as high as 250 feet with very little timber. Pillars are left where the ore is lean or the wall is weak. When breaking has been completed (about 4 feet below the level) a strong overhead bulkhead of poles is placed the entire length of the stope to protect the timber crews who follow the ore down, clean the walls of ore, and prop loose ground during final drawing of the stope.

Direct stoping costs during 1930, when 57,539 tons of ore were hoisted, were as follows:

[43] Park, John Furness, Mining Practices, Methods, and Costs at Elkoro Mines, Jarbidge, Nev.: Inf. Circ. 6543, Bureau of Mines, 1931, 12 pp.

Direct stoping cost, Elkoro mine, 1930

	Cost per ton of ore		Cost per ton of ore
Labor; drilling and blasting	$0.382	Explosives	$0.107
Labor; timbering and cleaning out stopes	.537	Timber	.164
Compressed air, drills, and steel	.162	Other supplies	.258
		Total	1.610

Direct stoping costs in man-hours per ton were as follows:

Occupation:	Man-hours per ton	Tons per man-shift
Breaking (drilling and blasting)	0.555	14.4
Timbering, mucking, and cleaning out stopes	.818	9.8
Total	1.373	5.8

Consumption of explosives was 1.239 pounds of 40-percent gelatin dynamite per ton of ore broken.

ENGELS MINE, PLUMAS COUNTY, CALIF.[46]

At the Engels mine the ore occurs in shear zones in diorite and altered diorite. The main ore body consists of three shear zones, of which two are nearly parallel while the third diverges from the other two by raking off to the north. The ore bodies strike N. 60° E. and dip 80° west. Above the fourth level much of the ore assayed 4 percent copper, but below this level it has averaged 1¾ to 2½ percent. The main ore body had a maximum width of 100 feet and a length of 830 feet. The diorite and altered diorite are very firm, tough rocks and stand well without support. The mine is developed by adits and shafts, with levels at intervals of about 150 to 200 feet.

The ore is mined in longitudinal shrinkage stopes carried on arch pillars over the drift backs. These pillars are 18 feet thick. Individual stope sections are 90 to 120 feet long and are separated by transverse stope pillars 20 feet thick. A manway and supply raise is run up in alternate pillars, with crosscuts driven into the stopes on each side at vertical intervals of 25 feet. Extraction drifts are driven along the footwall, and chutes are installed at 30-foot intervals along the drifts. At the chutes the drift is cut out to a height of 14 feet above the rail, and the chute raises are extended three rounds. Chutes are installed and the raises completed to 32 feet above the rail. The raises are belled out and connected so as to leave an 18-foot pillar over the back of the drift. One of the chute raises is used for a manway and for air and water lines until the stope is high enough to connect with the lower crosscut from the pillar raise. The drift connecting the chute raises is widened out to the width of the ore body, and stoping is begun by drilling the back with drifter machines. In flat backs two rounds are drilled in opposite directions from the same set-up. These rounds consist of a single row of 3 holes in narrow stopes drilled at an upward angle and 4 holes in wide stopes. This leaves breasts in which breast rounds consisting of two rows of holes are drilled. The average depth of holes is about 7.7 feet. During 1928 an average of 7.3 holes total-

[46] Nelson, W. I., Mining Methods and Costs at the Engels Mine, Plumas County, Calif.: Inf. Circ. 6260, Bureau of Mines, 1930, 22 pp.

ing 56.2 feet was drilled per man-shift. Stope holes are blasted with 40-percent ammonia dynamite (seven sticks to the hole) with stemming cartridges.

The crew in a stope consists of a miner and a mucker for each face in cutting-out operations. After stoping is under way 1 to 3 miners and 1 or 2 pluggers work in each stope. The pluggers assist the miners in setting up, plug the large boulders, and break up smaller boulders with hammers. Stope rounds are blasted electrically from a 220-volt current, using series connections.

Stopes are carried within 20 feet of the level above and are left standing full of ore until the stopes from the upper level have been emptied. The back of the stope then is mined through to the level. At the level the angle of the holes is changed from nearly flat to 45°. The first cut is drilled with holes 9 feet deep and 4 feet apart, and 10 or 20 holes are blasted at a time. This cut is continued for the length of the stope. The next cut breaks through to the level. In weak ground the second cut follows the first one closely. In removing the pillars from the level to the bottom of the stope above, 75 tons of ore are broken per machine-shift compared to an average of 27 tons in the stopes.

Vertical stope pillars containing 146,526 tons of ore have been removed at a cost of 58 cents per ton for ore drilled by hammer drills and 28 cents per ton for ore drilled by diamond drills. Before the pillars are removed the shrinkage stopes are drawn empty and then mined one at a time. The method of drilling the pillars from stations in the manway raises, using hammer drills and diamond drills, is described and illustrated in detail in Bureau of Mines Information Circular 6260. Comparative costs of breaking ore in vertical pillars are given as follows:

Comparative costs of breaking ore in vertical pillars, Engels mine

Method	Tons in pillar	Number of holes	Feet of hole	Total cost of drilling [1]	Total cost of loading and explosives	Total cost per ton broken
Hammer drills	7,475	1,206	12,144	$2,011	$1,489	$0.682
Diamond drill	40,178	17	9,208	8,815	3,052	.270
Do	11,205	6	636	2,440	260	.244

[1] Total cost of drilling includes estimated cost of sharpening and replacing worn steel, upkeep of machine drills, and labor for drilling.

Direct stoping costs, including pillar mining, in 1928 were as follows:

Direct stoping costs, Engels mine, 1928

	Cost per ton of ore hoisted
Labor	$0.318
Supervision	.093
Compressed air, drills, and steel	.135
Explosives	.157
Timber	$0.017
Other supplies	.040
Total	.760

The wage scale was as follows: Machinemen, $5; jackhammer men, $4.75; muckers, $4.50; timbermen, $5 and $5.50; timbermen helpers, $4.50; and shift bosses, $6.

Consumption of explosives in stoping averaged 0.813 pound per ton of ore mined.

MOUNT HOPE MINE, MOUNT HOPE, N. J.

The iron ores at Mount Hope occur within an ore zone 1,000 feet wide in a belt of pre-Cambrian rocks consisting of gray granitoid gneiss interlaminated with pegmatites. Disseminated magnetite is found throughout most of the country rock. Within the ore zone are three main parallel veins striking northeast, spaced 500 feet apart and dipping from vertical to 60°.[47]

The ore shoots are tabular, disposed edgewise, and conformable with the structure of the intervening gneiss. They dip southeast and pitch or rake 14° northeast. Although the ore bodies exhibit remarkable uniformity in their general attitude, local irregularities occur in the form of rolls and pinching and widening of the veins. The ore bodies are 150 to 400 feet high and a few feet to 40 feet thick; the average width of the largest deposit is 19 feet. They have been developed 7,500 feet along their downward pitch from the surface. There are two general types of magnetite ore; one is hard, massive, and granular, and the other is hard, massive, and laminated. The grade ranges from 35 to 65 percent iron. The granular type of ore is hard and tough, breaks large, and will stand unsupported over widths of at least 40 feet. The laminated type is weaker, drills and breaks more easily, and breaks in small pieces. The lateral limits of the ore usually are well-defined. The wall rock is a hard, solid, granitoid gneiss that stands well.

The mine is developed by vertical shafts and flat inclines driven parallel to the downward rake of the ore. The level interval ranges from 160 to 260 feet. Stopes usually are 340 feet long and separated by pillars of ore 30 to 40 feet thick. Entrance to the stopes is provided by a manway raise driven upward through the center of the pillar, which is connected to the stopes by short drifts. Paralleling the manway but driven from adjacent chute raises or grizzly chambers the pillar or mining raises are driven, forming the cut-off line between the stopes and the pillars. Usually each stope is provided with four ore chutes 81 feet apart. These are belled out over the chutes to make room for grizzly chambers, the bottoms of which are 27 feet above the level or the bottom of the incline. From the ends of the grizzly chambers undercutting raises are driven at 45°; from these, raises are cut back in the opposite direction at 45° to meet over the center line of the chute, thus forming a triangular pillar above the grizzly. In some stopes the grizzlies are eliminated, in which case there are eight chutes to a stope, spaced at 40-foot intervals.

The ore is mined entirely by shrinkage stoping on pillars without any timbering or filling. The stope is undercut over its entire length from the undercutting raises. The cuts are driven at 45° and extended upward until they reach the walls. Wet, self-rotating stopers and dry jackhammers equipped with standard stoper legs operated on a 2- by 12-inch plank are used for undercutting. In the weaker ore bodies two crews work in each stope, starting at opposite ends

[47] Sweet, J. R., Mining Methods and Costs at the Mt. Hope Mine of the Warren Foundry and Pipe Corporation, Mt. Hope, N.J.: Inf. Circ. 6601, Bureau of Mines, 1932, 21 pp.

of the stope and working toward each other. Each crew consists of two men provided with two machines. In the stronger ground drilling can be done in a greater number of faces, but at present (1932) only five machines are used in any one stope. Breast drilling in faces 6 to 15 feet high is practiced, the height of the face varying inversely with the width of the ore. Formerly, 6-foot holes were drilled vertically with stopers, but this practice has been superseded by breast stoping using holes about 14 feet deep. These holes are drilled with light hammer drills fitted on standard stoper legs, which are pushed ahead on top of a plank supported on staging. The following figures give comparative results of the old method of drilling (with stopers) and the new method. The new method is much the safer, as the miner is working under ground thoroughly tested and trimmed, and the drilling is done ahead of him rather than above him.

Comparative results, old and new drilling methods, Mount Hope mine

	1926, old method	1930, new method
Tons broken per machine-shift	26.66	82.53
Tons per man-shift, all labor chargeable to stoping	19.90	57.59
Man-hours per ton, all stope labor	.402	.138
Tons broken per foot of hole drilled (stoping)	.745	2.670
Pound of powder used per ton of ore broken (stoping)	.611	.384

The present stoping method permits recovery of 90 percent of the ore, the balance being left in stope pillars, toe pillars where the inclines leave the levels, stubs formed by undercutting raises, and grizzly-chamber pillars. Dilution of the ore with waste probably is less than 10 percent.

Stoping costs during 1930 were as follows:

Stoping costs, Mount Hope mine, 1930

	Cost per long ton (2,240 pounds)
Stope development:	
Labor	$0.106
Supervision	.004
Compressed air, drills, and steel	.110
Explosives	.042
Timber	.012
Other supplies	.024
Total stope development	.298
Stoping:	
Labor	.074
Supervision	.029
Compressed air, drills, and steel	.090
Explosives	$0.038
Other supplies	.002
Total stoping	.233
Blockholing:	
Labor	.043
Supervision	.003
Explosives	.048
Other supplies	.007
Total blockholing	.101
Total direct stoping cost	.632

Consumption of explosives in stoping was as follows:

	Pound per long ton
Stopes proper:	
40-percent strength	$0.302
50-percent strength	.012
Blockholing:	
40-percent strength	.426
50-percent strength	.005
Total explosives in stopes	.745

ORDINARY SHRINKAGE STOPING IN WIDE ORE BODIES

In wide ore bodies the ordinary or conventional shrinkage-stoping method may be employed for mining the ore in transverse stopes separated from each other by transverse pillars. In the narrower ore bodies described in the previous section the ore was mined in longitudinal stopes, the width of the vein being such that the back of the stopes stands unsupported over the full width of the ore body. In wide ore bodies, however, the width may be so great that the area of unsupported stope back would be too great to hold up if longitudinal stopes were cut out from wall to wall. To overcome this difficulty the stopes are driven across the ore body at right angles to its strike, and the width of the ore body becomes the length of the stopes, their width being determined by the strength of the ore and limited to a span that will stand safely.

After the shrinkage stopes are completed and drawn the pillars are mined. If the ore is very low grade it may not be profitable to attempt complete recovery, and the pillars may be whittled away until they finally fail, the remaining ore being lost. Sometimes the pillars are recovered by caving. This may be done as follows: The pillars between the stopes are kept as thin as possible during mining of the stopes but thick enough to resist disruption while the stopes are being carried up. When the stopes have nearly reached the level above, the stope pillars are cut through, and the back pillar is shot down or weakened until it caves. After standing for some time the thin pillars take weight, crush, and break up, and the pillar ore is drawn off with the stope ore as the stopes are pulled empty. With this system usually some ore is lost due to piping through of waste from the walls and upper levels; this prevents all the ore being drawn without excessive dilution.

If the ore is of moderate or high grade it usually pays to go to additional expense to recover as much of the pillar ore as possible with minimum dilution. To accomplish this the emptied shrinkage stopes are filled with waste rock, sand, or mill tailings, and the pillars then are mined from between the filled stopes by top slicing or square-setting, although cut-and-fill stoping has been employed.[48] The practice at the Homestake mine, Lead, S. Dak., is perhaps the best-known example of shrinkage stoping by transverse stopes in wide ore bodies and mining the pillars after filling the stope.[49]

HOMESTAKE MINE, LEAD, S. DAK.

At the Homestake mine in South Dakota the ore bodies occur as replacements of a limestone member 40 to 60 feet thick around the nose and flanks of a plunging anticlinal structure and are limited by a shear zone on the footwall side which dips about 70° toward the east. The axis of the anticline strikes a little west of north and plunges toward the south at an angle of about 40° to 45° from the horizontal. As a result of drag folding, the ore is very wide in places (up to 400 feet); this folding has also served to repeat the ore so that on a given horizontal plane there are apparently five principal ore lenses separated by interfolded country rock. The plunge of the structure

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[48] Ayvazoglou, W., The Method of Underground Mining of Iron Ore in the District of Krivoy Rog (translated from Bouldovsky, Min. Jour. of Moscow, 1922): Inf. Circ. 6254, Bureau of Mines, 1930, 48 pp.

[49] Jackson, Chas. F., Shrinkage Stoping: Inf. Circ. 6293, Bureau of Mines, rev. 1930, pp. 19–23 and fig. 20.

caused the ore body to pitch out to a surface outcrop on the north end of the mine; on successively lower levels the ore is further south and, due to the easterly dip, further east.

The ore-bearing member is one of a series of pre-Cambrian rocks, composed principally of slates, schists, and quartzites, much altered by the great pressure to which they have been subjected. These rocks have been intruded by dikes of rhyolite and phonolite. The rhyolite appears to parallel the structure and sometimes forms one or both walls of the ore bodies. In places it also cuts across the structure, however. The immediate walls of the ore bodies are usually slate but sometimes are schist or porphyry. They are hard and strong. Barren rocks are infolded with the ore so that in some places large horses of slate separate it into two or more parts.

The ore itself is hard, tough, and dense and is composed of hornblende, chlorite, and other heavy ferromagnesian minerals; quartz; and heavy sulphides, principally pyrite, pyrrhotite, and arsenopyrite with which the gold is associated. Visible gold occurs but is not common. Often the ore has a distinctly banded appearance developed by the pressure and folding to which it has been subjected.

The ore stands well except in old pillars which have remained for a long time subjected to considerable weight, in which case it is badly crushed and broken. It breaks large, is hard to drill, and is tough-breaking.

Work has been carried on at the mine for over 50 years. The first operations were at the north end where the ore was near or at the surface. These early operations were very haphazard and consisted of opencut work, followed by irregular underground work in open stopes and square-set stopes. Large stopes were opened up and mined until they started to crush, when they were abandoned and work was started in a new place with the result that much ore was left in floors or "crowns" and in stope pillars.

During 1929, 52 percent of the ore mined at Homestake came from caving operations beneath this area and from similar operations in the Star shaft pillar. As this paper deals primarily with shrinkage stoping, the caving operations will not be described.

In the newer and deeper parts of the mine to the south and extending from the 900 to the 2,300 level, the ore is mined in transverse shrinkage stopes between pillars, the pillars being later mined by square-set stoping.

The following notes on the mining methods were prepared by the author of the present paper after a study of the mine in March 1930. During 1929, 1,437,935 tons were mined and milled, 52 percent of which was from draw holes above the 900 level, 20 percent from shrinkage stopes, and 28 percent from crown and stope pillars.

The main orebody between the 1,400 and 1,850 levels averages 825 feet long by 130 feet wide, with maximum widths of about 250 feet. On the levels above, the width reaches a maximum of 400 feet. Other ore bodies separated from the main ore body by 100 feet or more of barren rock are 300 or 400 feet long by 25 to 55 or 60 feet wide.

The wide ore is mined by transverse shrinkage stopes 60 feet wide, separated by pillars 40 feet wide. Above the 1,250 level, the level interval is 100 feet, and below this level it is 150 feet down to the 2,600 level which is the lowest level (1929).

The mine is developed by vertical shafts from which crosscuts are driven at each level. From these crosscuts haulage drifts are driven in the hanging wall and footwall and at some distance therefrom. From these drifts crosscuts are driven to and through the ore body at intervals of 100 feet. Formerly, and until quite recently, these crosscuts were driven through the centers of the 40-foot pillars with drifts at 30-foot intervals driven at right angles into the stope. At present the practice is to drive the crosscuts through the center of the pillar as before, but they are branched before reaching the ore so that the branches will follow the pillar lines. (See fig. 32, *A*.)

The stope is next silled out 60 feet wide to a height of 11 or 12 feet and clear across the ore body. (See fig. 32, *B*.) The next step is to arch the back to a height of about 40 feet along the center line of the stope (fig. 32, *C*) and to run a waste raise to the level above from the top of the arch so that waste filling may be introduced later. A manway raise is driven in the footwall in the center of the pillar to the level above, from which crosscuts are driven later at 60, 90, and 120 feet above the level for connection to the stope.

The next step is to shovel out all the broken ore from the stope. This is done by hand-shoveling into 1-ton cars. Chute lines are then stood across the stope along both pillars. (See figs. 32, *A*, and 32, *C*.) The sets are regular sill-floor square-set timbers placed on 6-foot centers, with chutes in every fourth set.

The floor of the stope is covered with a mat of old timber, and the sets of the chute line are diagonally braced with long timbers which are later recovered. Filling is then introduced from the level above through the central raise, and the stope is filled to within a few feet of the back. (See fig. 30, *C*.)

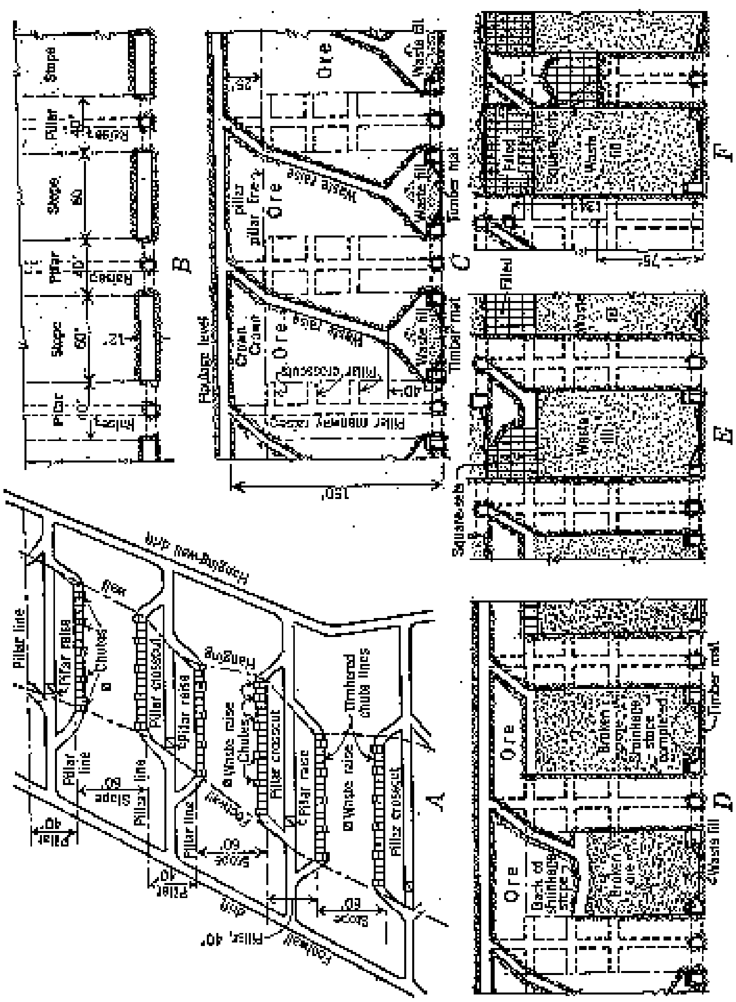

FIGURE 32.—Shrinkage stoping, Homestake mine, South Dakota: *A*, Sill floor plan showing part of a level, pillars, and chute lines; *B*, vertical section showing stope sliced out; *C*, vertical section showing stopes arched and center filled ready to start shrinkage stoping; *D*, vertical section through shrinkage stope; *E*, shrinkage stopes mined and filled, crown pillars being mined by square-setting; *F*, mining stope pillar by square-setting.

The stope is then carried up by regular shrinkage methods to within 25 or 30 feet of the level above, this much being left as a crown pillar over the stope. (See fig. 32, *D*.) In carrying the stope upward it is more often bellied in the middle than arched, and the holes are usually drilled from near the pillar lines toward the center. The drilling is done with medium-weight, Leyner-type machines mounted on two-jack column bars and using steels up to 11 feet long. Machines are equipped with anvil-block chucks, and 1-inch, quarter-octagon steel with plain shanks and cross bits is used. The ore is hard and tough, and about 25 to 40 feet are drilled per machine-shift.

At the beginning of each shift an average of about 2 hours is required for barring down the back. The ore breaks large, and the chunks are blockholed in the stope using jackhammer drills and steel equipped with rose bits.

The excess ore, amounting to about 40 percent of that broken, is drawn off through the chutes into 1-ton cars which are hand-trammed to the main haulage drift, whence they are taken by compressed-air locomotives to the shaft.

When the shrinkage stope is completed and the reserve of broken ore has been drawn off, the stope is filled from the level above with waste. This waste is derived from development in rock and from waste sorted from the ore at the draw points in the caving section of the mine. In 1929, 617,914 cars (1 ton) of waste were dumped into the stopes for fill. The waste runs from the raise directly into the stope until it builds up to the level of the crown, after which a waste chute is installed and the filling is spread by the use of 1-ton cars.

If the stope crown must be held for a long time due to operations above, it is caught up by props or cribs on top of the filling, or the filling is packed as tightly against the back as possible.

In mining the pillars, square-sets are used; the crown pillars over the stopes are mined out first, then the stope pillars are taken. (See fig. 32, *E*.) The sill sets are stood on sills laid on the leveled fill. Square-setting starts from either the hanging or foot wall, depending upon the location of raises for ore and filling, and is carried up to the mat above in sections 3 to 5 sets wide, depending upon the condition of the pillar. After the first section is mined out it is filled; however, the last line is usually kept open for starting the next stope if the ground is not too heavy. The balance of the crown is similarly mined and filled in sections, retreating to the opposite wall. Filling may be obtained from the waste raise or by tapping the fill in the old stopes above.

After the crowns on both sides of a pillar have been mined and filled, the pillar is then mined by square-setting. Longitudinal drifts are driven from the pillar raise through the center of the pillar from foot to hanging wall at 75 feet and 134 feet above the haulage level. (See fig. 32, *F*.) The top section is mined first, starting at the 75-foot sublevel and at the wall opposite the raise. (This would usually be the hanging wall, but sometimes there is a hanging-wall raise also, in which case stoping could start at the footwall.) The pillar is silled out in sections, usually 3 to 5 sets wide, leaving a thin skin of ore against the filled stopes on each side; a square-set stope 3 to 5 sets wide is carried up to the level above and filled, usually, however, leaving the last line of sets open for convenience in starting the next section. The pillar is thus mined back and filled in sections to the opposite wall. The bottom half of the pillar is then mined in the same manner, starting from the sill floor on the haulage level.

If the pillars have not stood too long, their recovery in the manner described is very easy and simple. Sometimes, however, if they have been standing for a long time they take weight and are badly crushed and broken. In this event the mining is not so easily done; the sets are thrown out of line, diagonal bracing may be required, or it may be necessary to follow the timbering closely with fill, keeping only a manway and a chute set open.

In some instances where the fill on either side of a pillar has become well consolidated and the pillar has remained solid and unfractured, it has been possible to mine the pillar by shrinkage stoping with only slight waste dilution when drawing the pillar stope empty.

The foregoing was in brief the standard method of mining in wide ore in 1929. In narrow ore, 50 feet wide or less, the stopes are sometimes mined by longitudinal shrinkage stopes employing only one line of chutes along the footwall of the ore, in which case the first filling is from the top of the chute line sets to the hanging wall at the angle of repose of the fill material.

Where the south footwall cuts a level, a triangular block of ore is left between that level and the next level above. Here it is customary to drive rock raises to the ore from footwall drifts or crosscuts under the ore body. These raises are usually spaced at about 30- to 40-foot intervals north and south, and as the bottom of the ore pitches up to the north each successive raise going north is longer than the preceding one. The ore is mined around the tops of these raises to the east and west walls and is then mined up to the level or crown above by shrinkage stoping. Where the ore is very wide, it may be necessary to use the regular 60-foot stope with 40-foot pillar system and two or more lines of raises. In narrow ore the full width is sometimes mined in one long longitudinal stope, with filling closely following the stoping.

Direct stoping cost at the Homestake mine during 1929 was $1.14 per ton. This figure is the average, however, for all stoping (shrinkage, square-setting, and caving) and includes $0.18 for filling. Twenty percent of the ore mined during that year was from shrinkage stopes, 28 percent from stope and crown pillars, and 52 percent from caving. The direct stoping costs were distributed as follows:

Distribution of direct stoping costs, Homestake mine, 1929

	Cost per ton	
Labor, mining	$0.5649	
Supervision, mining	.0453	
Compressed air, drills, and steel; mining	.1572	
Explosives, mining	.0913	
Timber mining	.1083	
Subtotal		$0.9670
Labor, back filling	.1102	
Supervision, back filling	.0105	
Compressed air, drills, and steel; back filling	.0306	
Explosives, back filling	.0029	
Timber, back filling	.0216	
Subtotal, back filling		.1758
Total direct stoping and back filling		1.1428

The average contract price for labor in pillar mining (square-setting) was about $1.10 per ton. The contract price for sorting and loading caved ore at the grizzlies was 10 cents per ton, and the labor cost of caved ore delivered at the chutes was probably not over 15 cents per ton (author's estimate), giving a total labor cost of 25 cents per ton in cars for this class of ore. The direct stoping cost in shrinkage stopes only, on the basis of the foregoing figures, is then estimated to be about $1 per ton, exclusive of back filling.

MODIFIED SHRINKAGE STOPING IN LARGE ORE BODIES

BRISTOL MINE, CRYSTAL FALLS, MICH.

Figure 33 illustrates the stoping method at the Bristol mine, where transverse shrinkage stopes separated by narrow transverse pillars are mined, and the pillars are caved and drawn with the shrinkage ore. A rather unusual condition exists in that a hard band of ore runs longitudinally down the middle of the ore body with softer and weaker ore on each side (especially on the hanging-wall side) extending out to the walls. The ore body is approximately 1,000 feet long and 100 feet wide. The ore is hematite.

Haulage levels are 125 feet apart vertically. On each level a drift is driven in the footwall close to the ore, and another drift is run in the ore along the hanging wall. These drifts are connected by crosscuts on 37-foot centers, from which chute raises are put up at 25-foot intervals. Stoping is begun by first cutting off the ore body at the east property line and at the west boundary by

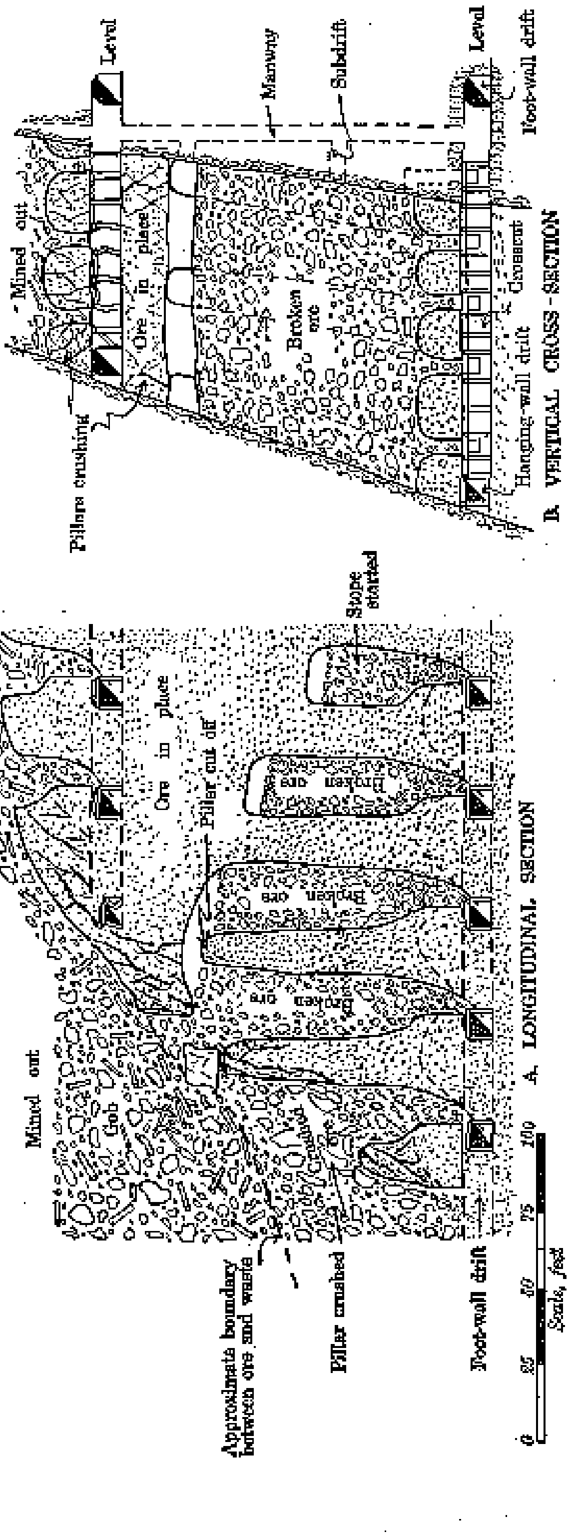

FIGURE 32.—Shrinkage stoping, Bristol mine, Michigan; wide ore body, showing breaking of pillar by caving.

means of shrinkage stopes. Then, 15 feet above the haulage level shrinkage stopes are run across the ore body over the tops of each line of chute raises. The stopes are 15 to 25 feet wide on 37-foot centers and separated by pillars 12 to 22 feet thick. The stopes are connected along both walls and usually through the centers of the stope pillars and are then mined by shrinkage stoping to within 20 or 25 feet of the level above. The stopes are widened as they go up, the pillars becoming correspondingly thinner toward the top. At this height the pillars and back begin to take weight. To make certain that the floor pillar will crush the stope pillars are cut over at the top by drilling them full of long holes and blasting them with heavy charges. The floor pillar of the upper level breaks as it settles down, crushing the stope pillars at the same time. Many large blocks of ore settle without being broken enough to permit loading them into cars, making it necessary to do considerable blasting in the chutes. When the shrinkage stopes are mined the presence of a hard band of ore in the middle of the ore body, with softer ore along the walls, permits a rather unusual procedure. The hard ore is drilled with heavy drifters and blasted with 60-percent gelatin dynamite. The softer ore on either side breaks away to the walls with little or no drilling and blasting. Thus, actual shrinkage stoping is confined largely to the band of hard ore. When necessary to force the breaking of the softer ore holes are drilled toward the walls with machines set up under the hard ore. About half of the ore mined is drilled and blasted, the balance being broken by caving. After all the ore in the end stopes has been broken drawing is begun and continued until the gob from the caved upper levels appears at the chutes. Drawing is continued in the next lines of chutes and retreats toward the main crosscut from the shaft.

During 1928 all stope labor averaged 0.121 man-hour per long ton, equivalent to 66.1 long tons per man-shift. Corresponding figures for 1929 were 0.190 man-hour per long ton, or 42.1 long tons per man-shift.

Consumption of explosives in stoping was 0.393 pound per long ton in 1928 and 0.401 pound in 1929.

CLIMAX MOLYBDENUM CO., CLIMAX, COLO.

Before 1930 most of the ore at this mine was mined by shrinkage stoping in a series of parallel stopes separated by regular pillars, the pillar ore being caved and drawn with the shrinkage ore. The ore body is a large, irregular mass 200 to 400 feet thick, the limits of which had not been determined in 1930. The ore mineral is molybdenite disseminated through a highly silicified mass of altered granite. The ore stands well over considerable spans, and the walls are of the same character. The ore body has no well-defined walls, and the values gradually fade out around the boundaries. The average grade of the ore is 0.9 percent MoS_2.

The method of mining by shrinkage stoping has been described by Coulter.[50] The upper level of the mine has approximately 100 feet of backs, and the lower level (1929) is 200 feet below the upper one.

[50] Coulter, William J., Mining Molybdenite Ore at Climax, Colo.: Eng. and Min. Jour., vol. 127, Mar. 9, 1929, pp. 394–400.

The shrinkage stopes are 400 feet long and 50 feet wide, with intervening pillars 40 feet wide. The pillars are recoverd by a coyote-hole system. From the main lower level adit haulage drifts are driven right and left for 250 feet on the center of each stope and pillar section; that is, 90 feet center to center. Service raises are put up from each of these drifts, 100 feet from the adit each way, to a height of 35 feet, and 5- by 7-foot grizzly-level drifts are driven from the raises directly over the haulage drifts below, leaving a 27-foot back between the haulage drifts and the grizzly-level drifts. Chute raises are put up from the haulage drifts at 50-foot intervals, one on each side of the drift. One is driven into the stope block to intersect the grizzly level on the center line of the stope, and the other is driven to the center line of the pillar at the same level. Short crosscuts are driven from the grizzly drifts to cut the top of the chute raises, and room is made for a 15- by 8-foot grizzly over each raise. Raises are run up from the ends of each grizzly; these are belled out near the top at the level on which stoping is to start, which is 30 feet above the grizzly level. When this work is completed there are two lines of belled raises on the stoping level, the raises in each line being 25 feet apart center to center.

Shrinkage stoping is begun from this level by breasting out the stopes from pillar to pillar with Leyner machines. Service raises are put up at the ends of the stopes and after stoping is well under way are carried up with the stopes and serve for entrance and exit.

The stopes are developed above the grizzly level in advance of the pillars. Initial development of the pillars is the same as that of the stopes. At the north and south limits of the pillar raises are driven from the center line of the pillar to the back of the ore, starting from the mining level, which is 25 feet above the grizzlies. They are connected with the upper level for ventilation and handling of supplies. At vertical intervals of 20 feet, 3- by 4-foot coyote drifts are driven along the center line of the pillar from these raises. Small crosscuts are driven at 15-foot intervals along these drifts, 8 feet each way into the pillar, and are teed at the ends for powder pockets. From the mining level and the south raise a cut 12 feet wide by 7 feet high is driven down the center of the pillar to the north a distance of 150 feet, leaving pillars 12 to 14 feet thick on each side. From this cut the side pillars are drilled with holes reaching within 2 feet of the broken ore in the shrinkage stopes. Since the coyote charges are blasted on virtually a vertical line from the south end, undercutting on the mining level is advanced in stages to give the pillar a substantial heel to rest upon.

The pillar on the south end next is cut off by widening the raise and drilling out to the broken ore in the shrinkage stopes. The first line of coyote charges then is blasted with 15 feet of the side holes on the mining level. The swell is drawn, the next row of coyote charges and 15 feet more of side holes are blasted, and this is continued to the north end of the pillar.

The ground in this mine is hard and uniform and is virtually all quartz, which breaks exceptionally well. In the shrinkage stopes, 15- to 30-foot faces are carried, two machines working in the same breast. Each machine advances along a pillar line, and they both reach toward each other to cut out the intervening ground. In the breast

16- to 20-foot holes spaced 4½ to 5 feet apart are drilled. Four tons of ore are broken per pound of 50-percent powder in stopes. Clay tamping is used in all holes. The average break per machine-shift in stopes is 160 tons.

In drawing the stope and pillar ore it is expected that the capping will cave over the entire block, and careful drawing will be necessary to insure proper settling and prevent piping through of waste.

During 1929 the average daily output was 1,060 tons. Tonnage per man-shift stoping was 43.42, equivalent to 0.184 man-hour per ton. Tonnage broken per foot of stope development was 400.

Future blocks will be mined by a system similar to that employed at the Alaska Juneau mine, which will be described next.

ALASKA JUNEAU MINE, JUNEAU, ALASKA

The stoping method employed at the Alaska Juneau mine has been variously termed "caving," "undercut caving," "sublevel caving," and "shrinkage stoping." The method is not a caving method in the strict interpretation of that term, although it is referred to as a caving method by the consulting engineer of the company.[m] It is true that much of the ore is broken by caving and that caving is an essential part of the system, but the caving is induced by a series of large blasts which break the ore down into stopes kept partly filled with broken ore. The system is not a true sublevel caving method, as stoping advances from the level upward, whereas in sublevel caving, as defined in this bulletin, stoping begins at the top and progresses downward to the level one slice at a time. The particular term applied to the system is not, however, as important as its remarkable results in attaining costs that have made it profitable to mine an ore which contained, over a period of years, an average of only $1.12 per ton in gold and from which the recovery was only about 80 to 85 cents per ton. As the system appears to the authors to be a variation of shrinkage stoping it is included under that classification in this paper.

The following is abstracted from Bradley's paper:

> The gold occurs in quartz stringers and gash veins in a wide shear zone in slate and metagabbro. Stringer lodes, usually near the slate and gabbro contacts, are found in a zone from 1,000 to 2,000 feet wide. These lodes are made up of a network of quartz veinlets and isolated lenses varying in width from less than 1 inch to 3 or 4 feet. The higher-grade ore bands are not over 300 feet wide, while the lower-grade material between them varies from 25 to 100 feet. Clean quartz will average $6 per ton within the areas of commercial grade, while the rock outside of the quartz stringers is practically worthless. The ore has no hard and fast boundary except where cut by faults, and profitable mining ceases on a vague and indefinite line where the auriferous stringers and gash veins become too few in number.

The nature of the ore body made it unprofitable to practice selective mining of the higher-grade erratic veins and lenses, and the entire mass had to be mined wholesale, allowing dilution by the barren material between the gold-bearing portions of the deposit. Waste is eliminated on the surface by sorting and rejection of 53 percent of

[m] Bradley, P. R., Mining Methods and Costs, Alaska Juneau Gold Mining Co., Juneau, Alaska: Inf. Circ. 6186, Bureau of Mines, 1929, p. 6.

the rock trammed, followed by fine milling of the remaining 47 percent.

In developing the mining system due consideration was given all physical conditions surrounding the ore bodies. Full advantage was taken of the fault system, consisting chiefly of the Silver Bow fault, the Nugget Creek fault, and a number of subparallel sympathetic faults. The Silver Bow fault cuts across all the ore bands at a horizontal angle of 53° and divides the mine into two parts, the north half and the south half. Nugget Gulch fault is a strike fault and is the marker for the footwall of the ore; it dips 55° to 60°. On each fault plane there is considerable fault breccia, which sloughs away when undercut.

The stoping system consists of (1) cutting out stopes of large area, located so that the footwall of the stope is one of the faults; (2) driving raises from the back of the cut-out stope area to the next two levels above; (3) driving numerous powder drifts from and radial to these raises; and (4) providing bulldozing chambers for drawing ore out of the stopes into the loading chutes. The sequence of operations is shown in figure 34.

In preparing a stope two parallel drifts 40 feet apart are driven under the stope area. From these drifts chute raises are put up 50 feet to an intermediate level where the ore is to be drawn from the stope through bulldozing chambers. From the bulldozing chambers cut-out raises are driven 38° from the horizontal toward the hanging wall on one side and the footwall on the other; these are then connected by back cut-out raises from each. The undercutting then begins from these raises. Pillars 80 feet wide are left between stopes where needed; in addition, temporary pillars frequently are used to support weak roof during cutting-out work. While undercutting is being done stope raises are driven to the level above at about 100-foot intervals. The upper level is then used as a supply level and as a means of access to the powder drifts below. From these raises powder drifts are driven radially, usually about 40 feet long and with not over 50 feet of burden. When the area has been undercut completely blasting is begun, and the rock is drawn through the bulldozing chambers into the chutes below. It has been found that, for best results from powder-drift blasting and caving, the stope should have a horizontal cut-out area of at least 50,000 square feet.

Powder drifts driven from stope raises over the back of the stope are 4 by 3 feet in section and on a grade of about 10°. They are seldom over 60 feet long. The average depth of ground placed on a powder drift is 35 feet, which would require an average charge of 4,000 pounds of 40-percent dynamite placed in two piles 35 or 40 feet apart. Three, four, or even five such piles make up an ordinary blast. As a rule, only one series of powder drifts on the same level is blasted at a time in any one stope. The powder is a 40-percent ammonia dynamite in 1¼- by 8-inch sticks, which are packed tightly against the back of the powder drift. Each pile is tamped for about 15 feet with fine rock and detonated with double-countered Cordeau Bickford fuse. Enough broken ore is kept in the stopes to prevent any ill effects from air blasts due to caving. Up to the end of 1928, 313 blasts had been made, using 1,632,950 pounds of dynamite and averaging 20 tons of ore broken per pound of explosive.

In stopes where blasting is not accompanied by caving 5.56 tons of ore are broken per pound of explosive. Of the total explosives consumed, 13 percent is used in development headings and stope preparation, 16 percent in blasting the powder drifts, and 71 percent in bulldozing.

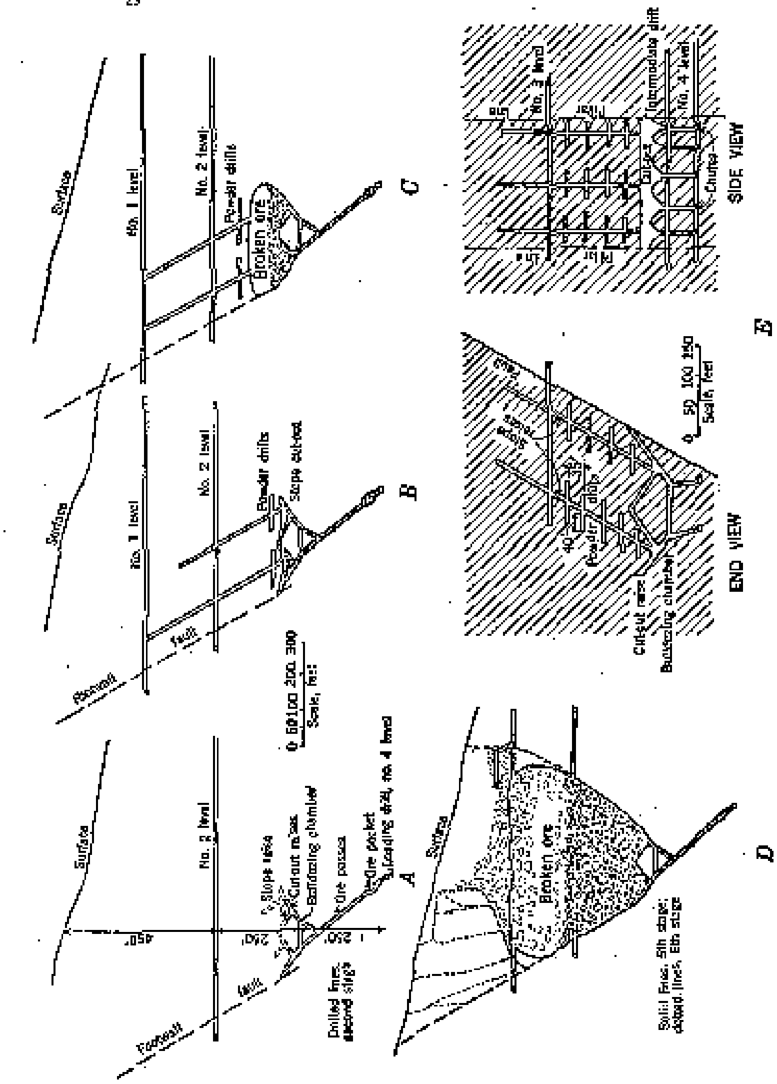

FIGURE 34.—Sequence of stoping operations, Alaska Juneau mine, Alaska: *A*, First and second stages; *B*, third stage; *C*, fourth stage; *D*, fifth and sixth stages; *E*, fully developed stope, north ore body ready for powder-drift blasting.

Development work, bulldozing, loading, tramming, and all preparatory mining work are done by contract, and the wages paid to contractors and their men comprise 75 percent of the total underground pay roll. Stope cut-outs, which are 7 feet high, are also on contract, and the price ranges from 35 to 40 cents per square foot, slope measurement. A bonus is paid to all men not included in the development or bulldozing and tramming contracts.

Direct stoping and bulldozing costs for 1928 were as follows:

Direct stoping and bulldozing costs, Alaska Juneau mine, 1928

Stoping:	Cost per ton of ore trammed	Bulldozing:	Cost per ton of ore trammed
Labor	$0.0016	Labor	$0.0601
Power	-------	Power	.0019
Explosives	.0069	Explosives	.0430
Other supplies	.0005	Other supplies	.0079
Total stoping	.0090	Total bulldozing	.1129
		Total stoping and bulldozing	.1219

Development in ore, most of which was stope preparatory work, cost $0.0611 per ton of ore trammed, making the total mining cost (except tramming) $0.1830. Tramming cost was $0.1136 per ton, thus the total underground cost was $0.2966 per ton. Total underground cost per ton was $0.2869 for 1930.

Labor performance in stoping and stope development in man-hours per ton was as follows for 1928:

Labor performance, stoping and stope development, Alaska Juneau mine, 1928

	Underground crew		All labor charged to mining	
	Man-hour per ton	Tons per man-shift	Man-hour per ton	Tons per man-shift
Development	0.030	267	0.048	167
Stoping	.001	8,000	.003	2,667
Bulldozing	.064	125	.083	96
Total chargeable to stoping	.095	84	.134	60
Tramming	.064	125	.132	61

The foregoing figures—that is, $0.183 per ton, 0.095 man-hours per ton, and 84 tons per man shift—are on approximately the same basis and cover the same work as the figures on direct stoping for the mines discussed previously. It is apparent from the cost of labor and explosives and from the man-hours per ton for stoping in the above tables that a large percentage of the ore must have been caved.

BEATSON MINE, LATOUCHE, ALASKA

The system employed at the Beatson mine is much like that at the Alaska Juneau mine. Instead of blasting down the backs of the stopes by large charges placed in powder drifts, long drill holes are fanned out from benches in raises, as shown in figure 35. The system has been described by Presley,[53] from whom the following has been abstracted:

The ore bodies are mineralized portions of a well-defined shear zone in slate, schist, and graywacke. The ore body is roughly lenticular in form, is 800 feet long, and has a maximum width of 340 feet. The hanging wall is clearly defined in the south end of the mine by the Beatson fault. There is no well-

[53] Presley, Bevan, The Latouche System of Mining as Developed at the Beatson Mine, Kennecott Copper Corporation, Latouche, Alaska: Trans. Am. Inst. Min. and Met. Eng., vol. 76, 1928, pp. 11–52.

defined footwall. The stringers and bunches of ore become less frequent until the ground is too low grade for profitable mining. Innumerable minor faults and slips run through the ore body in all directions and are invariably accompanied by talc and clay seams, making the ground treacherous and necessitating close timbering in the permanent openings. In that portion of the ore body where the slates and schists are mineralized the ground is soft; where the graywackes are mineralized it is hard and breaks in large blocks. The principal mineral, chalcopyrite, is associated with pyrite, pyrrhotite, and quartz, and occurs in masses, veinlets, and disseminated particles.

The ore body was laid out for mining with alternate stopes and pillars across the general strike. The width of the stopes was 70 feet and that of the pillars 30 feet. In order to prepare the stopes and pillars for mining operations under this system it is necessary to develop three levels; an upper level from which actual mining is carried on, an intermediate level on which the stopes are prepared and undercut and which is used later as a grizzly level, and a lower level for haulage.

Figure 35, *A*, is an ideal vertical section parallel to the strike showing the levels, branch raises, stopes, and pillars. Figure 35, *B*, is a section of stope 202, showing all preparation work completed and the stope undercut. Figure 35, *C*, pictures the same stope after 2 months of mining, and figure 35, *D*, after 6 months of mining. Figure 35, *E*, shows a plan and vertical section of a typical long round used in blasting a raise bench in hard ground. It will be noted from figure 35, *A*, that two lines of mining raises are put up in each stope from which the ore is blasted out to the pillars and between the raises, whereas only one line of raises is required in the pillars. The mining raises are put up from the undercutting raises parallel to the dip of the ore (55° to 70°) and are 5 by 5 feet in section. The raises are spaced 40 to 60 feet apart horizontally from foot to hanging wall.

The stopes are undercut by widening out the undercutting raises laterally from each grizzly until they break through in the center of the stope to the corresponding undercut from the raises on the opposite grizzly. On the pillar side of the raises widening is carried to the pillar line. Usually a temporary pillar about 10 feet thick running from apex to apex is left above each pair of opposite grizzlies. These pillars are drilled but are not blasted until the stope has been undercut completely. After a stope is undercut, except for these small pillars, the back of the stope is a series of more or less regular rills. If the rills are very solid it is necessary to drill and blast them at the same time the small pillars are blasted.

Raise mining is begun in the raises at the footwall end of the stope. Coming down from the mining level to a point 30 to 40 feet above the back of the stope cut-out temporary staging is installed in the raise, and holes are drilled downward in the sides, back, and foot to a depth of 8 or 10 feet. When these are blasted a bench is formed around the raise. Slabbing is continued if necessary until a bench 18 to 25 feet long and 12 to 14 feet wide is formed, with the mining raise in the center. The floor of this bench is 20 to 30 feet above the back of the stope cut-out. The long rounds (fig. 35, *E*) are then drilled. Steels up to 24 feet long are used, and 35 to 40 holes are required in very hard ground, whereas 20 to 26 holes are needed in softer or more broken ground. The holes are sprung 1 to 3 times to form powder pockets at the bottoms. They are then loaded with 40-percent dynamite, using 500 to 1,000 pounds per round, and the back of the stope is blasted down using electric detonators. Two miners constitute the crew in each raise. From 14 to 17 days are required to stage up, drill, spring, and blast the short round; clean off the bench; and bar down, drill, spring, and blast the long round. In blasting 379 complete rounds 396,091 pounds of explosive were used, breaking 1,618,212 tons of ore. The explosive used averaged

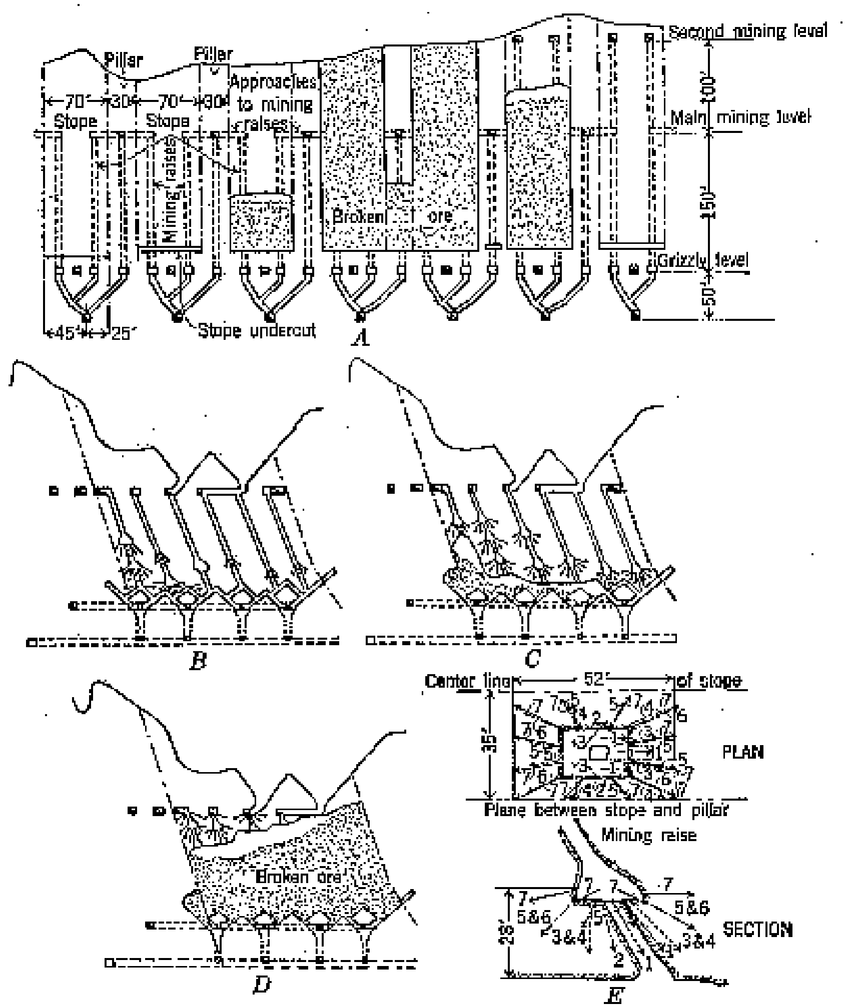

FIGURE 35.—Mining system employed at Beatson mine, Latouche, Alaska: *A*, Ideal vertical section parallel to strike showing levels, branch raises, stopes, and pillars; *B*, section of 202 stope, preparation completed and stope undercut; *C*, section of 202 stope after 2 months mining; *D*, same stope after 6 months mining; *E*, plan and section of typical long round in raise bench in hard ground.

1,045 pounds per round and 0.245 pound per ton of ore broken. Each round broke an average of 4,275 tons of ore. In breaking this tonnage 8,158 machine-drill shifts were required; thus 198 tons were broken per machine-shift, or 99 tons per man-shift drilling.

In undercutting the pillars, which is not done until the stopes on each side have been mined to the top, the undercutting raises are widened out 10 feet on each side of the center line, leaving a 5-foot pillar between the broken ore in the stopes and that in the raises. Generally this breaks down with the first raise blast above.

Costs of breaking 119,868 tons in benching, 1,618,212 tons in stopes, and 501,741 tons in pillars were as follows:

Breaking cost per ton, Bentson mine

	Benching	Stopes	Pillars	Average breaking cost, all ore
Tons	119,868	1,618,212	501,741	2,239,821
Labor, miners and helpers	$0.067	$0.054	$0.045	$0.0537
Timber and supplies		.002	.002	.0020
Explosives	.049	.030	.031	.0378
Compressed air, drill repairs, steel and steel sharpening, air- and water-line installations	.032	.039	.030	.0367
Miscellaneous and general expense	.056	.027	.029	.0278
Total	.204	.151	.137	.1580

The cost of bulldozing 1,682,079 tons of ore drawn from benching, stopes, and pillars was as follows:

	Cost per ton
Bulldozers	$0.083
Explosives	.040
Compressed air, drills, steel, etc	.028
Repairing chutes and grizzlies, labor	.008
Repairing chutes and grizzlies, supplies	$0.006
Miscellaneous and general mine expense	.043
Total	.208

The cost of preparation for stoping was as follows:

	Cost per ton
Stope preparation	$0.146
Pillar preparation	.042
Stope development	.046
Ventilation raises	$0.003
Total cost of stope preparation	.237

The total direct stoping cost is summarized as follows:

	Cost per ton
Breaking in benching, stopes, and pillars; average	$0.158
Bulldozing	.208
Stope preparation	$0.237
Total direct stoping cost	.603

Loading, tramming, and hoisting cost $0.137 per ton; preliminary development, prospect raises, and cuts, $0.044 per ton; and diamond drilling, $0.009 per ton. Thus the total mining cost was $0.793 per ton of ore delivered, and the direct stoping cost was 76 percent of the total mining cost.

Wage rates were as follows:

	1922	1923	1924–26
Miners and timbermen	$4.60	$5.40	$4.00
Miners' helpers and shovelers	4.00	4.75	4.25
Grizzlymen	4.50	5.40	4.60
Ore-pocket, grizzly helpers	4.25	5.00	4.50
Steel sharpeners	4.75	5.65	5.15
Steel sharpeners' helpers	4.00	4.75	4.90

SUMMARY OF SHRINKAGE STOPING

1. Applicability.—Shrinkage stoping is applicable to the mining of tabular deposits ranging in thickness from minimum stoping width to several hundred feet, dipping at an angle steeper than the angle of repose of the broken ore, and composed of firm ore enclosed within firm wall rocks. The ore itself must be strong so that it will support itself over the back of the stope under which the men work, and if the walls are not firm considerable dilution of the ore will take place, especially when the stopes are drawn. The method is well adapted to undercutting and isolating blocks of ground by boundary stopes in block-caving systems of mining. As in the Alaska-Juneau and Beatson mine variations of the method it may be employed to mine large ore bodies by partial caving induced by large blasts. The usual or conventional shrinkage-stoping system finds its best application in the mining of steep, tabular deposits of moderate thickness, having strong, regular walls and containing few waste inclusions in the form of bands or lenses. Horses of waste may be mined around and left in as pillars if they are large enough to stand after the ore is blasted away from them, but numerous small pillars may cause the broken ore to hang up in the stopes, especially if the ore is thin or the dip is not very steep. In mining wide deposits by this method it usually is necessary to drive the stopes across the ore body and to limit their width so that they will stand well without support, leaving pillars of ore between the stopes.

2. Flexibility.—The shrinkage-stoping method lacks the flexibility of other supported-stope methods. There is no opportunity to sort out waste in the stopes, but if a grizzly and double chutes are installed, as in figures 29 and 30, some coarse waste may be sorted out and thrown into separate cars on the level. There is little opportunity to follow tongues of ore out into the walls and to probe for parallel ore bodies, since this usually involves breaking considerable waste, which there is no opportunity to gob in the stope. If the ore is irregular in strike or dip, sharp projections of wall rock into the stope usually will break off and dilute the ore. If the dip is very irregular the rolls in the footwall form benches on which the broken ore will hang in drawing the stope. Dislocations in the deposit which offset blocks of ore over the lower portions of the stopes are apt to result in caving of large slabs of wall rock between the offset blocks and the stope below, causing undue dilution and making the stopes unsafe. Therefore, the method is not well adapted to mining deposits containing inclusions of waste, to those having very irregular walls or frequent tongues of ore in the walls, or to veins which are offset frequently by small longitudinal faults. A large proportion of the broken ore (usually about 60 to 65 percent) must be left in the stopes until they have been mined to the top. When they finally are drawn the ore must be taken as it comes, and there is no possibility of varying the grade of ore drawn from a given stope from day to day to make a desired grade for the mill. Control of grade for this purpose can be obtained only by drawing from a number of stopes of known average grade, and sometimes it may be necessary to have a number of stopes for grading purposes, whereas one stope would provide the same tonnage if worked and drawn as rapidly as possible.

3. *Recovery.*—If the deposit is very regular in dip and strike and has exceptionally firm walls almost all of the ore can be extracted by shrinkage stoping. Where there are numerous rolls in the footwall, ore may be left on the wall due to its being covered by broken ore during the working of the stope. If the walls are exceptionally strong such ore may be recovered later, when the stope is being drawn empty. If there are frequent offshoots from the main ore body into the walls the ore contained therein is apt to be left behind and lost, although much of this may be recovered subsequently while the stope is being drawn down if the walls are firm. Where the stopes are carried on pillars over the drifts much of the ore in these pillars may be lost, especially if the ore and walls have been weakened by mining or by exposure to the air. The same may be said of floor pillars under the levels. If the ore is of low or medium grade, pillars sometimes are deliberately left between stopes, over the drift backs, and under the levels, and these may contain 10 to 15 percent of the ore. The total tonnage recovered frequently is greater than the estimated tonnage in the deposit and usually is lower in grade than is estimated, due to dilution from the walls caused by irregularities in the walls breaking off and falling on the ore, sloughing of slabs of wall rock loosened by blasting or by air slacking, faulted conditions in the vein, blasting of wall rock in narrow places to make working room, or blasting out offshoots of ore in the wall rocks, which usually is attended by breaking of some waste. Ordinarily about 85 to 95 percent of the total ore is recovered, and dilution customarily amounts to 5 or 10 to 25 percent where the shrinkage method is applied to conditions to which it is suited. Sometimes dilution may run 30 percent or more, in which case some other mining method probably would be better.

4. *Development.*—In narrow ore bodies development usually is simple, and only a small amount of stope preparation is required. Where the stopes are supported on stulls over the drift backs the stope development consists merely in taking down the drift backs, placing the stulls, and installing chutes, with perhaps an occasional raise to the level above for ventilation. Frequently, no raise is put through until the stope has been advanced upward for some distance. In wider veins, where the stopes are carried on regular drift sets, somewhat longer is required to prepare for stoping, but the stopes can be brought into production in a comparatively short time. In mines where the stopes are carried on pillars over the drifts short chute raises must be put up and connected over the pillar by a stope drift, and if grizzlies are to be used further preparatory work is necessary in blasting out chambers, installing grizzlies, and raising from the grizzly chambers to the stope cut-out level. In some instances the ability to begin stoping quickly after the level has been developed has led to the adoption of shrinkage stoping. In most instances this would not be a valid reason for adopting the method, as usually only about 30 to 40 percent of the broken ore can be drawn during mining of a stope; therefore more stopes would have to be in operation at a time to supply the same tonnage that could be obtained from fewer stopes in which the ore is drawn as

fast as it is broken. Once a mine has reached its production stride, however, the deficiency in productive capacity of operating stopes can be met by drawing completed stopes.

Table 6 shows the relation between feet of stope development required and tonnages of ore developed at a number of mines employing shrinkage stoping.

TABLE 6.—*Relation between feet of stope development and tonnage developed, mines employing shrinkage stoping*

Mine	Variation of shrinkage method	Average width of ore, feet	Level interval, feet	Stope development per ton of ore developed, linear foot	Ore developed per foot of stope development, tons
Nevada-Massachusetts (tungsten).	On drift pillars	4.5	[1] 100	0.020	50
Harmony (copper)	On drift pillars with grizzlies over chutes.	6.0	100-150	.020	50
Hillside (fluorspar)	On drift pillars in wide ore; on stulls in narrow ore.	12.0	100	.010	100
Daisy (fluorspar)	On stull timbers	6.5	100-125	.008	125
Eighty-five (copper)	15 percent on drift pillars; 85 percent on stulls.	5.0	100-150	.008	126
Teck-Hughes (gold)	On drift pillars	9±	125	.0133	75
Lake Shore (gold) [2]	do [1]	9±	150	.0163	60
Sylvanite (gold)	On drift timbers	4.0	125	.020	50
Vipond (gold)	Partly on stulls, partly on drift pillars.	8±	100-133-200	.0045	222
McIntyre (gold) [2]	On timbered drifts	10.0	125	.0067	150
Engels (copper)	On drift pillars with pillars between longitudinal stopes.	(3)	150-200	.011	91
Homestake (gold)	Transverse stopes 40 feet wide between pillars 40 feet wide. On sill floor.	135.0	[4] 150	.005	200
Bristol (iron ore) [5]	Transverse stopes 15 to 25 feet wide on drift pillars; between pillars 12 to 22 feet wide. Pillars caved and drawn with shrink.	[6] 140.0	125	[5] .0028	[5] 357
Climax (molybdenum)	Stopes 400 feet long by 50 feet wide separated by 40-foot pillars. Pillars mined by coyote blasts. On pillars with grizzlies.	(7)	100-200	.0025	400
Alaska Juneau (gold)	Breaking and forced caving by large blasts in coyote drifts. On pillars with grizzlies.	2,000+	250	[8] .008 [9] .005	[8] 125 [9] 200
Beatson (copper)	Stopes 70 feet wide between pillars 30 feet wide. On pillars with grizzlies. Stopes mined by drilling large rounds around mining raises.	[6] 850.0	200	.01316	76

[1] On dip.
[2] Before cut-and-fill stoping and square-setting largely superseded shrinkage stoping.
[3] Quite wide.
[4] Below 1,250-foot level.
[5] Based on long ton (2,240 pounds).
[6] Maximum.
[7] Hundreds of feet.
[8] Including coyote drifts.
[9] Not including coyote drifts.

5. *Cost of shrinkage stoping.*—Direct stoping costs at a number of mines which have been described in the foregoing pages are presented in table 7. These costs include stope preparation and stope development but not general mine and level development. They vary between wide limits. The first 13 mines listed are in veins of narrow to moderate width, and the direct stoping costs range from $1.230 to $2.743 per ton, averaging about $1.70 per ton,

The next two mines listed are in somewhat wider ore, and the costs are of the same order—that is, $0.76 and $0.63 per ton, respectively, or an average of $0.70 per ton.

The figures for the Homestake mine do not reveal the shrinkage-stoping costs separately from the costs by the other methods employed there. The figures for the last two mines, the Alaska Juneau and Beatson, are obtained under conditions seldom encountered in other operating mines and with practices quite different from those used in conventional shrinkage stoping.

The principal physical factors controlling these variations in costs, in the order of their importance, are probably: (1) Width and continuity of ore and (2) hardness and toughness of the ore. Variations in wage rates and costs of explosives, timber, and other supplies between the different mines, as well as differences in physical conditions, make it impossible to draw further significant conclusions from comparisons of the costs.

The costs of shrinkage stoping in man-hours per ton at a number of mines are given in table 8 and consumption of explosives per ton of ore stoped in table 9.

ADVANTAGES OF SHRINKAGE-STOPING METHOD

The following advantages of the shrinkage-stoping method may be noted for mining bodies of strong ore enclosed within firm walls.

1. Little preliminary work is required in the conventional system to prepare the stopes for production.
2. The broken ore in the stopes provides a floor close to the back of the stopes upon which the men work. It also supports the walls during active working of the stopes.
3. Stopes can be worked rapidly, as there is little or no time lost in timbering in the stopes. In large stopes in good ground a number of crews can be worked in each stope.
4. Long ore passes do not have to be built and maintained from the level to the back of the stope, as the ore is drawn through a number of chutes or short raises at the drift level.
5. Tramming or wheeling ore in the stopes is not required.
6. A considerable reserve of broken ore is held in the stopes which can be drawn upon to maintain an even supply of ore to the mill, even though breaking may be at an irregular rate. (See disadvantage no. 5, p. 136.)
7. Large chunks of ore can be bulldozed on top of the broken ore pile in the stopes. (See disadvantage no. 7, p. 136.)
8. Low stoping costs, varying inversely as the width of the ore, are possible.

DISADVANTAGES OF SHRINKAGE-STOPING METHOD

Inherent disadvantages are as follows:

1. The method has frequently been applied to deposits having walls which slough to such an extent that excessive dilution of the ore takes place and clean mining is impossible. This may be considered a misapplication of the method rather than a disadvantage, but in borderline deposits where dilution is considerable, yet not prohibitive, it is an inherent disadvantage of shrinkage stoping.

TABLE 7.—*Direct stoping costs at mines employing shrinkage stoping*

Mine	Year	Variation of shrinkage method	Direct stoping costs per ton of ore mined: Labor	Supervision	Compressed air, drills, and steel	Explosives	Timber	Other supplies	Total
Nevada-Massachusetts	1928	On drift pillars	$1.062	$0.064	$0.300	$0.306	$0.146	$0.207	$2.743
Harmony	January to June 1929	On drift pillars with grizzlies over chutes	.650	.300	.430	.270	.180	.220	2.050
Hillside	1929	On drift pillars in wide ore; on stulls in narrow ore	.700	.020	.202	.211	.082	.053	1.270
Daisy	1929	On stull timbers	.710	.065	.190	.160	.120	.070	1.315
Cortez	1930	On drift pillars	1.092	.068	.250	.374	.054	.126	1.972
Eighty-Five:									
8 stope	1925	On stulls	.500		.480	.230	.140	.020	1.370
6 stope	1925	do	.750		.460	.270	.200	.020	1.700
26 stope	1928–29	On pillars	.720		.640	.310	.070	.010	1.750
2 stope	1927	On stulls	.400		.480	.240	.100	.010	1.230
Verde Central	June 1929 to August 1930	do	.734	.073	.425	.234	.135		1.601
Teck-Hughes	Typical month, 1928	On drift pillars	.761	.006	.309	.416	.042	.023	1.557
Kirkland Lake Gold	1930	On drift timbers	[1] .770	[1] .079	[1] .230	[1] .459	[1] .030	[1] .050	[1] 1.620
			[2] .671	[2] .061	[2] .200	[2] .400	[2] .027	[2] .053	[2] 1.412
Vipond	August 1929 to July 1930, inclusive	Partly on stulls; partly on drift pillars	[1] .526	[1] .064	[1] .250	[1] .380	[1] .016	[1] .039	[1] 1.275
			[2] .580	[2] .091	[2] .271	[2] .305	[2] .050	[2] .097	[2] 1.394
Elkoro	1930	On stulls	.382	.537	.103	.107	.164	.259	1.610
Mogollon, N. Mex.[3]	1922	On drift pillars	.681	.134	.340	.427	.070	.076	1.728
Sylvanite	3 months, 1930	On stulls	1.315	.105	.408	.410	.093	.027	2.358
Engels	1928	On drift pillars, with pillars between longitudinal stopes	.378	.033	.135	.157	.017	.040	.760
Mount Hope:[4]									
Stope development	1930	On drift and stope pillars	.106	.004	.110	.042	.012	.024	.298
Stoping	1930	do	.074	.029	[6] .090	.018		.012	.223
Blockholing	1930	do	.043	.003		.048		.007	.101
Total			.223	.036	.200	.108	.012	.043	.622
Homestake[5]	1929	Transverse stopes 60 feet wide between 40-foot pillars. On sill floor. Pillars mined by square-setting.	.675	.066	.188	.094	.120		1.143
Alaska Juneau	1928	Breaking and caving forced by large coyote blasts. On pillars, with grizzlies.	.0617		[6] .0019	.0499		.0084	.1219
Bralorne	1922–29, inclusive	Stopes 70 feet wide between 30-foot pillars. On pillars with grizzlies. Stopes mined by drilling large rounds around raising raises.	[7] .1447		.0817	.0773	.0020	.0769	[7] .3680; [8] .2370

[1] Cost per ton of ore hoisted.

[2] Cost per ton of ore broken.

[3] Kidder, S. J., Mining Methods in Mogollon District, N.Mex.: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 529–549; distribution of costs estimated by author from this article.

[4] Long-ton (2,240 pounds) basis.

[5] Average of all stoping, including back filling; 20 percent shrinkage stoping, 25 percent square-setting, 55 percent caving.

[6] Power.

[7] Includes benching in stope and pillar raises, also bulldozing.

[8] Stope preparation.

TABLE 8.—*Costs of shrinkage stoping in man-hours per ton*

Mine	Year	Variation of shrinkage stoping method	Man-hours per ton of ore				Tons of ore per man-shift
			Breaking	Timbering	Shoveling	Total	
Nevada-Massachusetts	1928	On drift pillars	$0.700	$0.376	[1] $0.383	$1.459	5.6
Harmony	January to June 1928	On drift pillars with grizzlies over chutes	1.100		.150	1.250	6.4
Hillside	1929	On drift pillars in wide ore; on stulls in narrow ore				.906	8.6
Daisy	1929	On stull timbers				1.230	6.2
Cortez	1929	On drift pillars	.901	.209	.475	1.585	5.0
Verde Central	June 1929 to August 1930	On stulls	.720	.350		1.060	7.6
Teck-Hughes	October 1928	On drift pillars	1.400	.117		1.517	6.8
Do	Sept. 1, 1928, to Sept. 1, 1929	do				1.312	6.1
Kirkland Lake Gold	June 1930	On drift timbers	.900	.070	.490	1.460	6.5
Vipond	Aug. 1, 1929, to July 31, 1930	Partly on stulls; partly on drift timbers				1.170	6.8
Elkoro	1930	On stulls	.555	.818		1.373	5.5
Mogollon, N. Mex	1922	On drift pillars				.740	10.8
Sylvanite	1929	On stulls				1.760	4.7
McIntyre	1930	On drift timbers	.400				
Lake Shore	1929	On drift pillars				.659	12.0
Engels	1929	On drift pillars with pillars between longitudinal stopes				.598	13.4
Walker	1929					.215	37.2
Isle Royal	1928	On stulls				.360	22.2
Mount Hope	1930	On drift and stope pillars				[2] .113	[2] 60.0
Homestake	1929	Transverse stopes 60 feet wide between 40-foot pillars. Pillars mined by square-setting.				[3] .475	[3] 11.6
Bristol	1929 and 1930	Transverse stopes on drift pillars; pillars caved and drawn with shrink.				[2] .166	[2] 51.6
Climax Molybdenum	1929	Stopes 50 feet wide between 40-foot pillars. Pillars mined by coyote blasts.				.122	48.4
Alaska Juneau	1929	Breaking and caving forced by large coyote blasts				.086	93.0

[1] Includes some tramming.
[2] Per long ton of 2,240 pounds.
[3] Includes shrinkage stoping, square-setting, and caving.

TABLE 9.—*Consumption of explosives per ton of ore broken in shrinkage stoping*

Mine	Kind and grade of explosive used	Explosive used per ton of ore broken, pounds
Nevada-Massachusetts	Gelatin dynamite, 35 percent	1.05
Harmony	do	2.65
Hillside	Ammonia, 35 percent	1.50
Daisy	Special gelatin, 30 and 35 percent	3.00
Cortez	Gelatin dynamite, 40 percent	2.68
Verde Central	do	1.10
Kirkland Lake Gold	do	2.54
Vipond	do	1.05
McIntyre	do	.77
Elkoro	do	1.24
Mogollon, N. Mex.	do	1.58
Sylvanite	do	3.27
Lake Shore	do	.83
Engels	Gelatin dynamite, 25 percent	.128
	Gelatin dynamite, 40 percent	.293
	Gelatin dynamite, 60 percent	[1].035
	Gelatin dynamite, 60 percent	[2].047
Mt. Hope	Gelatin dynamite, 40 and 50 percent	[3].74
Homestake	Gelatin dynamite, 40 percent	[4].50
Bristol	Extra dynamite, 40 percent, and gelatin dynamite, 60 percent	[3].40
Climax	Gelatin dynamite, 40, 50, and 60 percent	.246
Alaska Juneau	Ammonia, 40 percent	.34
Benson	Gelatin dynamite, 40 percent	.315

[1] Blasting hammer drillholes in pillars.
[2] Blasting diamond drillholes in pillars.
[3] Per ton of 2,240 pounds.
[4] Including shrinkage, square-setting, and caving.

2. Although rapid development of stopes for mining is possible, permitting a mine or level to get into production quickly once the level development is well under way, only 30 to 40 percent of the ore in each stope is available for removal as it is broken, the balance being tied up until the stope has been completed. This ties up working capital in breaking ore which cannot be turned into cash for some time.

3. Except in deposits having exceptionally firm walls, some dilution of the ore with waste is bound to occur.

4. Although long ore passes need not be built and maintained, as in some other methods of stoping, chutes usually must be spaced closely, as otherwise the broken ore often will hang up between the chutes, preventing its being drawn down evenly. If chutes are spaced widely considerable shoveling may be required to clean out the stopes, and this may be dangerous, or at best expensive, if the walls are not exceptionally good.

5. Although reserves of broken ore may be accumulated and employed to insure a steady supply of ore to the mill prolonged storing of ore in the stopes is detrimental to good recovery in later concentration operations with some types of ore. Some sulphides acquire a partly oxidized film soon after they are broken, which inhibits their recovery in flotation plants.

6. No opportunity is afforded for sorting ore from waste in the stopes.

7. Although large chunks of ore on top of the ore pile can be bulldozed in the stopes, there is often an element of danger in working out under the back, especially in large stopes. Furthermore, large chunks often are covered with finer muck, and their presence

is not revealed until the chunks arrive at the chutes, where they give trouble.

8. Serious, even fatal, accidents often have occurred due to broken ore hanging up in the stopes, followed by its sudden collapse under men working in the stopes. In other instances men have been drawn down into the ore owing to mistakes of the trammers in pulling chutes under sections of the stopes while miners were working. Such accidents, while due to carelessness, nevertheless are inherent in the stoping method.

9. Travel and handling of drills, steel, and drilling gear in the stopes often are difficult and consume considerable time which could better be employed in drilling, particularly if the top of the broken ore is uneven, as often happens.

CUT-AND-FILL STOPING

Cut-and-fill stoping is employed for mining tabular deposits of high- or medium-grade strong ore, dipping at angles steeper than the angle of repose of the broken ore and having a wall or walls too weak to stand more than a short time without permanent support, even over comparatively small spans. It also is employed for mining thick deposits of large horizontal area, which may range from regular to very irregular in outline, by means of a series of cut-and-fill stopes which usually are separated by vertical pillars of ore. It is used in veins which are thinner than practicable stoping widths, requiring the removal of wall rock to provide room to work in the stopes. When the vein and walls are broken separately the method is termed "resuing" or "stripping." In resuing, one wall may be shot down first and left in the stope for fill, after which the vein is stripped from the side of the excavation thus made, or the vein may be shot down first, the broken ore removed, and the wall then stripped off to make working room and to fill the stope below. Cut-and-fill stoping also may be used where the strength of ore and wall rocks would permit the use of shrinkage stoping but where inclusions of waste in the ore make sorting of considerable broken ore and waste in the stopes desirable, or where it is frequently necessary to mine stringers of ore out into the walls of the stopes. Since the filling in cut-and-fill stoping does not directly support the back but the walls only, the ore itself must stand well for a short time at least. If the ore itself and one or both walls are weak square-set stoping or a caved-stope method usually would be preferable to cut-and-fill stoping. Stoping costs are higher per ton of ore in cut-and-fill stoping, other factors remaining the same, than in open stoping or shrinkage stoping owing to the cost of handling and spreading the filling material; therefore higher-grade material is required to make ore when this mining method is used.

LONGITUDINAL CUT-AND-FILL STOPING IN TABULAR, STEEP-DIPPING DEPOSITS OF SMALL OR MODERATE WIDTH

In tabular, steep-dipping deposits of narrow or moderate width the ore usually is mined by longitudinal stopes in which the ore is stoped out from wall to wall. The longitudinal stopes may be carried the full length of the ore shoots or may be limited in length, to reduce the area of the stope back, by separating the individual stopes from each other by pillars of ore. As in shrinkage stoping the stopes may be carried either directly upon timbered-drift backs or upon pillars over the back of the drifts. The ore is stoped by horizontal or inclined cuts from the bottom of the ore block upward.

BLOCK P MINE, HUGHESVILLE, MONT.[53]

The ore occurs in a crescent-shaped vein in a large syenite chimney or stock cut by rhyolite dikes. The ore minerals are galena, pyrite, sphalerite, and marmatite. The gangue is principally altered syenite and rhyolite with some calcite, barite, marcasite, quartz, and rhodochrosite. The mineralization is banded and is distributed in lenses throughout the vein. The vein is 1 to 4 feet wide and dips 65° to 88°. The ore breaks easily, so that 95 percent will pass an 8-inch grizzly as blasted in the stopes. The walls stand well for the short time that they are required to remain unsupported, except in rhyolite, where stulls are required to support blocky ground.

The level interval is 100 feet from the collar of the shaft to the 400-foot level; below this level the interval is 200 feet. The levels are developed by drifts along the vein, 6 feet wide by 8 feet high; they are timbered with regular drift sets. In preparation for stoping, through-raises to the level above are driven 690 feet from the shaft and at 690-foot intervals thereafter. These raises improve ventilation and permit exploration of the ground between the levels. When a stope is begun the lagging is removed over the drift sets and a 5-foot cut is blasted out of the back of the drift along the full length of the stope section, the ore being shoveled into cars by hand. The lagging is replaced, and chutes are installed on 46-foot centers with provision for a manway alongside each chute. The stopes are advanced as a single face by horizontal slices the entire length of a stope section.

Selective breaking and hand sorting are required to obtain a high-grade product and to prevent loss of ore in the waste filling. The waste more than fills the stopes. When ore and waste occur together in alternate bands across the vein they are blasted together and the waste is sorted out by hand. Where the ore occurs as a single strong band in the vein the waste is blasted first and then the ore (resuing). Where the ore is weak and in a single band it is blasted first and then the waste (also resuing). Before the ore is blasted a floor is laid on the fill to prevent mixture of ore and waste. This floor consists of 5-foot lengths of 3- by 8- to 10-inch plank. The ore is shoveled by hand into the chutes for a maximum of 20 feet. The waste is thrown aside for filling and leveled off by hand. A stope crew usually consists of two men, a miner and a shoveler. The miner drills on one side of a chute one day and on the other side the next day, and the shoveler works on the opposite side. Holes are drilled in the back with hand-rotated stopers; a miner drills an average of 15 holes per day. The holes are inclined slightly to throw the ore toward the chute as much as possible. Holes are shot with 30-percent gelatin dynamite. Stopes are carried through to the level above. About 5 feet below the level stulls are placed at

[53] Vanderburg, Wm. O., Mining Methods at the Block P Mine of the St. Joseph Lead Co., Hughesville, Mont.: Inf. Circ. 6416, Bureau of Mines, 1931, 14 pp.

5-foot intervals, lagged over, and covered with waste to support the track. At the end of a stope section the fill is lagged off from the solid ore by a vertical row of stulls set on 5-foot centers and lagged with 3- to 5-inch poles. Where there is a horse of waste in the vein the ore passes are raised through the waste to the bottom of the ore above, and inclines are driven from this point to the adjoining raises. Rock walls are built around the top of the chute raises, and when these are high enough another set of raise timber is installed inside the rock wall. The timbering is done by a regular timber crew of two men.

The advantages of this method at Hughesville are as follows: (1) The ore is completely extracted; (2) little timber is required; (3) the broken ore remains in the stopes only a short time; (4) a high-grade product is obtained by careful hand sorting; (5) good ventilation is obtained; (6) safe working conditions are afforded; and (7) the fire hazard is small.

Due to the narrowness of the vein and the necessity of hand sorting in the stopes and breaking the wall in many places to make working room, output per man is necessarily low and stoping costs are correspondingly high.

Vanderburg gives the following figures on productivity of stoping labor for 1929, when 106,242 dry tons of ore were mined and hoisted:

Productivity of stope labor, Block P mine, 1929

Occupation:	*Man-hours per ton*	*Tons per man-shift*
Breaking	1.636	4.9
Shoveling (and sorting)	1.812	4.4
Timbering	.508	15.7
Total	3.956	2.0

Wages were as follows: Miners, \$5.50; shovelers, \$5.00; timbermen, \$5.50; and timbermen helpers, \$5.00.

On the basis of the above figures, stope labor cost about \$2.60 to \$2.75 per ton of sorted ore hoisted. Consumption of explosives was 2.568 pounds per ton in stoping.

QUESTA, N. MEX.[54]

Narrow veins containing molybdenite as the only mineral of economic value occur in wide, east-west fracture zones and consist of groups or sets of closely spaced, branching, and interfingering fractures. Barren north-south fractures offset the veins in many places. The space between the small, branching quartz-sulphide veins is occupied by altered, sericitized, porphyry country rock, and the vein filling comprises only a small part of the ground broken in mining. The veins are commonly 200 to 500 feet long, and the largest single stope is 240 feet along the strike and 170 feet on the dip; the average stope is not over half this size. The dip of the veins ranges from 20° to 90° and averages about 60°. The width of mineralization ranges from 1 inch to 6 feet and averages 12 to 18 inches. Relatively wide stopes have been worked on veins averaging 4 feet in

[54] Carman, J. B., Mining Methods of the Molybdenum Corporation of America at Questa, N. Mex.: Inf. Circ. 6514, Bureau of Mines, 1931, 15 pp.

thickness. The vein material and wall rocks drill and break easily. Drifts along the veins usually stand well and are not timbered before stoping is begun; about one-fourth of the drift footage is in badly broken ground requiring support. Stulls are used for temporary support in some stopes, especially on the flatter dips. The ground near the veins is not heavy, and the chief difficulty in mining is caused by the shattered and blocky character of the ground where transverse slips have cut the main fissure. The molybdenite itself is very friable, and numerous fines are produced in blasting unless it is done carefully.

About nine-tenths of the ore mined is extracted by horizontal cut-and-fill stoping. Usually the ground requires the support afforded by filling. Other factors that favor the use of the cut-and-fill method are (1) the frequent necessity of breaking more than the width of the ore to provide working room and (2) the few very long tramming distances from stope to surface ore bin which make handling of waste outside the stope expensive.

The vertical interval between levels on the same vein is 40 to 100 feet. A cut-and-fill stope is begun by taking a cut out of the back of the drift, placing sets and lagging, and installing chutes at 50-foot intervals with manways at every other one. Although ventilation or prospect raises sometimes are driven in ore from level to level, which later are used in stoping, no through-raises are driven for stoping purposes alone.

Drilling and blasting account for only a small part of the labor in stopes at this mine. The ground is broken by stoper holes drilled almost vertically into the back. A short length of back is drilled at a time, and the depth of the holes is seldom more than 3 feet. An effort is made to employ hand drilling in soft ground to obtain more complete and cleaner recovery. Machine drilling is done with 89-pound, hand-rotated stopers. Holes are blasted with 40-percent gelatin dynamite.

Where the ore is wide enough to give working room and the walls need not be broken to provide filling for the stope (that is, if enough filling is obtained by sorting), the ore is broken down from wall to wall and the walls are left intact. If, as is more common, the ore is narrower than stoping width, the footwall is broken away from the ore, any ore is sorted out of the waste, and the ore exposed on the hanging wall is then taken down by picking or very light shooting. Loose sections of back are supported temporarily on stulls placed from wall to wall. The fill is leveled, and sometimes a plank floor is laid before the ore is shot. The quantity of waste sorted in the stopes varies greatly. It is estimated that 60 percent of the material broken as ore is rejected by sorting or screening in the stopes and that 10 percent of the remainder is rejected by sorting on the surface.

COLD SPRINGS MINE, BOULDER COUNTY, COLO.

In the Cold Springs mine ferberite occurs in lenses along vein fissures.[55] The lenses range in length from a few feet to a maximum

[55] Vanderburg, William O., Methods and Costs of Mining Ferberite Ore at the Cold Springs Mine, Nederland, Boulder County, Colo.: Inf. Circ. 6678, Bureau of Mines, 1932, 15 pp.

of 200 feet, the average length being about 80 feet. The width of the ore ranges from a fraction of an inch to a maximum of 6 feet; the average width mined is 8 to 10 inches, about 3 inches being the least that can be mined economically when the price of WO_3 is $12 to $13 per unit. In the narrower parts of the lenses the ore consists of high-grade ferberite associated intimately with fine-grained quartz. In the wider parts the ore commonly occurs as a series of stringers with layers of barren country rock between them. The country rock is granite, frequently gneissoid. The average dip of the vein is 70°. Generally the walls of the veins are traversed by a series of intersecting planes of weakness so that they are apt to slough if openings are more than 6 feet along the dip. The ore is fairly soft, so that a lens exposed along one side often can be removed easily by moil and hand hammer.

The first three levels are 50 feet apart measured on the dip, and below that the levels are 100 feet apart. The ore is extracted by horizontal cut-and-fill stoping, much of it being recovered by resuing. The factors determining the use of this method are as follows: (1) The ferberite occurs in narrow, rich veins so that considerable waste must be broken to provide working space; (2) the walls generally are weak and require artificial support which can be provided cheaply by the waste from selective mining and hand sorting in the stopes; (3) in milling, the higher-grade ores are at a premium so that it is desirable to obtain as high-grade a mine product as is consistent with economy; (4) the ferberite is distinguished easily from the rock, and fine breaking is not required to separate ore and waste; and (5) virtually all of the ore can be extracted with adequate supervision.

After an ore shoot has been opened by drifting a stope is begun by taking a cut 5 or 6 feet high out of the back of the drift. The ore is kept separate from the waste by drilling them separately or by blasting them together and hand sorting. After the back is taken down the drift is timbered with 8- to 12-inch round stulls on 4-foot centers at the chutes and 5-foot centers between the chutes, and the stulls are lagged over with 4- to 6-inch round timber between the chutes. As the stopes are short a double-compartment raise at each end is all that is required for access to the stopes. Cribbed ore passes are installed at 25- to 30-foot intervals between the raises. The stope then is advanced by successive 4- to 6-foot cuts the full length of the ore shoot. Where the ferberite is in a single band stripping (or resuing) is employed; the hanging wall first is blasted carefully and drops into the fill. It is leveled, a sheet-iron shoveling plat is laid, and the exposed ore is blasted down carefully with light charges of explosive or, if it is loose enough, is taken down by hand with hammer and moil. Where the ore and waste are mixed too intimately to be removed separately they are broken together on sheet iron and the waste is sorted out. More than enough waste to fill the stope is obtained. Excess waste is handled in the chutes separately from the ore and hoisted to the surface. Of the total material broken in stoping during 1931 about 70 percent was discarded underground, about 15 percent was rejected by hand sorting on the surface, and the balance (15 percent) went to the mill.

Vanderburg[36] gives figures showing the advantages of close sorting and the resulting high-grade material milled. They show that 5.51 tons of waste were discarded for every ton of ore sent to the mill. The cost of sorting was 30 cents per ton of run-of-mine ore, and the total mining cost (except for sorting) was $2.34 per ton of run-of-mine ore. The profit without sorting would have been $2.45 per ton of run-of-mine ore, whereas with sorting the profit was $5.89 per ton.

Direct stoping costs in 1931 follow:

Direct stoping costs, Cold Springs mine, 1931

	Cost per ton of sorted ore recovered	Cost per ton [1] of run-of-mine ore broken
Labor, including sorting	$2.845	$0.437
Compensation insurance	.393	.060
Supervision	.672	.103
Compressed air, power, drills, and steel	1.116	.172
Explosives	.323	.050
Timber	.432	.066
Other supplies	.170	.026
Total direct stoping cost	5.951	.914

[1] Cost per ton of run-of-mine ore equals cost per ton of sorted ore recovered divided by 6.51. All mine labor was paid $4.00 per shift.

Performance of stoping labor follows:

Performance of stoping labor, Cold Springs mine, 1931

Occupation	Man-hours per ton		Tons per man-shift	
	Ore sorted	Run-of-mine broken	Ore sorted	Run-of-mine broken
Breaking	1.24	0.190	6.45	42.09
Shoveling	.61	.094	13.11	85.12
Sorting	3.84	.590	2.09	13.56
Total	5.69	.874	1.40	9.16

Consumption of explosives was 4.48 pounds per ton of sorted ore or 0.69 pound per ton of run-of-mine ore broken.

LUCKY TIGER MINE, SONORA, MEXICO[37]

The ore occurs in fractures in rhyolite and rhyolite tuff of Tertiary age. There are three principal veins about 600 feet apart, all striking nearly north and south and dipping steeply to the west. Individual ore bodies within the veins are irregularly lenticular, usually with the horizontal axis longer than the vertical axis. The lenses range from 500 by 2,000 feet to 10 by 50 feet, and the ore averages 1.7 feet in width. In rare instances the ore is 20 feet wide, but in such cases the veins usually consist of multiple stringers sepa-

[36] Vanderburg, W. O., Work cited, pp. 9–11.
[37] Mishler, R. T., and Budrow, L. R., Methods of Mining and Ore Estimation at Lucky Tiger Mine: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 463–483.

rated by waste. Usually the ore is less than stoping width, requiring stripping (resuing) or breaking of ore and waste together. Stope width has averaged 3.4 feet. The ore, consisting of intergrowths of sphalerite, galena, pyrite, chalcopyrite, tetrahedrite, and stromeyerite in a gangue of kaolinized or silicified rhyolite, has averaged 73 ounces of silver per ton and, as delivered to the concentrator, 40 ounces. In the upper part of the mine the walls are fairly solid, but on the lower levels, where they usually either are kaolinized and soft or silicified and fractured, closely filled stopes or considerable timber is required. Stripping is practiced wherever possible, but where both the vein and the walls are friable loss in the fill is prevented by breaking ore and waste together and sending both to the concentrator. The average grade of clean sulphides is 550 ounces of silver per ton. Dilution in stoping averages 82 percent.

The stoping method depends on the character of the walls and the width of the vein. Shrinkage stoping is used where the walls are firm and the vein is more than 2 feet wide and open stopes where the ore extends only 20 or 30 feet above the level and the walls are firm; however, cut-and-fill stoping is used most, and this method is especially suitable where the vein is narrow or the walls are soft.

Chutes are installed at 50-foot intervals in narrow ore, and the stope backs are carried horizontally. The ore and waste are blasted separately, the ore being blasted onto cowhides to prevent admixture of fine ore with the waste. The broken ore is sorted carefully, and all fines are shoveled into the chutes, the coarse waste being left in the stope. When all ore has been removed the hides are taken up, and enough waste is blasted from the walls to fill the stope. The broken waste is sorted to remove pieces of ore, but some fine ore unavoidably enters the fill. In very rich veins ore and waste are blasted together onto hides to minimize this loss, and all fines are shoveled into the chutes, only the coarse waste being left in the stope. In narrow stopes chutes are formed by two lines of stulls spaced 5 feet apart and lagged on the outside to support the fill. In wide stopes cribbed chutes of 6-inch round timber are carried at 25-foot intervals.

Where the vein is more than 3 feet wide the stopes usually become too wide and dangerous if the fill is blasted from the walls; therefore, diagonal raises are driven to obtain filling material, or development rock is dumped in through the development raises. In this event the chutes at the ends of the stope cannot be employed as ore passes, and wheelbarrows must be used for handling both ore and waste. Flat-back stopes are used only where the ore requires much sorting or the waste can be blasted separately and left in the stope.

Inclined cut-and-fill stopes are employed where the vein is wide and no sorting is required. A single chute is established in the center of each stope, from which rills are extended diagonally upward at an angle of 40° to intersect the development raises at each end of the block. The ore falls on inclined plank floors nailed to stulls. Temporary grizzly timbers over the chutes retain large boulders until they are broken. When vein and walls are fairly firm, slices up to 10 or 12 feet in thickness usually are taken. After the ore is removed the floors are taken up, and waste is run in from the development raises to fill the stope to within 4 feet of the back. Hand drilling is used in fairly soft rock where the ore and waste must be broken separately; machine drills are used where the ore is hard and there is little danger of the waste falling with the ore. In soft and medium-hard ore hand drilling is cheapest. Stoper drills are used for machine drilling.

Direct stoping costs for November 1924 were as follows:

Direct stoping costs, Lucky Tiger mine, November 1924

	Cost per ton of ore	
Hand drilling, company account	$0.418	
Hand drilling, contract	.158	
Machine drilling, company account	.531	
Machine drilling, contract	.003	
Total drilling	1.110	
Labor, timbering and filling	1.142	
Labor, shoveling and sorting	.593	
Total stoping labor		$2.845
Supervision		.407
Compressed air, steel sharpening, tool nipping, pipe lines and drill repairs, including labor and power		1.051
Explosives		.505
Timber		.710
General supplies		.225
Total direct stoping cost		5.743

These costs include also some shrinkage and open stoping.

EIGHTY-FIVE MINE, VALEDON, N. MEX.

Stoping conditions at this mine have been discussed already under Shrinkage Stoping, as most of the ore has been mined by this method. Cut-and-fill stoping was employed, however, (1) where the walls were weak owing to a system of fractures closely paralleling the walls or to kaolinization of the walls; (2) where the vein was wide but (*a*) unmineralized country rock occurred as a wide band within the enclosing walls, or (*b*) there were splits in the vein; (3) where high-grade ore occurred in a narrow (2- or 3-foot) vein; and (4) where a cleaner product was desired.

The inclined cut-and-fill system had been employed in six stopes up to 1931. A two-compartment stull raise was begun at the near limit of the ore body, and later six more raises were started at 140-foot intervals. Sublevel drifts were driven 20 feet above the rail to connect the raises (see fig. 36) and there silled out the full width of the ore. Two successive, 5-foot, inclined cuts next were mined out in both ends of the stope. If the walls were firm the miners worked on top of the broken ore and took one or two additional 5-foot cuts. After the back of the stope had been cleaned thoroughly and barred down the chutes were pulled, and the remainder of the broken ore was cleaned out. If the drift pillars were in high-grade ore they were floored over before filling was run in. Waste from stopes above or from development work then was run in through the two fill raises to within 2 or 3 feet of the back. The angle of repose of the broken ore is 39°, and the back is mined at that angle. When filling was completed 2-inch flooring was laid on 2- by 10-inch sills, and breaking of the back by inclined cuts was continued. While one cut was being filled that on the other end of the stope was being mined. Before each filling the manway and the two chutes were raised. Before mining was begun grizzlies of 8- by 8-inch timber or of 90-pound rails spaced 10 inches apart were placed over the chutes. Hand-rotated stopers were used in overhand slicing and pluggers in brow-slabbing and plugging boulders.

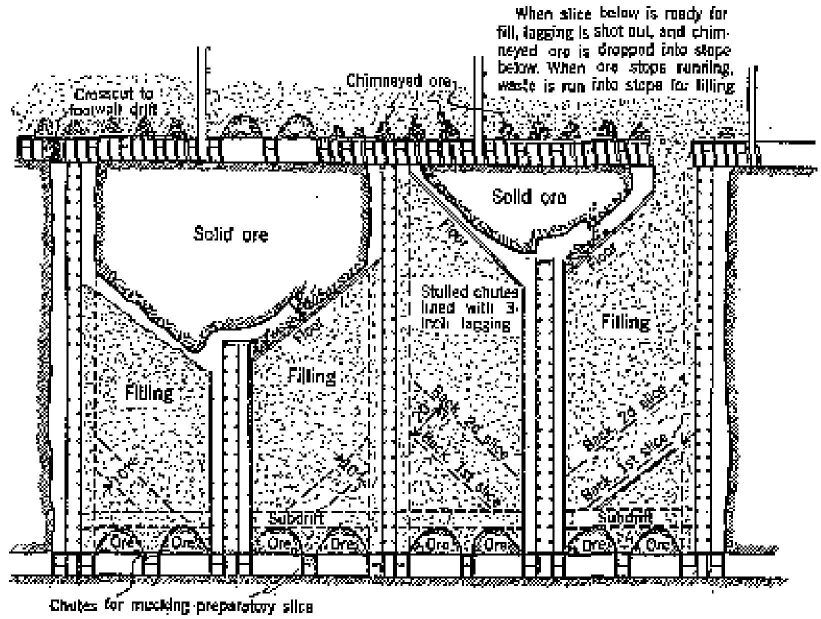

FIGURE 36.—Details of inclined cut-and-fill stoping, Eighty-Five mine, New Mexico.

Where the vein was narrow and the walls were firm several successive cuts sometimes were taken from the top of broken ore, as in shrinkage stoping, before the ore was drawn and filling run in. Typical cut-and-fill stoping costs were as follows:

Cut-and-fill stoping costs, Eighty-Five mine

Stope no.	Years	Costs per ton of ore					
		Labor, including bonus	Air drills	Explosives	Timber	Other supplies	Total
10	1928–30	$1.26	$0.60	$0.28	$0.20	$0.03	$2.37
26	1929–30	1.62	.38	.17	.27	.02	2.48

MCINTYRE MINE, SCHUMACHER, ONTARIO

Mining conditions at this property have been described briefly under Shrinkage Stoping. Cut-and-fill stoping has been employed more and more in recent years, and in 1932 about 50 percent of the output was mined thus. Shrinkage stoping is employed now only in isolated stopes, in stopes where it is anticipated that the walls will be strong and regular, and in low-grade stopes of considerable width.

Cut-and-fill stoping is superseding shrinkage stoping for the following reasons: (1) The walls are irregular; (2) there is less dilution by waste rock in sections with bad walls; (3) waste may be sorted out and left in the stopes; (4) workings are stabilized in bad ground where they must remain open for some time; (5) safety is greater, considering the nature of the ground; and (6) manways may be spaced about 200 to 250 feet apart, fill and ventilation raises 250 to 300 feet apart, and ore chutes 50 feet or more apart.

Through raises are spaced 250 to 300 feet apart and are driven at an angle of approximately 48°, usually without timber. They are wide enough at the start for a standard chute and manway to be built at the level and are reduced to about 6 by 8 feet in the section above. When a stope is begun the level is slashed the full width of the ore, the backs are taken down, and the sill is timbered with standard drift sets or with stringer caps and posts (in wide ore). However, if the back of the drift is in lean ore or waste with ore above the stope is silled out at the bottom of the ore, leaving a pillar over the back of the drift. The stopes are carried horizontally with breasts 8 to 10 feet high, which are drilled with horizontal holes. Ore broken per man-shift, including filling, timbering, and all stope labor, averages 9 tons. Slashing is regulated by drilling test holes in the walls. About 1 pound of 40-percent gelatin dynamite is used per ton of ore broken. Waste for filling is obtained from development headings, from small waste stopes opened in the hanging wall of the stopes, or from two large waste stopes. Waste is trammed from waste raises and spread by hand. Filling and stoping are carried on simultaneously. The ore is broken on plank flooring laid on top of the fill.

Some scraping has been done in cut-and-fill stopes with small scrapers and air hoists for dragging the broken ore into the chutes. In an 8-foot stope in January 1931, four men (2 men per shift) using a scraper averaged 65 tons of ore per day and spread the fill. The scraper also was used to drag the filling from a small hanging-wall waste stope. In this stope chutes were 60 feet apart, and a manway was carried along every third one.

Direct stoping costs in cut-and-fill stopes are not segregated in Bureau of Mines Information Circular 6741, but the average cost of all stoping during 1931, including cut-and-fill stopes and some shrinkage and square-set stopes, was as follows:

Direct stoping costs, McIntyre mine, 1931

	Cost per ton of ore hoisted		Cost per ton of ore hoisted
Labor	$0.891	Lumber	$0.240
Supervision	.038	Other supplies	.088
Compressed air, drills, and steel	.085	Total direct stoping cost	1.489
Explosives	.147		

Costs in units of labor for stoping only were as follows during a typical month in 1932:

	Man-hours per ton	Tons per man-shift
Breaking	1.52	5.26
Timbering and filling	.14	57.12
Total stoping labor	1.66	4.52

GROUND HOG MINE, VANADIUM, N. MEX.[94]

This mine produced about 130 tons of lead-zinc-copper ore per day in 1930 from irregularly shaped lenses in a fault fissure along the contact between diorite porphyry and a granodiorite dike. The ore body is 3 to 25 feet wide, measured normal to the dip, which averages 50°. The ore is a heavy sulphide of little natural strength and generally requires more support than the walls. The dike forming the hanging wall is 15 to 90 feet thick in most places and will stand well if not opened over too great distances; if too large an area is opened under it the ore breaks into large slabs or blocks. The footwall consists normally of a thick gouge which may cause considerable trouble by sloughing away. The worst condition is where the vein has formed in diorite a slight distance from the dike, as the thin shell of diorite then forming the hanging wall is fractured and crushed badly and is difficult to support.

Where the vein is narrow it is mined by cut-and-fill stoping with very little timber; where it is wide filled square-set stoping is employed. In either case stoping is begun by mining out the full width of the ore on the haulage level and placing 10- by 10-inch floor sills from wall to wall on 5-foot centers. Double 2-inch flooring is laid on these sills. In ore more than 10 feet wide square-sets are placed on the sills. If the vein is less than 10 feet wide drift sets are stood on the floor sills with caps from wall to wall. When the sill is completed the first floor above is mined and timbered with square-sets. The sill floor is then filled with waste, except for a line of sets left open for track. Stulled raises on 50-foot centers which had been run up to the level above are then enlarged and timbered with square-sets, one set wide, from foot to hanging wall. The two methods of stoping are similar up to this point.

Formerly flat-back or horizontal cut-and-fill stopes were used in narrow ore, but trouble was experienced because the ore and walls caved. Horizontal slips entering from the hanging wall cut the ore at about 6-foot intervals, and often when a slice was finished and ready for filling the ore caved to the next slip above, bringing down the hanging wall with it. To eliminate this danger and to allow filling closer to the back this method was abandoned in favor of rill stoping. A rill stope is begun by stoping as high a block as possible next to the center raise of a block and then running in fill through the the raise to its natural angle of repose. Floor is laid on the fill and a 3-foot inclined cut is taken out and filled. This process is repeated until the toe of the slope is within 10 feet of the chute, after which the length of the rill is not increased farther, but the stope is carried up with this 10-foot level space maintained as a sorting and shoveling floor. The ore is shoveled into the chute, and the sorted waste is transferred to the adjoining section on the other side of the chute. In filling a slice, as little room as possible is left under the back. Mining is begun at the bottom of the rill and advances upward. With this system the slabs of ore can be held better, as each slip is exposed for a shorter distance, although more slips are exposed in a single cut up the incline than would be in a horizontal cut.

[94] Richard, F. W., Mining Methods and Costs at the Ground Hog Unit, Asarco Mining Co., Vanadium, N. Mex.: Inf. Circ. 6377, Bureau of Mines, 1930, 13 pp.

Direct stoping costs for regular cut-and-fill stoping and square-set stoping combined during January, February, and March 1930 were as follows:

Direct stoping cost, Ground Hog mine, January, February, and March 1930

	Cost per ton of ore mined and hoisted		Cost per ton of ore mined and hoisted
Labor [59]	$1.187	Timber	$0.120
Supervision	.085	Other supplies	.235
Compressed air, drills, and steel	.087	Total direct stoping cost	1.762
Explosives	.058		

Labor in stopes during the same period was 2.140 man-hours per ton of ore or 3.74 tons per man-shift for the two stoping methods combined. The consumption of explosives in stopes was 0.63 pound per ton of ore broken.

MATAHAMBRE MINE, PINAR DEL RIO, CUBA

The ore deposits in the Matahambre mine occur as large, irregular pipes, usually lenticular in cross section and containing primary chalcopyrite, associated locally with quartz and pyrite, in three well-defined fracture zones.[60] The zone in which the most important ore bodies are found trends N. 30° E. and dips 42° to 45° northwest. The other two zones, about 1,200 feet apart and at each end of the main zone, strike northwest and dip steeply northeast. In Matahambre ore bodies the footwall usually is quartzite and the hanging wall thin-bedded shale. The footwall stands well and gives no trouble, whereas the hanging wall will not stand for any length of time if exposed for an appreciable distance and for a height of more than 10 feet. This characteristic of the hanging wall and its rather flat dip are responsible for the use of the cut-and-fill stoping method.

Before 1929 all but three levels were 100 feet apart; two of these were 130 feet apart. It was planned at that time to space new levels 150 feet apart. An ore body is first developed by a drift along its full length. Chutes are installed at 50-foot intervals, and a manway is carried along every third chute. The stope is then silled out 14 feet above the floor of the drift, leaving a 7-foot pillar between the bottom of the stope and the back of the level. The sill is widened to the full width of the ore from end to end of the stope, and the sill cut is usually about 12 feet high. If the stope is very wide cross-cuts are driven from the drift below, from which additional ore chutes are put up. On the first or sill cut all stringers of ore are followed to make sure that the ultimate walls of the ore body have been reached. As there are horses of waste in some places and the ore is cut by slips and fractures it is easy to mistake the walls of slips for the limits of the ore.

All stopes are mined by horizontal cuts, with hand-rotated dry stopers or 48-pound jackhammers. Holes are loaded with 30-percent

[59] Wage rates were as follows: Timbermen $4.50 to $5.00, timbermen helpers $4.00, machinemen in stopes $4.25, machinemen on wet sills $5.00, muckers in stopes $3.35, and muckers on wet sills $4.00.

[60] Richert, George L., Mining Methods at Minas de Matahambre, Matahambre, Pinar Del Rio, Cuba: Inf. Circ. 6145, Bureau of Mines, 1929, 18 pp.

gelatin dynamite. A cut is begun in the center or at the end of a stope with a stoper and carried through with jackhammers. Most of the ore is then slabbed off to the walls by flat holes drilled with jackhammers. A round breaks an average of 8 tons along the sides to 15 tons in the center of the stope if the back is good. All broken ore is shoveled directly into the chute raises or transferred to the raises by wheelbarrows. The ore is hard but breaks comparatively small.

While the stope is being silled out fill raises are run to the level above. These raises were on 80-foot centers when surface waste brought from centrally located glory holes or development waste was used for filling. When sand tailings first were used for filling the fill raises were spaced 100 to 150 feet apart. In the sand-fill system mill tailings are pumped to a 30-foot bowl classifier on the surface near a raise leading from the mine. The slime overflow goes to the tailing pond, and the rake product (or sand) drops into a hopper. Water is added to the hopper, and the course tailings are washed into a 2½-inch rubber-lined pipe strung through a line of raises to the 1,200-foot level and thence through rubber pipe to the stope to be filled. Before the stope is filled the cribbed raises and manways are wrapped with burlap to prevent the sand from washing between the cribbing into the raises. A pipe or hose is employed to direct the sand wherever it is wanted, and by building up small sand dams any desired section of the stope may be filled. The advantages of this system of filling are as follows: No spreading of fill by hand is required; enough filling material always is available; filling can be changed from a stope in one end of the mine to one at the other end by breaking and making a few pipe connections; and the sand enters all cracks and fissures forming a solid filling between the stope walls. Details of the system have been described elsewhere.[61]

Pipe lines have handled 35 tons of sand per hour without difficulty. From February 1929 to June 1931, inclusive, 205,400 tons of sand were put through the pipe without appreciable wear on the rubber lining. During a 6-month period the average cost per ton of sand placed in stopes was as follows: Labor 10.5 cents, burlap 4.8 cents, pipe 7.1 cents, pump 0.6 cent, and power 2.0 cents, or a total of 25 cents. The cost of the pipe covered all new pipe lines installed, and as no replacement was necessary the actual cost per ton of sand placed in stopes probably does not exceed 20 cents.

When a stope has been completely silled out, the first cut taken, and the broken ore removed, a floor of hardwood slabs is laid before the fill is run in. After each succeeding cut is taken and the ore is removed the back is 12 feet above the floor, and a 6- or 7-foot fill is required before the next cut of ore is taken. The contractor in the stope was paid (1928) 20 cents per ton for spreading rock filling by hand and 5 cents per ton for spreading with a scraper.

With one exception no stopes have required square-sets on the first or sill floor. Ordinarily the only timber used in the lower 70 or

[61] Richert, G. L., Filling Stopes with Mill Tailings: Eng. and Min. Jour., Mar. 2, 1929, p. 348.
Hosmer, D. D., Rubber Pipe Lining Minimizes Pulp Abrasion: Eng. and Min. Jour., Oct. 26, 1931, pp. 367–368.

80 feet of the stopes is cribbing for the ore passes and manways. However, in a number of ore bodies the ore has been so fractured that the stopes required square-set timbers. The upper 15 or 20 feet of all stopes are timbered with square-sets, which permits carrying them up past the level and removing the drift pillars under the stopes above.

In 1928 about 30 percent of the ore came from square-set stopes. During that year direct stoping costs were as follows:

Direct stoping cost, Matahambre mine, 1928

	Cost per ton of ore hoisted		Cost per ton of ore hoisted
Labor (mining and filling)	$0.693	Explosives	$0.095
Supervision	.095	Timber	.192
Compressed air, drills, and steel	.085	Other supplies	.099
Power cost	.008	Total direct stoping cost	1.267

Labor performance in stopes was as follows:

Labor performance in stopes, Matahambre mine, 1928

Occupation:	Man-hours per ton	Tons per man-shift
Breaking	0.69	11.59
Timbering and filling	.67	11.94
Shoveling	.92	8.70
Total	2.28	3.51

Consumption of explosives in stopes was 0.44 pound of 10-, 30-, and 40-percent gelatin dynamite per ton of ore mined. Consumption of timber was 0.802 board foot of sawed material and 0.125 piece of prop timber per ton of ore mined.

TEZIUTLAN COPPER MINE, PUEBLA, MEXICO

Cut-and-fill stoping is employed in the Minerva ore body at this mine because of the soft hanging wall.[60] Three other ore bodies with stronger walls are mined by room-and-pillar open-stoping methods. The ore occurs in flat-lying lenticular bodies in metamorphosed igneous and sedimentary pre-Cambrian rocks. Numerous post-mineral faults have caused considerable displacement of the ore. The hanging wall of the Minerva ore body is a soft phyllite. The ore is massive sulphide, the principal minerals being chalcopyrite, sphalerite, pyrite, and galena in a gangue of quartz and, occasionally, mica schist. The Minerva ore body is 890 feet long, 330 feet high, and 5.6 feet wide; the dip is 20° to 65° and averages about 35°. The ore averaged $6.85 in gold and 3.16 ounces of silver per ton, 3.20 percent of copper, 13.82 percent of zinc, and 1.72 percent of lead from July 1930 to June 1931.

Main haulage levels are spaced at 100-foot intervals. In cut-and-fill stoping raises are put up at 100-foot intervals along the level, and

[60] Hezivel, Ernest Ph., Mining Methods and Costs at the Teziutlan Copper Mine of the Mexican Corporation, S. A., Teziutlan, Puebla, Mexico: Inf. Circ. 6780, Bureau of Mines, 1933, 15 pp.

from these the stopes are silled out above the level, leaving a 16-foot pillar over the back of the drift. Alternate raises are driven to determine the upper limits of the ore, usually 66 to 164 feet above the level (dip measurement). The intermediate raises are carried up with the stope to serve as ore passes. One raise is carried through to the level above for ventilation. The sill of the stope is cut out from wall to wall and is 12 feet high. The broken ore is removed, and 7 feet of filling are put in. Succeeding cuts are 7 feet high. The ore is broken by inclined back holes using stopers. Where the stopes are straight enough the broken ore is scraped into the chutes, the scraper hoists being midway between the raises. The scrapers pull first from one side and then from the other, so that while one side is being stoped the other is being filled. Where the ore body is folded and the stopes consequently are crooked wheelbarrows are employed. Filling is handled by shovel and wheelbarrow. In the higher sections of stopes scrapers pull the ore down the chutes, as the dip is too flat for the ore to run by gravity. Waste for filling is obtained by breaking caving stations above the ore body at the top of the waste raises.

By the end of 1931, 90 percent of the output of the Teziutlan mine came from the Minerva ore body, the three other bodies having been virtually worked out. Although some open stoping was done the following costs represent cut-and-fill stoping principally.

Direct stoping cost, Teziutlan mine, July 1930 to June 1931

	Cost per dry ton mined		*Cost per dry ton mined*
Labor	$0.877	Timber	$0.107
Supervision	.041	Other supplies	.078
Compressed air, drills, and steel	.252	Total	1.571
Explosives	.216		

Labor performed in stopes, as measured in man-hours per ton, was as follows:

Labor performance, Teziutlan mine

Occupation:	*Man-hours per ton*	*Tons per man-shift*
Breaking	1.068	7.49
Timbering	.386	20.72
Filling	1.056	7.57
Shoveling	1.788	4.48
Total	4.298	1.86

PECOS MINE, TERRERO, N. MEX.

The Pecos mine is in the heart of the main mass of the pre-Cambrian rocks of New Mexico.[63] An extensive northeast-southwest shear zone has altered the igneous rocks (granite and diorite) into parallel bands of schist. This zone is a few feet to several hundred feet wide and contains ore bodies which are replacements in the schist. The ore minerals are sphalerite, galena, chalcopyrite, and

[63] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Inf. Circ. 6368, Bureau of Mines, 1930. 21 pp.

pyrite carrying appreciable gold and silver. They are associated with talc, hornblende, and mica and chlorite schist. The ore bodies have been developed over 2,000 feet and from the surface to the 1,200-foot level. They consist of irregular, disconnected, often overlapping lenses of sulphides which often pinch out abruptly along the strike and dip. The dip is nearly vertical. Generally the ore and walls are very loose, and close timbering is required. In many places the ground runs freely, and serious caves are apt to follow. If initial movement can be prevented, however, support is not difficult, as the ground neither is heavy nor swells when dry. Water, however, makes it very difficult to support, and the stoping areas must be well-drained.

On levels spaced 100 feet apart vertically timbered drifts are driven along or in the footwall to develop the ore shoots. From the main haulage drifts cribbed chute and manway raises are driven at 25- to 60-foot intervals up to the stope sill, which is about 20 feet above the level. The raises are cribbed with round timber. Short pony sets are erected over the drift sets at the raises. Fill raises from the stope sills to the level above generally are stulled and about 4 by 6 feet in section.

Cut-and-fill and square-set-and-fill stoping were used about equally in 1930. The choice between them is governed by the character of the ore and walls where stoping is begun. Strong ore and walls favor the cut-and-fill method. Often, however, it is necessary to change from one method to the other after the stoping has advanced several floors. The conditions responsible for the selection of these methods at the Pecos mine are as follows: (1) The ore shoots are irregular and discontinuous, and sometimes two to four parallel veins are separated by only 10 to 20 feet of barren loose schist or blocky diorite (occasionally the parallel veins can be mined simultaneously, leaving the waste in place, although usually the waste must be broken and gobbed back in the stope after the ore has been mined); (2) since the veins contain low-grade or barren schist bands and diorite intrusions it must be possible to sort waste in the stopes (at least 20 percent of the stoping ground is sorted and rejected as waste in the stopes); (3) it must be possible quickly to check runs from the walls; (4) it must be possible to prospect the walls for parallel ore shoots at close intervals; (5) gouge 1 to several feet thick on the walls must be supported to prevent runs; (6) the surface must be kept intact, as the veins dip toward the Pecos River which parallels them 400 feet away on the surface; and (7) all complex ore must be milled, and only one class of ore has to be handled.

The inclined cut-and-fill system has been tried and abandoned because of (1) the tendency of ore and walls to cave and run, with consequent flooding of the stope and danger to the workmen below; (2) the difficulty of erecting stulls and cribs on an inclined floor, and (3) the lack of storage space for waste. The horizontal cut-and-fill system therefore has been employed.

Two-compartment raises are driven 12 feet above the back of the drift, and the stope is silled out the full width of the ore at this elevation. The ore is stoped in sections, the lengths of which depend upon the width of the vein and ground conditions. Recent practice in wide-vein areas is to cut the first stopes 30 feet long and 100 to 150 feet apart, with extraction chutes in both ends of each section. After these stopes have nearly reached the level above, new sections about 15 feet long are begun on the ends of the first stopes, using the former extraction chutes for fill raises. Pillars are left only where two stope blocks come together. The stopes are mined by successive, horizontal, 7-foot cuts. Stope ends are timbered off to separate them from the solid by vertical gob posts 5 feet apart and faced with lagging. In narrow ore the stopes are silled several hundred feet long with ore chutes 30 feet apart. Fill raises are driven in ore 30 to 60 feet apart. Stopers are used where possible, but in many places the ground is so loose that it is better to drill flat holes with jackhammers. A shoveling floor of 2-inch plank is laid on the fill before blasting. All shoveling is by hand because of the necessity for sorting. Waste filling is obtained from development headings, from prospect workings driven into the stope walls, from sorting, and from a surface gloryhole.

Direct stoping costs for 1929 were as follows in cut-and-fill stopes only:

Direct stoping costs, cut-and-fill stoping, Pecos mine, 1929

	Cost per ton mined [88]	
Labor:		
Breaking	$0.42	
Shoveling	.44	
Timbering	.41	
Filling	.25	
Total stoping labor		$1.52
Supplies:		
Breaking	.09	
Timber	.26	
Explosives	.14	
Total supplies		.49
Total direct stoping cost		[89] 2.01

The wage scale in 1929 was as follows: Timbermen, $5; miners, $4.50; and shovelers, $3.50.

UNITED EASTERN MINE, OATMAN, ARIZ.

Mining methods and costs from January 1917, when production began, to May 1925, when the known ore bodies were exhausted, have been discussed in detail by Moore.[90] This was a high-grade gold mine from which the average recovery was $19.20 per ton. Seven or eight very productive ore shoots were mined along the Tom Reed-United Eastern vein, which occupies a fault fissure in andesitic lava flows. Maximum dimensions of the Tom Reed extension ore shoot were height 750 feet, length 950 feet, and thickness 48 feet, and 511,976 tons of ore were mined from this body. The maximum dimen-

[88] Costs include distributed accounts, such as compressors, blacksmith, steel sharpening, steel and tools, timber framing, etc., totaling about 25 cents per ton.

[89] Average total direct stoping cost, 1927–29, was $2.11 per ton.

[90] Moore, Roy W., Mining Methods and Records at the United Eastern Mine: Trans. Am. Inst. Min. and Met. Eng., vol. 76, 1928, pp. 56–92.

sion of the Big Jim ore body, from which 220,552 tons were mined, were height 450 feet, length 850 feet, and thickness 35 feet.

Levels were driven at intervals of 100, 150, and 200 feet between the first level at 585 feet and the lowest level at 1,298 feet. Each level was developed by drifts driven to the limits of the ore shoot and beyond, and crosscuts were driven at 50-foot intervals and continued well into the walls at greater intervals to prospect for possible parallel veins.

The horizontal cut-and-fill system was used almost exclusively for mining both large ore bodies. Shrinkage stoping was employed for mining small tonnages in narrow parts of the vein at the lower limits of both ore bodies but was not adaptable to the larger stopes on account of the tendency of the vein to slab off owing to horizontal slips and wall pressure and on account of sheeted, soft hanging wall sloughing off if not supported. Use of the more costly method of stoping was justified by the resulting higher grade of ore milled and the minimum tonnage treated. The method also afforded opportunity to prospect for possible nearby parallel veins by driving wall raises, the waste from which was used for stope filling. Untimbered rill stoping was tried but was not adaptable on account of the tendency of the vein to slab off.

In preparation for stoping the vein was cut out its full width on the sill floor, beginning at the main crosscut from the shaft and high enough to allow 6 feet above the sills. At first the sill was timbered 2 square-sets high, but later only 1 set high. Cribbed manways and chutes were spaced on 22½-foot centers where the vein was 18 feet or less in width. Where the vein was wider double rows of manway chutes were installed, with the same longitudinal spacing. The sill sets were covered with a double thickness of 2- by 12-inch cut lagging to support the fill. After the silling was completed, the stope backs were drilled to a depth of 7 feet. The sets were then floored over, chutes were raised 3 feet, and filling was placed to that height. The fill was leveled and floored with 2- by 12-inch by 6-foot cut lagging before the ore was blasted. The ore, after blasting, was shoveled into the chutes, the floor was swept and then taken up, and 6 feet of fill was run in. The stopes then were carried up in successive 6-foot cuts, using stopers for drilling the backs. Waste for filling was obtained from development headings and from stope-wall raises driven between the chutes. These raises often extended 60 feet from the vein in the widest stopes. They weakened the walls considerably and thus caused trouble when the stopes approached the level above. At this stage the backs usually had to be supported by timber cribs. Waste was spread by small air hoists and scrapers in clear stopes and by wheelbarrows in cramped quarters. In the Big Jim ore shoot the walls were more solid than in the Tom Reed Extension, and often it was possible to take two 6-foot cuts of ore before filling. Waste for filling in the Big Jim stopes was obtained cheaply from raises to millholes on the surface. The raises were put up from a waste distribution level on which the waste was trammed to the stope waste raises. The use of scrapers for spreading filling resulted in material saving in filling cost.

Direct stoping costs, 1917 to 1924, inclusive, were as follows:

Direct stoping cost, United Eastern mine, 1917–24, inclusive

	Cost per ton of ore [67]			Cost per ton of ore [67]
Labor, mining	$1.236		Other supplies	$0.170
Labor, filling	.514		Power	.136
			Miscellaneous	.008
Total labor		$1.750		
Timber		.445	Total stoping cost	2.923
Explosives		.414		

The figures given did not separate tramming costs from total stoping cost, but assuming a tramming cost of 35 cents per ton, which would be reasonable, the direct stoping cost, figured on the same basis as that used in setting up costs at other mines discussed in this bulletin, would be $2.57 per ton.

PARK UTAH MINE, PARK CITY, UTAH

The square-set method of stoping is used for mining most of the ore at the Park Utah mine. Cut-and-fill stoping has been used, however, where the hanging wall and ore were strong. The system employed at this mine has been described by Hewitt.[68]

The stopes are served by waste raises spaced about 200 feet apart and by a square-set raise half-way between for manway and ore chute. The level is widened to the full width of the ore and is either timbered with stulls, as shown in figure 37, or with square-sets. In the first method a line of 10- by 10-inch stulls is placed on floor stringers so as to leave a drift-size opening along the hanging wall; the opening is lagged off with 3-inch plank, and waste is run onto the sill. This method is used in only a few places; the usual plan is to use square-sets for sill timbering. Standard square-set timbers are erected and lagged over the top with 3-inch plank. After the sill cut is worked out fill entering from the raises at the ends of the stope is spread along the stope by drag scrapers. When a stope is mined by advancing upon broken ore as high as the walls will permit without sloughing, the ends are lagged, and the ore is dragged to the chute by scrapers. After the fill is leveled a floor is laid before the next cut is blasted. If the filling consists of fine material the floor is laid directly on it, but if the filling is coarse waste, cross-stringers are laid and the floor is placed on top of them. Three-inch planks cut to 8-foot lengths are used for flooring. Each section of flooring overlaps the previous one by about 8 inches to prevent the loaded scraper from tearing up the planks. With very flat stopes or sticky waste the fill is washed into place with water.

The cycle of operations is as follows: Assume that the stope has advanced some distance above the sill floor and the ore has been removed. Filling is run in at the extreme higher end through a waste raise and dragged into place by a scraper. This work requires about a week. When the fill is within 5 feet of the back it is

[67] Cost includes tramming stope ore.
[68] Hewitt, E. A., Mining Methods and Costs at the Park Utah Mine, Park City, Utah: Inf. Circ. 6290, Bureau of Mines, 1930, 18 pp.

leveled off carefully and the floor is laid; this work requires about two shifts. From the ore chute at the opposite end of the stope from the waste raise a flat 8-foot cut is drilled and blasted the length of the stope. At the same time scraping is in progress to remove excess ore. Enough ore is left in the stope to allow the machines, which are mounted on columns set on the broken ore, to reach the back. When the cut reaches the waste raise a return 8-foot cut is made. The scraper is operating continuously, and when the second cut has been completed the ore is cleaned out of the stope, the floor is removed, and the stope is ready for filling again. About 2

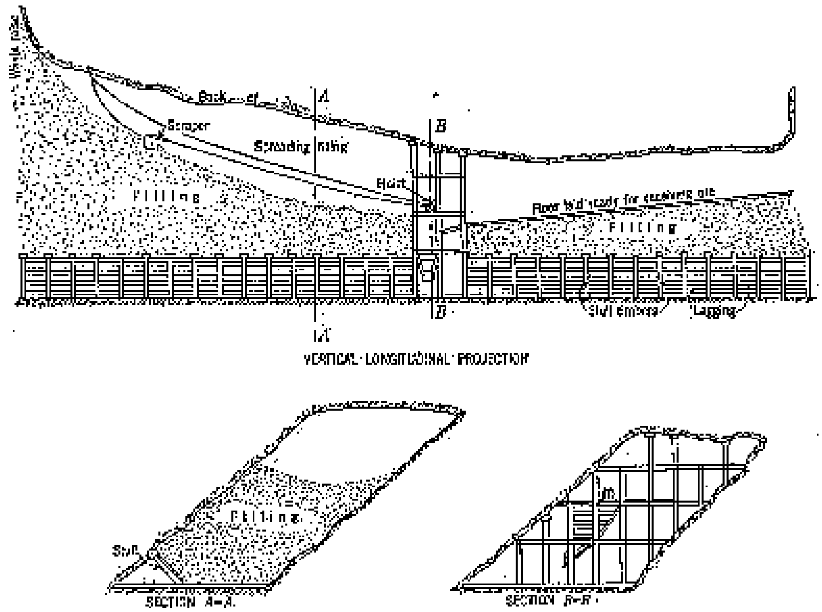

FIGURE 37.—Cut-and-fill stoping on stull timbers, Park Utah mine, Utah.

weeks is required. It has been found that the scrapers work best on a floor sloping slightly toward the hoist, as shown in figure 37. One cut-and-fill stope produced 100 tons per day for 6 weeks, using 3 men per shift. The use of scrapers has reduced stoping costs 30 percent.

CHAMPION MINE, PAINESDALE, MICH.

The lode mined at the Champion mine is the heavily brecciated top of what is known in the district as the "Baltic lava flow." [69] The brecciated portion of the lode is 6 to 50 feet wide. Native copper occurs in the rock fragments of the breccia, in the cementing material of the breccia, in the amygdules, and in the trappy rock under the amygdular part of the lode. The Baltic lode on Champion property is 8,000 feet long, is very straight along the strike, and dips uniformly 70° to the west for 3,000 feet in depth, at which point the dip flattens slightly. The width of the copper-bearing part of

[69] Mendelsohn, Albert, and Jackson, Chas. F., The Sublevel Inclined Cut-and-Fill Stoping System: Trans. Am. Inst. Min. and Met. Eng., vol. 102, 1932, pp. 43–58.

the lode ranges from 10 to as much as 80 feet; the average width now mined is 17 feet. The ore is low-grade.

The lode rock is hard, and the hanging wall is blocky and full of minute seams running in all directions, with larger seams paralleling the strike and dip. When exposed over an area of only a few square yards and unsupported by timber or filling the wall begins to disintegrate and fall into the stope.

Copper is not distributed uniformly in the lode but largely follows pitching shoots. The copper may occur on the hanging side, in the middle, or on the foot side of the lode, or it may be distributed across the lode from hanging wall to footwall. The conditions that affect mining practice and costs are essentially as follows:

1. The lode persists along the strike but varies greatly in width, with zones of barren or very lean rock separating workable ore shoots in which copper is distributed irregularly.

2. Hard lode material lies beneath a seamy, often badly broken, hanging wall which is more treacherous and harder to hold as mining operations deepen.

3. A low-grade ore is present in which copper occurs in the native form and is distinguished readily from barren rock, so that sorting of waste from ore in stopes is feasible; about 40 percent of the rock broken is sorted out as waste.

4. Present stopes are in areas lying below old workings which have been filled with sand or waste rock and from which a plentiful supply of waste for filling purposes is readily available.

The level interval is 100 feet. Formerly stopes were opened above the advancing levels. The levels were silled the full width of the ore, the backs were taken down to a height of 16 feet, pack walls were built to form the sides of the haulageway 7½ by 7½ feet in cross section, and waste was run in behind the walls. Openings were left at 25- to 30-foot intervals for chutes, above which circular mills built of rock were carried up with the stope for ore passes. Ordinary flat-back cut-and-fill stoping was practiced. Details of this system have been described by Schacht.[70] Direct stoping costs with this system have been given by Crane for 1927.[71]

Direct stoping cost, Champion mine, 1927

	Cost per ton of ore milled
Labor[72]	$1.340
Supplies	.405
Power[72]	.009
Miscellaneous	.009
Total	1.763

The stoping system was changed late in 1929. As the mine was deepened the average grade of the ore decreased, and, owing to the depth and the large proportion of lode removed by stoping, the remaining ground gave evidence of being unable to withstand the weight of the hanging country rock. The reasons for changing the system were as follows:

[70] Schacht, W. H., Mining Methods of the Copper Range Co.: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 346–369.

[71] Crane, W. R., Mining Methods and Practice in the Michigan Copper Mines: Bull. 306, Bureau of Mines, 1929, 192 pp.

[72] Includes tramming, timbering, track, compressor, and drill expense items.

1. The cost of maintaining the horizontal stopes was excessive when, to prevent falls of ground on account of crushing, every stope and every horizontal slice had to be supported from end to end with props and cribs.

2. Building the ore passes of rock was slow and costly; timber could not be used because it would rot before the floor pillar could be mined.

3. On account of the long life of a level and movement of the ground the level walls became distorted, the track bulged upward, and the level timbers broke or rotted out. Level-maintenance expense became excessive, and a retreat system of stoping had to be substituted.

4. Filling and building up ore passes interrupted the production of ore in the stopes; only 3 of the 5 stopes on a level could produce at one time because of these interruptions.

5. Although horizontal stoping lent itself admirably to hand sorting, the copper in numerous patches of ground was distributed so uniformly that hand sorting was impossible. In such instances the horizontal arrangement did not permit large tonnages to be handled quickly.

6. The retreat system provided only one short stope at each end of a level which limited greatly the productive capacity of a level.

The new method has been termed the "sublevel, inclined, cut-and-fill system."[73] Haulage drifts are driven from the shaft crosscut to the ore boundaries. About 200 feet back from the boundary, a two-compartment chute and manway raise is driven to the level above. (See fig. 38.) Stations are cut in the raise 33 and 67 feet above the floor of the level, and from them subdrifts are driven in the lode in both directions. These subdrifts are driven the full width of the ore. Scrapers operated by 15-hp. double-drum, electric hoists are used for dragging the broken rock into the chute. When the upper subdrift reaches the boundary a raise is driven to the level above, after which stoping is begun.

From the boundary raise on the upper sublevel the back of ore is stoped by a series of inclined slices. (See fig. 38.) The miners advance the breast up the slope at an angle of 38° until only a small pillar 2 to 4 feet thick is left to support the fill in the worked-out level above. This pillar is drilled and left standing until all copper rock has been picked out of the stope. The pillar is then blasted, and the fill runs in from above, completely filling the stope. Most of the copper rock in the small pillar runs to the bottom of the pile, where it can be picked out. The miner now begins again at the bottom of the rill and cuts out another inclined slice. Loose ground is supported on props or cribs. Holes are drilled with mounted drifters, the burden on the holes being 3 to 4 feet. When a barren or lean section of the lode is encountered it is left in as a pillar, and the miners move back to the next appearance of ore and begin a new sublevel stope. When a stope has been carried back about 45 feet a similar stope is begun on the next sublevel below.

The copper rock is either picked out, loaded into a sublevel car, and trammed to the chute, or it is picked out, thrown into the trough formed by the scraper, and dragged to the chute by the

[73] Mendelsohn, Albert, and Jackson, Chas. F., Work cited.

scraper. At first it was thought that it would be impracticable to use scrapers to transfer the ore from the stopes to the chutes because of the necessity for sorting. It was demonstrated later that not only was scraping much more efficient where sorting was not re-

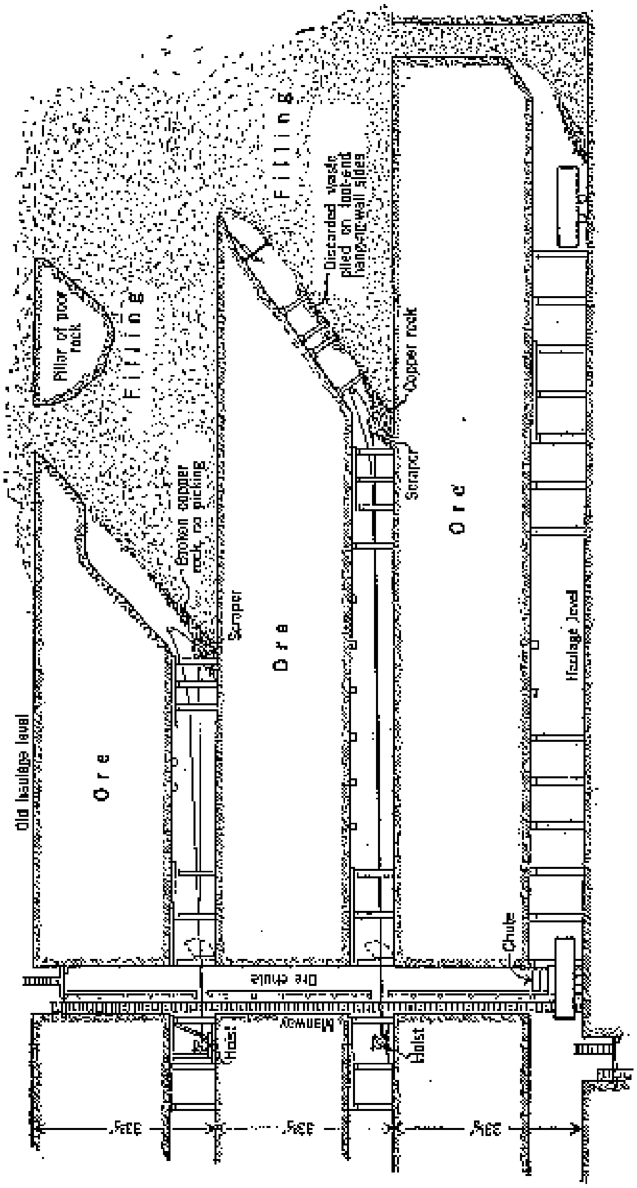

FIGURE 88.—Sublevel, inclined cut-and-fill system, Champion mine, Michigan.

quired, but also it often could be employed with appreciable savings compared to car loading, even where sorting was necessary. The use of scrapers in drifts and subdrifts has reduced the cost of drifting nearly $2 per foot and increased the speed of drifting.

With the stope car the most that two men could shovel without picking was 30 tons per 8-hour shift. With the scraper two men can handle 40 tons per hour when no picking is done.

The advantages of this system of mining as applied at the Champion mine are as follows:

1. High rate of extraction from a block of limited size (of particular advantage when mining on the retreat) is possible.
2. In weak ground the time element has an important bearing upon failure of ore and wall rocks before extraction of ore is complete and the affected area can be abandoned. Rapid extraction of the ore reduces the extent of failure, the cost of artificial support, and the hazards of mining.
3. Similarly the dilution of ore by waste from the walls is reduced, and the ore is extracted more completely.
4. The system permits close supervision by a minimum number of bosses due to a high degree of concentration of operations, which promotes safety and efficiency.
5. Filling is run into place by gravity, and little or no handwork is required to spread and level the fill, an advantage also possessed by conventional rill-stoping methods. In the latter, however, the stopes usually are long; there is inadequate space for storing planking and timber, drills, gear, and drill steel; and little opportunity is afforded for sorting and keeping blocks of waste out of the ore.
6. The sublevel system affords ideal conditions for the use of power scrapers, as the scraper works over a solid bottom and no floor is required on the bottom of the sublevel.
7. Ore broken in taking down backs often is the cheapest ore in the mine, and power scraping affords a cheap method of handling this broken ore. The system is essentially one of taking down backs.
8. The method commends itself from the standpoint of safety, as the stopes are short, permitting quick retreat of the workmen, the broken ore runs down to the toe of the rill, and the shovelers or scraper attendants stay at one place close to the back, which can be inspected and kept secure at that point.
9. By working on the retreat level maintenance costs are reduced to a minimum because by the time the level becomes heavy under the stope area the ore usually will have been extracted and that portion of the level can be abandoned.

Direct stoping cost per ton of ore hoisted, August 1 to December 31, 1930, was as follows:

Direct stoping cost, Champion mine, Aug. 1 to Dec. 31, 1930

	Cost per ton of sorted ore hoisted	Cost per ton of stope rock broken [14]
Labor	$0.794	$0.461
Supervision	.056	.032
Compressed air, drills, and steel	.128	.074
Explosives	.103	.060
Timber	.061	.035
Other supplies	.021	.012
Total	1.163	.674

[14] Of the rock broken, 40 percent was rejected as waste in the stopes.

During the same period labor performance in stoping was as follows:

Labor performance in stopes

	Man-hours		Tons per man-shift	
	Per ton hoisted	Per ton broken	Based on tons hoisted	Based on tons broken
Breaking	0.55	0.33	14.55	24.24
Timbering and filling	.21	.13	38.09	60.47
Shoveling and hand sorting	.73	.44	10.96	18.15
Total	1.49	.90	5.39	8.90

When these figures were taken the new system had not been in use long enough for full benefits to be derived therefrom.

PILARES MINE, NACOZARI, MEXICO

The Pilares ore bodies occur in a vertical mass of volcanics and monzonite porphyry having an elliptical horizontal section of 2,000 by 1,000 feet that has slumped downward. This mass is highly brecciated and badly faulted and fractured. The more continuous ore bodies occur around the margin of the slumped mass, but disconnected ones occur in the central portion. Mining conditions and stoping methods have been described by Leland.[76]

Levels have been driven approximately 100 feet apart from the surface down to the 1,800-foot level. Below that, levels are 133 feet apart.

In general the ore bodies are irregular in size and shape, the ore is hard and contains varying amounts of waste, and the walls are weak. The horizontal cut-and-fill method of stoping is most suitable, as irregular ore bodies can be mined clean, admixed waste can be sorted out, and by judicious use of bulkheads and umbrella stulls weak walls and backs can be controlled. Inclined cut-and-fill or rill stoping is employed where the stope walls are solid and the ore body is fairly regular in shape and has enough vertical extent. Some waste can be sorted by this method, but its best application is where the grade of ore will permit the complete section to be mined without the sorting of any waste. Square-setting is used for mining pillars, crushed and broken areas, and heavily faulted sections and is supplemented by top slicing where the latter will not disturb ore areas above or cut off roadways to other operating areas.

The percentages of the ore produced by the various methods were as follows in 1929:

	Percent
Horizontal cut-and-fill	59.4
Inclined cut-and-fill	17.6
Square-setting	17.7
Top slicing	3.7
Shrinkage stoping	1.6

Where the ore extends considerably above the level it is customary to sill the stopes 20 feet above. Where the ore is of small vertical

[76] Leland, Everard. Mining Methods and Costs at the Pilares Mine, Pilares De Nacozari, Sonora, Mexico: Inf. Circ. 6307, Bureau of Mines, 1930, 34 pp.

extent the stopes are silled on the level; and, except in very wide ore, they are mined from wall to wall in longitudinal stopes.

In silling on the level an 8-foot cut is taken to the limits of the ore or stope section, and the back then is taken down to 14 feet above the floor. While this work is progressing fill raises are put up at 30- to 50-foot intervals along the long dimension of the stope. The level is timbered with gangway sets 6 feet apart, and filling is run in around the timbers. The timbers are not framed. They consist of 10- by 10-inch by 9-foot posts, 10- by 10-inch by 5-foot caps, and 8- by 10-inch by 5.33-foot ties. Caps are placed lengthwise of the timber line, and the gangway is covered with 6- by 6-inch cribbing on pole lagging. At 30-foot intervals chutes and manways are raised, and 12-inch grizzlies of 60-pound rails are placed over the chutes.

After the chutes have been installed and filling around the timbers has been completed, stoping is resumed, beginning at the weaker end of the stope. An 8-foot cut is taken, using a mounted machine for drilling. As the cut advances, shovelers work at the toe of the ore pile behind the machine, shoveling the ore into the chutes and piling sorted waste to one side. Waste filling follows the cut closely, so that production from the section is virtually continuous. Wherever possible, "joker" chutes are installed at the bottoms of the fill raises, from which the waste is drawn into cement buggies used for spreading the filling. Slushing has worked out successfully for spreading the fill in several large stopes.

Where stopes are silled above the level a raise first is driven in ore to the level above. An 8-foot sill cut is started from the raise and 20 feet above the level; while this cut is being taken chutes and manways are driven from the level below at 25-foot intervals to connect with the sill of the stope. At the same time waste fill raises are driven to the level above at 30- to 50-foot intervals. When silling has been completed the back of the stope is taken down 12 feet high, and the chutes and manways are raised 8 feet above the sill floor with crib sets. Each succeeding cut is 8 feet high. A 3-inch plank flooring is laid on the filling in the higher-grade stopes to prevent loss of fines in the fill.

Preparatory work for rill stoping is similar to that for horizontal stoping. The sill is cut flat 20 feet above the level and carried to the limits of the ore or stope section. Where the ore body exceeds 30 feet in width stope sections 30 feet wide are laid out across the ore body from foot to hanging wall. Where the ore is less than 30 feet wide stopes are laid out lengthwise of the ore body. Individual sections are not over 30 feet wide or 70 feet long. At one end of the stope an ore chute is installed 10 feet from the section boundary or, if the section is laid out across the ore body, 10 feet from the waste wall on the core side of the ore. Placing the chute 10 feet from the end of the stope provides a space as wide as the stope behind the chute where sorted waste may be thrown. An auxiliary chute and manway usually are installed near the center of the stope to facilitate handling broken ore during the rill-forming period. At the opposite end of the stope from the ore chute a waste raise is driven to the level above. Upon completion of the raise fill is run in, an inclined cut is taken parallel to the slope of the filling, and the stope is filled again. Inclined cuts are followed by filling until the toe of the fill reaches the end ore chute. Succeeding cuts 8 feet high are advanced upward from the toe toward the fill raise. A floor of 3-inch plank is laid on the fill after each filling operation and before the next inclined cut is taken.

Direct stoping costs for all stopes in 1929, when 77 percent of the ore was mined by cut-and-fill stoping, were as follows:

Direct stoping costs, Pilares mine, 1929[76]

	Cost per ton mined		Cost per ton mined
Labor	$0.630	Timber	$0.243
Supervision	.181	Other supplies	.007
Compressed air, drills, and tools	.175	Total	1.370
Explosives	.134		

STRINGER-SET CUT-AND-FILL STOPING IN ORE BODIES OF MODERATE WIDTH

The stringer-set cut-and-fill system is employed at some mines in the Coeur D'Alene district of Idaho. The junior author[77] prefers to class this system as a modified square-set method. Because the timbering employed is not composed of framed square-sets and the cycle of operations—that is, breaking, removing broken ore, and filling—is the same as the cycle in conventional cut-and-fill stoping, the senior author prefers to consider the system a modification of cut-and-fill stoping.

MORNING MINE, IDAHO

The method employed at the Morning mine has been described by Wethered and Coady.[78] The Morning vein is a metasomatic fissure in a zone of close sheating cutting the quartzite country rock at a small angle to the slaty cleavage. The stoping width of the vein ranges from 6 to 30 feet and averages about 13 feet; it dips 80° to 90°. The fissuring extends into the walls to considerable depths in some places, whereas in others it dies out quickly after leaving the main fracture. Mineralization occurs where the fissuring was most intense and has completely replaced the quartzite with ore.

The ore is not heavy and the timbers are not required to bear much vertical pressure. The tendency of the wall rock to swell and to slab off in large, platy pieces as soon as air reaches it causes the greatest concern in mining. Because of side pressure from the walls close timbering is necessary at all times. The principal ore minerals are galena and sphalerite. The gangue minerals are siderite, barite, and quartz.

The vein is developed by drifts in ore on levels 200 feet apart vertically. The drifts are timbered with regular drift sets on 5-foot 2-inch centers. Three-compartment raises in ore, with a timber slide, manway, and chute, are carried up at 125-foot intervals

[76] 77.0 percent cut-and-fill, 17.7 percent square-setting, 3.7 percent top slicing, and 1.6 percent shrinkage stoping.

[77] Gardner, E. D., and Vandenberg, William O., Square-Set System of Mining: Inf. Circ. 6691, Bureau of Mines, 1933, 73 pp.

[78] Wethered, C. E., and Coady, Leo J., Mining Methods at the Morning Mine of the Federal Mining & Smelting Co., Mullan, Idaho: Inf. Circ. 6288, Bureau of Mines, 1930, 13 pp.

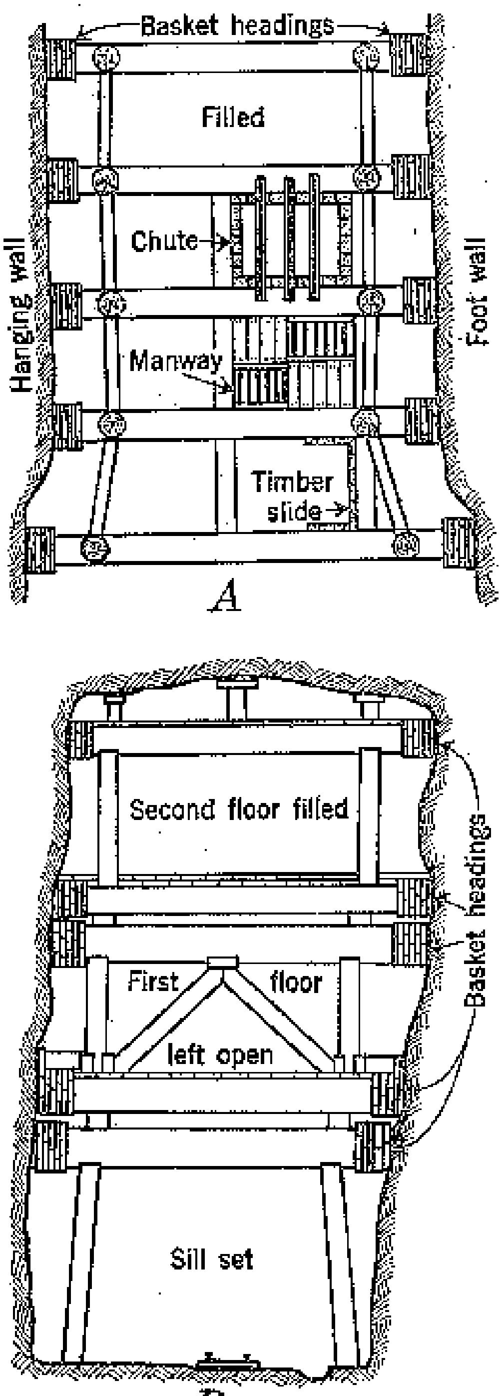

FIGURE 39.—Stope timbering, Morning mine, Idaho: A, Partial plan; B, vertical cross section.

along the drift. Formerly chutes were 25 feet apart, but the expense of maintaining so many chutes was too great. The stope filling is carried on the lagged drift sets.

Drifting machines are used in stoping. The miner is aided in setting up his machine by a mucker and then drills and blasts his round, which consists of ten to twenty 5-foot holes. In blasting, five or six sticks of 25-percent gelatin dynamite are used per hole. The ore breaks well, and the few large boulders produced are easily broken with a hammer.

Figure 39 is a partial plan and vertical cross section of a stope, showing the timbering method. As the stope advances upward three floors are open at a time, one above the other—the top or mining floor where the back is being drilled and blasted, a shoveling floor beneath the mining floor where waste is sorted from the ore and the ore is shoveled through portable steel slides into cars, and a tramming floor one set below the shoveling floor. A stope set consists of two round posts 8 feet long and one round cap 8 to 16 feet long extending from wall to wall. Girts of round lagging are placed between the posts of adjacent sets. The over-all height of a stope or floor is 9 feet, and sets are 5 feet apart.

The ore is blasted onto the shoveling floor, and the loaded cars on the tramming floor are trammed to the chutes and dumped. As soon as a face on the mining floor has advanced 125 feet, or to the next chute, rails, car, and steel slides are moved up one floor. The rails are relaid on the previous shoveling floor, the previous mining floor becomes the new shoveling floor, and a new mining floor is begun above. Timbermen working in pairs prepare for a set of timbers and block the sets in place. Waste for filling is run in to fill the old tramming floor. The waste is obtained from hand sorting in the stopes and from waste raises or crosscuts driven into the stope walls. Scrapers are used for dragging waste from the waste crosscuts to the stope.

Direct stoping costs during 1928 were as follows:

Direct stoping cost, Morning mine, 1928

	Cost per ton of ore hoisted		Cost per ton of ore hoisted
Labor	$1.550	Timber	$0.322
Supervision	.142	Other supplies	.054
Compressed air and power, drills, and steel	.189	Total	2.371
Explosives	.114		

The wage scale was as follows: Miners, $5; timbermen, $5.50; timbermen helpers, $4.75; and muckers, $4.50.

Consumption of explosives in stopes averaged 0.693 pound of 25-, 35-, and 40-percent gelatin dynamite. Consumption of timber, including that used in development, was 12.753 board feet per ton.

HECLA AND STAR MINES, BURKE, IDAHO

In an information circular, Foreman[19] has described the mining method at the Hecla and Star mines. The main Hecla ore body (maximum width, 40 feet) occurs along a virtually vertical shear zone in the Burke quartzite. The Intermediate ore body occurs along a similar shear zone parallel to the Hecla ore body. It dips slightly northeast and joins the Hecla vein near the 1,400-foot level. Both veins follow a distinct gouge along which some strike faulting has occurred. Moreover, they generally are associated with a lamprophyre dike which averages about 2 feet in width but is 12 feet wide in places. The ore occurs on one or both sides of the dike. The principal ore mineral is galena; a small amount of zinc is associated with the galena in the Hecla mine and a larger amount in the Star mine.

There is no definite line of demarcation between the lode and country rock, and stringers of ore often are found in the wall. Horizontal, irregular quartz stringers also are found running into the walls, and parts of the wall tend to break to them and to the gouge, causing extremely heavy side and top pressures. It often is necessary to mine a face having a horse of waste between two stringers of ore. The horse may be 6 or 7 feet wide, but the whole face usually is stoped and the waste sorted from the ore. This practice sometimes is varied by mining one stringer for 2 or 3 floors, filling tightly, and then stoping the other stringer.

[19] Foreman, Charles H., Mining Methods and Costs at the Hecla and Star Mines, Burke, Idaho: Inf. Circ. 6232, Bureau of Mines, 1930, 21 pp.

The veins in the Hecla mine are developed by drifts on the 300-, 600-, 900-, 1,200-, 1,400-, 1,600- 2,000-, 2,400-, and 2,800-foot levels. In the Star mine, levels are approximately 200 feet apart. When the drifts encounter ore their height is increased to 13 feet to give stoping room above the timbers, and the entire width of the ore is removed to a maximum of 18 feet.

The early practice was to install three-compartment raises with a chute, manway, and timber slide at 50-foot intervals along the drift, but because of the heavy side pressure and resulting excessive repair cost the interval was changed to 100 feet. With the old method only 70 percent of the stope excavation was filled with waste. When a stope is begun, a central raise first is put through to the level above. From the raise, horizontal floors are mined 100 feet in each direction. The stope is widened to the full width of the vein on

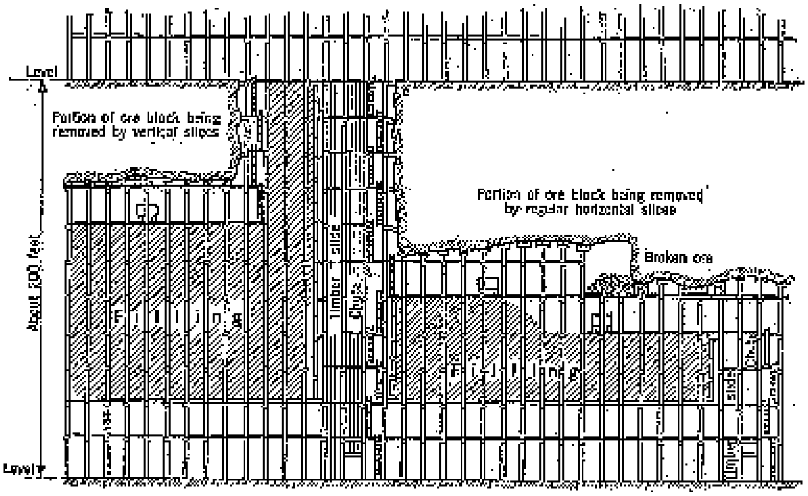

FIGURE 40.—Generalized section, showing method of stoping and timbering, Hecla mine, Idaho.

the first floor. After the first three floors are mined the ore is removed and trammed to the chutes in the end raises. Waste for filling is drawn from the chute in the central raise.

The first floor is mined to the end of the block and timbered as it advances. The ore is passed through the top lagging of the drift sets into cars. The second and third floors are mined in the same manner. While the third floor is being mined the timbers can be placed for supporting filling. The bottom of the waste corral is on top of the first mining floor. (See fig. 40.) This floor is kept open to permit repairs to the drift timbers and corral bottom. After the third floor is advanced far enough, the second floor can be filled from the central raise, a track being placed on the third floor for tramming the waste. The waste is obtained from development on the level above or from a main rock raise connecting with the surface. When the third floor is completed and the fourth floor begun, the track on the third floor is used for transferring broken ore from the face of the stope to the ore chute. Under the system employed usually only one floor is open for any considerable period at one time. When wall conditions permit, however, the track for waste filling is placed on every second floor only, and two floors are then open from the top of the filling to the back of the stope.

The general method of advance, which is carried upward until within 3 or 4 floors of the level above, and the method of timbering with stringer or stull sets are shown in figure 40. The last 3 or 4 floors are mined up to the level by a series of vertical slices, starting at the raise.

The stope rounds consist of vertical rows of three holes each, with one row to each $2\frac{1}{2}$ feet of face. The rounds are blasted with 30-percent gelatin dynamite. Round timber is used for the drift and stope sets.

Direct stoping cost during 1928 was approximately as follows:

Direct stoping cost, Hecla and Star mines, 1928

	Cost per ton mined		Cost per ton mined
Labor	$1.030	Timber [59]	$0.357
Supervision	.080	Miscellaneous supplies	.101
Compressed air, drills, and steel [59]	.215	Total	1.895
Explosives [59]	.112		

Some shrinkage stoping is practiced in parts of the smaller ore bodies, but most of the ore is mined by the variation of cut-and-fill stoping described.

Segregated figures, for horizontal cut-and-fill stoping only, in the Hecla ore body, showing the productivity of stoping labor during 1928, are as follows:

Productivity of stoping labor, Hecla ore body, 1928

Occupation:	Man-hours per ton	Tons per man-shift
Miners	0.256	31.25
Shovelers and stope trammers	.451	17.74
Timbermen and helpers	.611	13.09
Total	1.318	6.07

CUT-AND-FILL STOPING IN WIDE ORE BODIES

In ore bodies that are so wide that the stops will not stand if mined longitudinally the full width of the ore, they are driven across the strike from wall to wall and are of a predetermined, safe width. These stopes usually, but not always, are separated by pillars of ore which are mined after the stopes on each side have been completed and filled.

LA COLORADA MINE, CANANEA, MEXICO

During the first 6 months of 1929, 262,278 tons were mined from the La Colorada ore body, of which 104,057 tons were obtained by inclined (or rill) cut-and-fill stoping, 37,949 tons by horizontal cut-and-fill stoping, 32,220 tons by square-setting, 36,354 tons by shrink-

[59] Includes development in ore; however, since labor, supervision, and miscellaneous supplies for developing in ore amounted to only $0.210 per ton, only a small percentage is for development, and most is for stoping.

age stoping, and 51,608 tons by drifting. The total ore produced by stoping was thus 210,580 tons, of which 142,006 tons (67 percent) were mined by cut-and-fill methods. Catron [31] has described the stoping methods.

The deeper primary ores occur in vertical, pipelike deposits. The upper levels of the Colorada pipe are in the form of a ring surrounding a waste core. The ring of ore converges downward until the waste core disappears on the bottom levels and the pipe becomes a solid, elliptical body of ore. Stope boundaries always are determined by sampling, because the copper minerals extend outward from the main ore body as a network of veinlets which make low-grade ore of the wall rock adjacent to the pipe. The ore of the main pipe is high-grade. The ore and walls of the Colorada ore body are of such a nature that little or no support is necessary in mining.

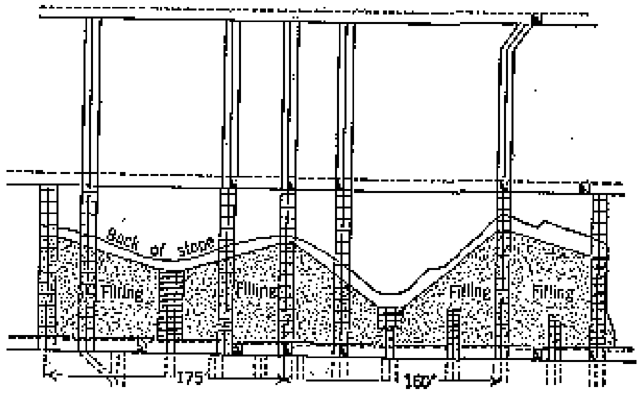

FIGURE 41.—Inclined cut-and-fill stoping, La Colorada mine, Cananea, Mexico.

On the 600- and 700-foot levels, where the ore shoot is narrow, long, filled rill stopes were laid out. Some of these were converted later to horizontal cut-and-fill stopes. Below the 700-foot level the ore becomes much thicker, and just below the 900-foot level the waste core disappears entirely. It was evident that if cut-and-fill stoping were employed pillars would have to be left and later be extracted either by top slicing, which might allow a general subsidence of the ground above, or by the Mitchell slicing system, which would prevent caving. The former course promised more certain recovery of all ore. It was therefore decided to use top slicing for removing pillars between the 800- and 1,000-foot levels and to top-slice the entire ore body below the 1,000-foot level. Horizontal cut-and-fill stoping was the method in general use for mining the cut-and-fill stopes when the mine was visited by the senior author late in 1931. Inclined cut-and-fill stoping was confined to the narrower ore sections.

Figure 41 shows inclined cut-and-fill stoping as practiced at this mine. Drilling is done with drifter machines mounted on 3- or 4-inch columns. In advancing a cut up the slope, 10-foot holes spaced 5 feet apart in both directions are drilled parallel to the slope of

[31] Catron, William, Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico: Inf. Circ. 6247, 1930, 41 pp.

the rill, and a 10-foot round 11 feet high is broken across the width of the stope. After the ore is removed waste is run in from the waste raises to within 3 feet of the back. The waste is leveled and covered with a floor of 3-inch plank nailed to plank sills before the next cut is begun.

The horizontal cut-and-fill system is shown in figure 42, *A*. The typical stope is 30 feet wide and 50 to 160 feet long and has a row of raises spaced at 40-foot intervals along the center line. Stopes and pillars from the 900- to the 800-foot level are laid out exactly over those on the 1,000-foot level. A line of raises also is carried along the center lines of the pillars. Both sets of raises are driven from the 1,000- to the 800-foot level as stulled raises, except that they are cribbed for about 20 feet above the rail. On the 900-foot

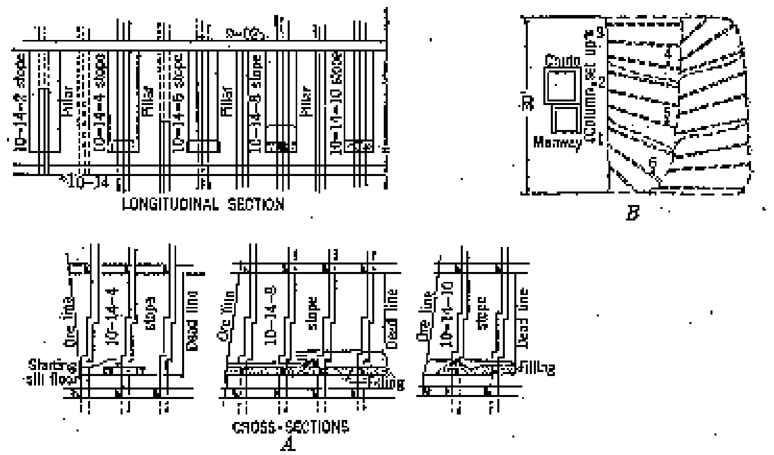

FIGURE 42.—Horizontal cut-and-fill stoping, La Colorado mine: *A*, Vertical sections showing stoping system; *B*, method of drilling holes for advancing sill cut.

level the stopes are silled 11 feet above the rail and on the 1,000-foot level 22 feet above the rail (two top-slice floors) to protect the level. The ground is taken out to the pillar lines on each side of the stope as the stope rises, giving a face 30 feet wide and 11 feet high. This face is then advanced across the ore body by breast stoping. Figure 42, *B*, shows the method of drilling holes for advancing the sill cut. Each row of holes consists of 4 or 5 flat holes, the rows being spaced 3½ to 4 feet apart vertically. Drilling is done with heavy drifter-type machines mounted on 4½-inch columns. As soon as a stope is large enough a scraper is installed to drag the ore to the chutes.

The extraction raises are built up directly under the stulled raises. The manway side is timbered with 4- by 10-inch cribbing and the chute side with 10- by 10-inch cribbing for the first 50 feet and above that with 4- by 10-inch cribbing. Chute lining is unnecessary. The chute is covered with a grizzly of 30-pound rails, with 10-inch openings.

When a breast has been advanced past a second raise the first part of the stope is cleaned out, and a floor of 2- by 10-inch plank, nailed to 4- by 10-inch sills, is laid. The sills are laid parallel to the direction in which top slicing will advance. (Top slicing is employed for mining the pillars over the drifts.) Gob fences are built around the stope, except on the waste sides. These are of 2- by 10-inch plank nailed to other planks used for posts. Any space between the fence and wall is filled with broken ore. The grizzly rails are removed from the chute, the extraction raise is cribbed up to 3 or 4 feet from the back, and both compartments are covered with timber.

Fill is run in through the waste raise and spread by scraper and wheelbarrow to within 3 feet of the back. After filling a temporary floor is laid on the waste, and the second cut is begun. In this and succeeding cuts the rounds are drilled differently from the method shown in figure 42, *B*, as there is now a free face underneath. Flat holes, 5 feet apart horizontally and vertically, 12 feet long, and drilled straight into the face, break the ore readily with 4 to 8 sticks of 40-percent powder in each hole. Carrying the back 2 or 3 feet higher in the center of the stopes than at the sides gives an arch that reduces trouble from loose back.

Direct stoping costs, January 1 to June 30, 1929, when 49 percent of the ore stoped was mined by inclined cut-and-fill stoping, 18 percent by horizontal cut-and-fill stoping, and the balance by square-setting and shrinkage stoping, were as follows:

Direct stoping costs, La Colorada mine, January 1 to June 30, 1929

	Cost per ton of ore produced		Cost per ton of ore produced
Labor	$0.3748	Timber	$0.1979
Supervision	.1310	Other supplies	.0966
Compressed air, drills, and steel	.1879	Total	1.2034
Explosives	.2148		

From January 1 to June 30, 1931, direct stoping costs in horizontal cut-and-fill stopes only were as follows:

Direct stoping costs, cut-and-fill stopes only, La Colorada mine, January 1 to June 30, 1931

	Cost per ton		Cost per ton
Labor (timbering, breaking, and shoveling)	$0.2557	Explosives	$0.1025
Gobbing	.2209	Timber	.1188
Compressed air, drills, and steel	.0996	Miscellaneous supplies	.0059
		Total	.7984

During this period 48,953 tons were mined by horizontal cut-and-fill stoping, and the output per man-shift was 6.88 tons.

CAMPBELL MINE, WARREN, ARIZ.

Stoping methods at the Campbell mine have been discussed by Lavender.[89] The Campbell ore body may be classified as a fairly homogeneous body of sulphide ore of rather high silica content. On the 1,600-, 1,700-, and 1,800-foot levels it averages about 500 feet in length and ranges from 50 to 250 feet in width. The footwall contact is fairly well defined except for fracture zones which have caused some displacement. The dip ranges from nearly 90° to as

[89] Lavender, H. M., Mining Methods at the Campbell Mine of the Calumet & Arizona Mining Co., Warren, Ariz.: Inf. Circ. 6289, Bureau of Mines, 1930, 18 pp.

flat as 25°. The hanging wall ranges from an economic silica-pyrite contact low in copper to a sharply defined wall conforming to the local bedding planes. The ore is hard, and heavy blasting is required to break it fine enough for handling.

Levels are about 100 feet apart vertically, and every other level is a motor-haulage level.

Before stoping the ore body is divided into stope and pillar sections. In general, the plan adopted was to have two adjoining stope sections, each 45 to 50 feet wide, separated from the next two stoping sections by a pillar 45 feet wide. Where the width of the ore body was not too great the stopes were to be mined from hanging wall to footwall. Where the ore body was wider the sections were cut off by either a single cap or regular lead, and new sections were established. When the stope and pillar sections were laid out the plan was to work both stope sections by cut-and-fill stoping and to recover the pillar by a Mitchell slice running at right angles to the long dimension of the stope. It also was planned to mine each section up in blocks 200 feet high where continuity of the ore permitted.

Inclined cut-and-fill stoping was employed for mining the stope sections. Two general types of stopes were used: "Double-lead" stopes, in which a line of square-sets is carried along both sides of the section, from foot to hanging wall, and "single-lead" stopes, in which a line of square-sets is carried on one side only and a line of caps and posts along the other side. In development for double-lead sets crosscuts are driven in the pillar and adjoining stope sections along the section lines, and the ground is cut out for stringer sets. (See fig. 43, *A*.) After stringers have been placed chutes are built in alternate sets, and stoping is begun above the stringers. While the lead sets are being cut out mining is begun in the stope between them. In a flat-bottomed stope the ore is allowed to pile up to form a natural slide toward the lead; the back of the first cut is shaped for the initial fill. Meanwhile, work on the fill raise is pushed, and it is holed to the level above or in some instances even to the next haulage level 200 feet above. The natural angle of repose of the fill is 37°, and the back of the stope follows this angle roughly.

Upon completion of the lead sets and the section between leads the stope is cleaned out by hand or by drag scrapers, and the bottom of the stope is covered with a mat of scrap timber. Waste is run through the fill raise until it is within 2 or 3 feet of the back of the stope. The fill is covered with 2-inch flooring nailed to 2- by 10-inch sills laid flush with the fill. Breaking then is resumed; a cut is begun at the bottom of the incline at the lead set and carried up the slope, the average cut being 12 feet high. Grizzlies of 10- by 10-inch timber spaced 10 inches apart are placed in the lead sets at the intersection of the fill floor line with the sets. After a cut is completed and the ore drawn into the chutes the floor is taken up and filling run in. Each cycle of cutting and filling is approximately the same. The lead sets are raised with the main cut and carried enough in advance of the fill to provide height for gob lagging and to give access to the working floor for the succeeding cut. A manway set is usually carried in each row of leads.

Ordinarily drifter-type machines are used for stoping, and 7- to 8-foot holes are drilled; stopers are used to some extent, principally in lead sets and raises. Figure 43 shows that the back of the stope slopes upward at a flat angle across the stope from the lead sets (fig. 43, *A* and *C*) and also slopes upward from hanging to foot wall (fig. 43, *B*). Figure 43, *C*, is a cross section of a single lead-set stope.

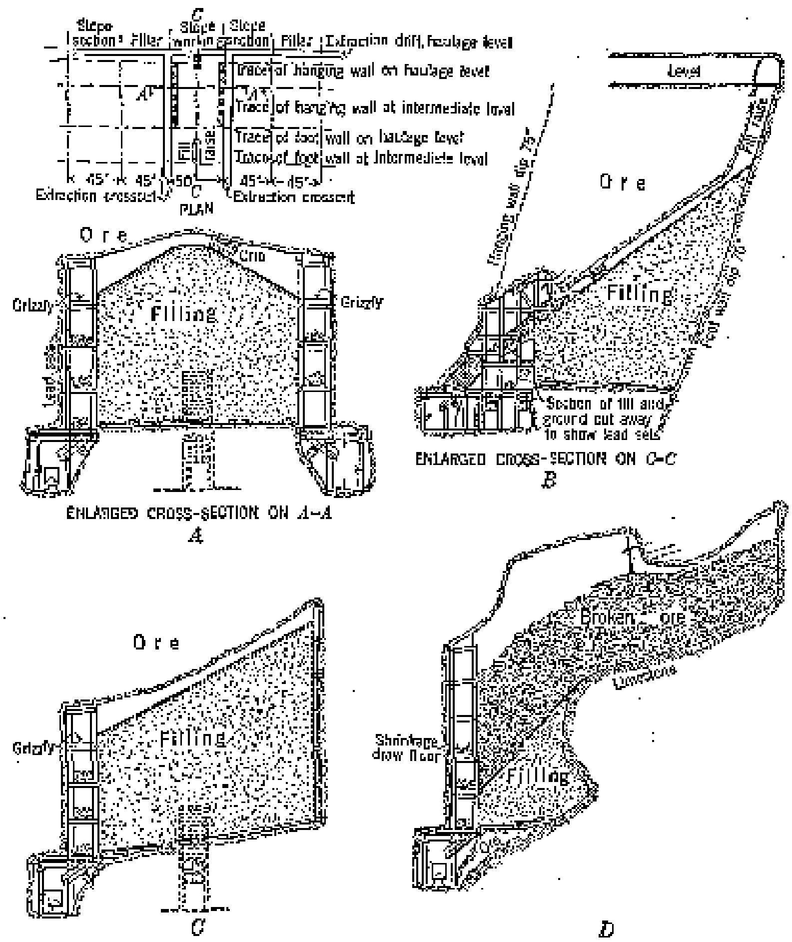

FIGURE 43.—Inclined cut-and-fill stoping system, Campbell mine, Arizona: *A*, Plan and section of double-lead stope; *B*, cross section on lines *C–C* of plan *A*; *C*, cross section, single-lead stope; *D*, cross section of semi-shrinkage stope.

Where the ground is uniformly good the practice sometimes is modified by using a semi-shrinkage-stoping method or a semi-cut-and-fill system. This method is shown in section in figure 43, *D*. Instead of the ore being cleaned out and filling run in after each cut, two or more 12-foot cuts are taken, excess ore being drawn off to make working room; additional cuts are made by working on top of broken ore, as in shrinkage stoping. The balance of the ore from the two or more cuts is drawn off, and filling is run in as in the regular cut-and-fill stoping method. This system effects a large reduction in flooring expense, as the cuts are sometimes as much as 40 feet above the fill before the ore is drawn and the stope filled.

The aim is to attain complete recovery of the ore in the Campbell mine area, as it is high grade. A slight loss of fines is occasioned by their sifting through the cracks in the flooring, and a little dilution may occur at the edges of the stopes.

The following are typical direct stoping costs at the Campbell mine:

Typical direct stoping costs, Campbell mine

Stope no.	Period covered	Type of stope	Cost per ton of ore				
			Labor	Explosives	Timber	Air	Total
69	May 1929 to Jan. 1, 1930.	Double-lead, inclined cut-and-fill.	$0.62	$0.135	$0.165	$0.215	$1.125
90	Jan. 1, 1929, to Jan. 1, 1930.	Single-lead, inclined cut-and-fill.	.70	.10	.13	.20	1.13
412	June 1928 to Jan. 1, 1930.	Single-lead, semishrinkage; first cut-and-fill section.	.81	.20	.26	.35	1.62
All stopes in Campbell area.	Jan. 1, 1928, to Jan. 1, 1930.[1]		.765	.120	.175	.220	1.28

[1] The costs shown here represent the entire production of the Campbell area during the 2-year period, when a large number of stopes were begun or in the initial stage. The higher proportional expense due to these causes and to ore being tied up in shrinkage operations is reflected in the costs. Almost all stoping in this area was inclined cut-and-fill and semishrinkage.

UNITED VERDE MINE, JEROME, ARIZ.

Although more than 85 percent of the ore is mined by horizontal cut-and-fill and square-set stoping other methods are employed for exceptional conditions. Stoping methods at the United Verde mine have been described by Quayle.[a] In 1929 the methods employed and the percentage of the total production by each method were as follows:

Method	Percent	Method	Percent
Horizontal cut-and-fill	61	Inclined square-set	2
Inclined cut-and-fill	3	Shrinkage stoping	1
Horizontal square-set	25	Top slicing	8

The ore deposits are of the massive sulphide, schist-replacement type. The mineralized zone has a horizontal area of 200,000 to 400,000 square feet on various levels and a proven depth of 4,000 feet. Irregular interfingering of the sulphides with schist and porphyry forms an irregular and indefinite footwall boundary. The chief primary copper mineral is chalcopyrite, and the gangue is principally pyrite, quartz, and chlorite.

The main ore bodies occur along the contact of the massive sulphide and schist; smaller bodies are on the schist-porphyry contact. The physical characteristics of the massive sulphide, schist, and porphyry differ so greatly that the location of an ore body with respect to these rocks determines not only the mining method but also the size of stope that can be worked. The massive sulphide is very

[a] Quayle, T. W., Mining Methods and Practices at the United Verde Copper Mine, Jerome, Ariz.: Inf. Circ. 6440, Bureau of Mines, 1931, 20 pp.

hard and dense and stands well over large areas without support, except that an occasional bulkhead is required under a loose slab or where the ore body is weakened by an andesite dike. In this ground cut-and-fill or shrinkage stoping is applicable. The schist varies in hardness and strength; in places cut-and-fill stoping can be employed, while in others small square-set sections are necessary. Shrinkage stoping cannot be employed because of the dilution that would occur when the stopes are drawn. Porphyry ore bodies usually can be mined by cut-and-fill stoping.

Levels are 150 feet apart vertically, and each level is developed by a drift in the schist roughly paralleling the sulphide-schist contact. In places the ore body is so wide that two parallel drifts are required to insure proper chute spacing for the stopes.

Owing to the great variation in width of the ore bodies and to the irregularity of ore boundaries, it is impossible to establish any fixed dimensions for the cut-and-fill stopes. Each is opened to what is considered the maximum size consistent with safe mining. The sizes range from 30 to 60 feet across the ore body and from 60 to 200 feet along the strike. The dimensions also are influenced by the need to maintain regularity in the vertical pillar system; that is, pillars on one level must come under pillars on the level above. Pillars between stopes usually are about 50 feet thick.

Before stoping operations are begun, the boundaries of the ore body are determined approximately by diamond drilling from the contact drift, and the vertical pillar scheme and size of stopes are determined. A raise then is driven to the level above, as near the center of the stope as possible. This raise will serve later as a waste raise. In stopes longer than 100 feet two raises are employed, the second being placed near one end of the stope. After the waste raise is completed silling operations are begun from the raise, 21 to 25 feet above the rail. The sill is cut out 7 feet high, and during silling chute raises are put up from the drift below. Chutes are 16½ feet center to center in massive sulphide ore and 22 feet apart in schist and porphyry.

After the sill cut has reached the pillar lines and the ore limits a second 7-foot cut is begun from the waste raise. In stopes having two waste raises the cut is started from the raise near the end of the stope. All drilling is done with drifter machines. Flat holes 7 to 8 feet deep are drilled in two rows. As many holes as can be used effectively are drilled from each set-up by fanning them out. Holes are shot with 50-percent gelatin dynamite. The ore breaks into fragments, of which 30 percent are too large to pass an 11-inch grizzly. Secondary blasting, which consumes 20 percent of the explosives used, is done with 35-percent gelatin dynamite.

Before the first fill is introduced a sill floor is laid, consisting of two layers of 2-inch planks nailed to 4- by 12-inch sills spaced 5 feet 4 inches apart. Temporary floors are laid on the filling before each cut is begun. After 2 or 3 rounds have been blasted from a new cut shoveling operations are started and follow 10 to 25 feet behind the mining face. As soon as the shovelers pass a chute and are shoveling into the next, timbermen raise the chute 7 feet and filling is begun. About 2 percent of the rock broken is sorted out by the shovelers and discarded as waste.

Filling is trammed from the waste raise in a specially designed car having a scoop-shaped body and running on sectional track laid on the fill. A gob fence of 6- by 8-inch posts 8 feet long, spaced 3 feet 7 inches apart and covered with 2-inch planks, is built along the pillar line on each floor before waste filling is introduced.

Filling material, consisting mainly of diorite and porphyry, is waste rock from open-pit stripping and waste from underground development. Only waste that is virtually free from sulphides is used on account of the fire hazard in this mine.

Inclined cut-and-fill stoping has been employed in a few special cases such as the mining of level pillars, where the weight on a flat back and square brow had to be relieved or square-setting used, or in isolated ore bodies, the general shape of which was suited better to inclined than to horizontal stoping. Horizontal square-set stoping is employed for mining vertical pillars between cut-and-fill stopes and in some areas in the main stopes where the ground is heavy. Inclined square-setting is used principally for extracting level pillars.

Labor performance in cut-and-fill stopes is given by Quayle as follows:

Labor performance, cut-and-fill stopes

Occupation:	*Man-hours per ton of ore stoped*	*Tons per man-shift*
Breaking	0.135	59.3
Shoveling	.547	14.6
Timbering and filling	.466	17.2
Total direct stoping labor	1.148	6.97

Consumption of explosives is 0.55 pound (principally of 50-percent gelatin dynamite) per ton of ore broken. Consumption of timber is 6.95 board feet per ton.

MAGMA MINE, SUPERIOR, ARIZ.

Mining methods at the Magma mine have been described by Snow.[54] The ore occurs in an east-west fault fissure which dips 45° to 80° to the north from the surface to the 900-foot level, where it turns over and dips 80° south to the lowest levels in the mine. A north-south fault of large displacement cuts the vein into several blocks in the western part of the mine. Postmineral movement along the vein has resulted in treacherous hanging and foot walls. Considerable gouge and crushed wall rock, with frequent included lenses of ore, occur along the walls of the wider parts of the vein.

On the lower levels of the mine the principal ore shoot is 1,200 to 1,500 feet long, averages 20 to 30 feet in width, and runs from 6 to 8 percent of copper. The ore carries important amounts of associated silver and gold. About 20 to 25 percent of the ore mined is direct-smelting ore averaging 12 percent of copper, and the rest is milling ore averaging 6.5 percent of copper. Selective mining is practiced to keep these grades separate, but close separation in mining is not required to maintain these grades.

[54] Snow, Fred W., Mining Methods and Costs at the Magma Mine, Superior, Ariz.: Inf. Circ. 6168, Bureau of Mines, 1929, 32 pp.

Factors that affect the stoping methods at the lower levels are as follows: A body of ore averaging 20 to 30 feet and a maximum of 60 feet in width; hard ore between soft, treacherous walls; and ore of relatively high grade.

In narrow ore (less than 15 feet wide) a timbered-rill cut-and-fill system is employed. In wider ore a rill stope-and-pillar system is used. The latter consists of a combination of 16-foot, rill cut-and-fill stopes alternating with 14-foot pillars across the vein from wall to wall.

At the lower levels the haulage levels are 250 or 300 feet apart vertically. The haulage drifts are driven in the footwall 25 feet or more from the vein. Intermediate levels not connected to the shafts are driven about half-way between the haulage levels. Extraction drifts are driven on each level in ore along the footwall, and hanging-wall drifts are carried in the hanging wall 13 feet from the vein. The latter are used for handling waste for filling, timber, and supplies.

In the stope-and-pillar system, crosscuts are driven from the footwall extraction drift to the hanging wall on 30-foot centers under the centers of the future stopes. These are widened to 16 feet and blasted down to a height of 10 or 12 feet, forming the sill floors of the stopes. The length of each stope is the width of the vein at that point and may be 15 to 60 feet. Stringers of 10- by 10-inch timber 16 feet long are placed across the sill on 5-foot centers, and a double floor of 2-inch plank is laid thereon. The extraction drift is timbered with battered sets. From this drift the three sets next to the footwall of each 16-foot stope are carried up as a three-compartment raise as mining progresses; the outside sets are used as ore chutes and the center set as a manway. These are used later in extracting the pillars. Square-sets are placed across the vein on one side of the rill stope on the sill floor and single 8-foot posts of 10- by 10-inch timber are stood across the vein on the opposite side to form gob lines. When the stope is filled the square-sets are held open as a crosscut to provide access to future stopes below and to the hanging-wall drift.

After sill timbering has been completed a cribbed raise is started in the hanging wall about 12 feet from the bottom of the sill, and the broken waste from this raise is dropped into the stope for filling. When the waste lying on its natural slope is within 4 feet of the back a floor is laid on the waste, and a 7-foot cut of ore is mined, inclined upward toward the hanging wall and parallel to the slope of the waste fill, or about 35° from the horizontal.

After the broken ore is cleaned out, new gob lines, consisting of single posts end on end and 2-inch lagging, are placed along each side of the stope. The ends of the posts are half-framed, and a single strand of old hoist cable is used to tie the two lines of posts together across the stope. The waste raise is carried up only fast enough to provide the necessary stope filling and usually reaches the level above when the stope is about 35 feet high on the hanging-wall side. After this, waste from other sources is dumped into the stope through the raise as required for filling. The top of the waste raise connects with the hanging-wall drift on the upper level.

When the stope reaches the level after the succession of inclined cuts, square-setting is used to extract the ore under old filled stopes. The stopes are carried up the full lift between haulage levels, and intermediate levels are used for bringing in filling and supplies and for access to the stopes.

Drilling is done with hand-rotated stopers. Holes are spaced about 3 feet apart and drilled so as to shatter the back as little as possible and maintain an arch with its ends resting on the pillars. A crew for a rill stope consists of two miners who do all the drilling, timbering, and shoveling. Very little shoveling is necessary because the ore runs on the sloping floor of the fill to the grizzly over the chute.

Pillars between filled rill stopes are extracted by a modified form of the Mitchell slicing system developed at Bisbee; mining is done from the top downward. The top of the pillar is shot out, and a line of diagonally braced segment sets is placed immediately below the filled sill floor of the finished stope above. Sometimes a second cut below the first also is timbered with segment sets. The pillar is then mined downward by successive cuts sloping 40° toward the footwall. The broken ore runs by gravity into the square-set raise, which was used in the adjoining rill stope. As stoping progresses, 10- by 10-inch stringers are placed across the stope; these are 5 feet apart horizontally between corresponding rill-stope gob posts and 7 feet apart vertically. Only enough ground is broken at a time to make room for one stringer. Drilling is done with jackhammers, and most of the holes are drilled flat from staging nailed on the stringers. Comparative partial costs of different types of stopes are as follows:

Stoping costs per ton, Magma mine

System	Labor				Supplies			
	Mining	Mucking	Timbering	Bonus	Waste fill	Explosives	Timber and material	Total
Timbered rill	$0.46	$0.03	$0.53	$0.08	$0.18	$0.05	$0.46	$1.79
Rills	.32	.014	.40	.17	.03	.08	.34	1.354
Pillars	.26	.063	.39	.22	.23	.04	.28	1.483

Total stoping costs covering all methods were as follows during 1928:

Total stoping cost, Magma mine, 1928

	Cost per ton		Cost per ton
Labor	$1.490	Timber	$0.470
Supervision	.267	Other supplies	.058
Compressed air, drills and steel, and power	.414	Total stoping cost	2.849
Explosives	.150		

The wage scale to October 1928 was as follows: Miners $4.95, timbermen $5.23, and shovelers $4.13. Thereafter it was 10 percent higher.

Stoping, timbering, and filling averaged 1.706 man-hours per ton, or 4.68 tons per man-shift. Consumption of explosives for stoping only was 0.7 pound per ton of ore mined and consumption of timber for stoping only, 17.0 board feet per ton.

EAGLE MINE, GILMAN, COLO.

Borcherdt[95] has described the mining practice at the Eagle mine. The most important ore bodies are in a dolomitic limestone bed about 125 feet thick and are of the replacement type. Just above this bed, but separated therefrom by a thin bed of shale and quartzite is a porphyry sill averaging 60 feet in thickness. The deposits occur as typical blanket or manta ore bodies, long cylindrical deposits elliptical in cross section, and chimneys. The massive sulphide ore is a mixture of pyrite, sphalerite, siderite, and galena.

Most of the production is obtained from horizontal cut-and-fill stopes and from square-set stopes. In elliptical, cylindrical ore bodies, which are 35 to 60 feet high and 100 to 250 feet long, the ore body is sectioned into transverse stopes and pillars. Stopes are 35 feet wide and pillars 15 feet. They are worked by ordinary cut-and-fill stoping from the base to the top of the ore. In some instances the ore is shoveled by hand into the chutes, but more often it is scraped. Filling from development or waste caving is passed into the stopes through raises from an overlying level and spread by scrapers. Filling either may follow the ore cut closely or be introduced after removal of a complete 8-foot slice. A square-set aisle one set wide is carried along each side of the stope, thus reducing the unsupported span; this span rarely is as much as 35 feet. The square-sets also partition the waste from the pillar walls and provide a tie for 15-foot stringers and trusses that comprise the timbering in mining the pillars.

The ore is weak at the top of the ore bodies, and often the final cut must be mined by square-setting. Completed stopes are filled to the top and cribbed tightly against the back. Pillar removal follows mining of the ore from adjacent stopes. The ore is broken from one end of the pillar and drawn through chutes in the square-set aisles erected during mining of the cut-and-fill stopes. Mining progresses from one end of the pillar to the other.

The following data apply to the operation of nine cut-and-fill stopes in zinc ore bodies, and the figures include square-setting of the stope tops. Mucking and filling in the stopes were done largely with scrapers.

Data on 9 cut-and-fill stopes, Eagle mine

Production per machineman shift	tons	28.4
Production per timberman shift	do	21.4
Production per mucker shift	do	19.5
Production per fill-labor shift	do	44.2
Production per man-shift, all stope labor	do	6.4
Timber per ton of ore	board feet	7.3
Powder per ton of ore	pound	.53

[95] Borcherdt, W. O., The Empire Zinc Co.'s Operation at Gilman, Colo.: Eng. and Min. Jour., vol. 132, 1931, pp. 99–105.

KRIVOY ROG DISTRICT, RUSSIA

Cut-and-fill stoping has been the method used most for mining the iron ores of the Krivoy Rog district in Russia.[30] It has been applied to strong and weak ore bodies with solid or weak walls and to both large and small ore bodies. Although inclined cut-and-fill methods have been employed horizontal cut-and-fill stoping generally is used. In narrow ore bodies the stopes are driven longitudinally, but transverse stope sections are laid out in the wider ore bodies. These stope sections may alternate with stope pillars or may be driven alongside each other. Pillars are extracted by cut-and-fill stoping on top slicing. An interesting feature of the practice is the use of filling composed of a mixture of clay and quartzite (70 to 75 percent of clay and 30 to 25 percent of quartzite). This mixture is sometimes leveled off in thin layers in the stopes, each layer being moistened before the next layer is introduced. Filling of this nature has been found to consolidate to such an extent that adjacent stopes can be mined in slices 7 feet high alongside the filling without employing gob fencing. Dry filling shrinks somewhat; pure clay shrinks 30 percent and clay with quartzite 15 percent. Experiments have shown that the best mixture is obtained with 25 percent of quartzite and 75 percent of clay; if more than 30 percent of quartzite is used, the mass separates easily into flakes. Semiwet filling is most satisfactory when composed of clay or a mixture of clay with 25 percent of granulated quartzite. This mixture may shrink as much as 50 percent of the original volume.

SUMMARY OF CUT-AND-FILL STOPING

1. *Applicability.*—Cut-and-fill stoping is applicable under a wide range of conditions. It is employed for mining both small and large deposits of regular or irregular outline, which dip at angles less than the angle of repose of the broken ore and filling to 90°. Where the dip is less than the angle of repose of the broken ore means must be provided for moving the ore from the face of the stope to the haulage level; usually the method is employed where the broken ore can be moved to the level by gravity—that is, in steeply dipping tabular deposits or in large bodies of ore of considerable vertical extent. The method finds its widest and best application in the stoping of firm-, medium-, or high-grade ore, enclosed between walls one or both of which are weak and require permanent support. The ore itself should be firm enough to support itself over the width of the stope, as the filling does not directly support the back but only indirectly by sustaining the walls, which carry the ends of the span of unsupported ore. It also has an important application in mining deposits of less than stoping width.

2. *Flexibility.*—Cut-and-fill stoping is flexible. Irregular ore boundaries can be followed in the stopes, and stringers of ore from the main ore body can be followed and mined. Waste inclusions in-

[30] Ayvazoglou, W., The Method of Underground Mining of Iron Ore in the District of Krivoy Rog (Translation from the original Russian of A. K. Bonidovsky): Inf. Circ. 6254, Bureau of Mines, 1930, 48 pp.

the ore body can be sorted from the ore and gobbed in the stopes. Ore of less than stoping width can be mined by the stripping or resuing variation of the method with minimum loss of ore and dilution with waste. Moreover horses of waste may be left intact. Cribs or stull timbers can be erected on top of the filling to support local patches of weak back or exceptionally weak wall, and changes can be made readily from cut-and-fill stoping to square-set stoping in sections where the ore is too weak to be mined without regular timbering.

Cut-and-fill stoping, however, lacks flexibility in rate of production from individual stopes, except that favorable changes in ground conditions may allow more rapid extraction by permitting higher cuts to be taken or two or more cuts to be mined (as by "semi-shrinkage") before removal of the ore and subsequent filling. The rate of output is restricted by cessation of ore breaking during filling or, where breaking and filling are continuous, by the relatively small individual rounds that can be blasted.

The ore is removed as fast as it is broken (except in the "semi-shrinkage" variation). This is advantageous, as capital is not locked up in broken ore left in the stopes. Moreover, if the ore is a sulphide ore which is to be concentrated by flotation, or ore which will heat and may take fire if left in the stopes immediate removal of the broken ore is an obvious advantage. On the other hand, the method does not provide a reserve of broken ore which is sometimes advantageous for supplying a steady output to the mill if some of the stopes are tied up temporarily.

3. *Recovery.*—A high recovery usually can be obtained. Stringers of ore, which often would have to be left or would be missed inadvertently in shrinkage stoping, can be followed and stoped. The walls can be prospected readily from the stope floors for parallel ore shoots, and if any are found they can be developed and mined from the stope. If nearby ore shoots are discovered after the stopes are completed these usually can be developed and mined without undue difficulty, as the filling in the stopes will have supported the walls and prevented movement and fracturing of the newly discovered adjacent ore bodies. Clean mining is possible because waste broken inadvertently with ore, or waste which may slab off the walls, can be removed from the broken ore and gobbed in the stope.

4. *Development.*—Somewhat more stope development is often (but not always) required than for shrinkage stoping, because it generally is necessary to drive waste raises through to the level above before stoping can begin. An exception is where the filling material is composed entirely or partly of waste sorted in the stopes, and the balance is obtained from waste stopes or waste raises driven into the stope walls. In shrinkage stoping it may be necessary to drive as many raises to the level above for ventilation as are required to introduce waste in cut-and-fill stoping, in which case raise development would be a stand-off, whichever method was employed. The stopes may be silled on the level or at some distance above as in shrinkage stoping. With cut-and-fill stoping it is usually unnecessary to drive chute raises through the drift pillar close together, and they may be 25, 50, or even 100 feet apart, whereas in shrinkage

stoping they seldom are farther apart than 15 or 20 feet. Thus, where the stopes are silled above the level stope development actually may be less for cut-and-fill stoping than for shrinkage stoping.

Since filling is commonly introduced from the level above through waste raises, it ordinarily requires longer to prepare a stope for cut-and-fill stoping than for shrinkage stoping owing to the delay incident to driving the raises.

Johnson and Gardner have worked out and tabulated ideal development ratios for various cut-and-fill stoping methods at a group of representative mines.[57] The ratios given are calculated on the basis of information supplied in articles describing the details of practice at the several mines, and the authors do not claim that they closely represent the results obtained in actual practice. They do represent, however, the results that could be obtained if each stope were developed and mined in strict accordance with the standard plans described. On this basis 14 mines averaged 0.0058 linear foot of stope development per ton of ore, or 172 tons of ore per foot of stope development. Excluding three mines working very narrow ore bodies the figures are, respectively, 0.0048 foot per ton and 208 tons per foot. Averages for these three mines are, respectively, 0.0451 foot per ton and 22 tons per foot.

In actual practice the tonnage per foot of development would generally be considerably lower than the theoretical figure owing to the driving of auxiliary openings such as sublevels and short crosscuts and raises, the need of which cannot be anticipated in advance by any systematic plan of development. Such openings may have to be driven to mine around horses of waste, to follow stringers of ore, or to mine parallel ore bodies and to provide additional manways, fill raises, and ore chutes. Actual stope development required may exceed theoretical development by 25 to 50 percent in some instances.

5. *Cost of cut-and-fill stoping.*—Direct stoping costs at a number of mines which have been described in the foregoing pages are shown in table 10. The costs are not directly comparable, as conditions at the different mines vary greatly; moreover, the costs cover some mines where stoping methods other than cut-and-fill are employed, but cut-and-fill stoping is the principal method used. High costs per ton of ore hoisted are to be expected in narrow veins where resuing is practiced or where much close sorting is required in the stopes. The costs at the first three mines indicate to what extent these factors may increase the stoping cost.

Costs naturally are lower in wide ore bodies than in narrow ones, although the cost shown for the United Eastern mine is relatively high. In this instance, however, the tabulated costs include tramming and probably other distributed items not included in the figures for the other mines.

If the costs in table 10 are compared with the shrinkage-stoping costs in table 7 cut-and-fill stoping may not appear to be much more expensive on a per-ton basis than shrinkage stoping, provided the last 5 mines in table 7 and the first 3 in table 10 are omitted. For deposits of similar width, however, shrinkage-stoping costs are con-

[57] Johnson, C. H., and Gardner, E. D., Cut-and-Fill Stoping: Inf. Circ. 6688, Bureau of Mines, 1933, 56 pp.

siderably lower. Thus, at the Eighty-Five mine shrinkage-stoping costs averaged about $1.50 per ton, whereas cut-and-fill-stoping costs average about $2.40 per ton.

In most of the mines using cut-and-fill stoping some sorting at least is done in the stopes, and cleaner mining is possible than with shrinkage stoping. The resulting higher grade of product and smaller tonnage handled, hoisted, and milled usually will offset the higher per-ton cost and result in a lower cost per unit of recoverable metal, which is the end to be sought.

Table 11 summarizes data on man-hours per ton at a number of mines employing cut-and-fill stoping as the principal mining method.

Table 12 shows the consumption of explosives in cut-and-fill stopes, and table 13 gives data on the consumption of timber.

ADVANTAGES OF CUT-AND-FILL STOPING

The principal advantages of cut-and-fill stoping are summarized as follows:

1. It is applicable to the mining of large or small deposits having moderately firm ore and weak walls dipping at an angle of about 45° to vertical and may also be employed in flatter deposits of considerable vertical height. Moreover, it may be used in low-dipping, thin deposits if special means are provided for moving the broken ore from the stopes to the haulage levels.

2. The filling forms a permanent support to the walls, preventing caving thereof, although it cannot be relied upon to support backs of large horizontal span.

3. Cut-and-fill stoping is flexible in that irregular ore margins and offshoots and stringers from the main vein can be followed and mined.

4. Dilution of ore with waste can be kept low; waste inclusions can be left in place or broken, sorted, and left in the stopes for filling; and a high percentage of the total ore in the deposit can be extracted.

5. The ore is removed as rapidly as it is broken, so that little capital is tied up in unavailable broken ore. Moreover, quick removal of the ore after it is broken is an advantage with some ores which are to be concentrated by flotation.

6. Little timber is required, except in the stringer-set-and-fill variation of the method; hence the fire hazard is low.

DISADVANTAGE OF CUT-AND-FILL STOPING

1. The ore itself must be fairly strong, especially in wide deposits, as the filling supports only the walls and not the back.

2. The rate of output from each stope is somewhat limited owing to delays occasioned by filling. The transportation, handling, and placing of filling adds to the direct stoping cost and to the cost of transportation. Filling must be provided promptly when and in the quantities required to maintain a steady production.

3. The cost per ton of ore is higher than that for shrinkage stoping but lower than that for square-set stoping under similar conditions.

TABLE 10.—*Direct stoping costs at mines employing cut-and-fill stoping*

Mine	Year	Variation of cut-and-fill method	Width of ore, feet	Direct stoping costs per ton of ore produced						
				Labor	Supervision	Compressed air, drills, and steel	Explosives	Timber	Other supplies	Total
Questa, N. Mex.	1929–30	Horizontal; largely resuing	1 to 1.5 average							[1] $8.51
Gold Springs	1931	Resuing	0.06 to 0.83 average	[2] $4.239	[2] $0.672	[2] $1.116	[2] $0.323	[2] $0.432	[2] $0.170	[2] 6.951
Do	1931	do	do	[3] 1.497		[3] .172	[3] .059	[3] .004	[3] .026	[3] 1.914
Lucky Tiger, Sonora, N. Mex.	November 1924	Mostly horizontal resuing; fill stoping in wide ore.	1.7 average; average stoping width, 3-5.	2.640	[illegible]	1.051	.505	.710	.225	[1] 6.740
Eighty-Five:					[4] .103					
No. 10 stope	1928–30	Inclined or rill	5.0 average	1.96	.407	.60	.26	.20	.08	2.37
No. 26 stope	1929–30	do	do	1.62		.33	.17	.27	.02	2.46
Ground Hog	March 1930	do	3.0 to 25.0	[5] 1.197		[5] .087	[5] .055	[5] .120	[5] .235	[5] 1.702
Matahambre, Cuba	1929	Horizontal [6]	More than minimum stoping width.	.893		.009	.095	.103	.090	1.267
Teziutlan, Mexico	July 1930 to June 1931.	do [7]	5.6 average	.377	[4] .095 .096	.252	.216	.107	.078	1.071
Pecos	1929	do		[8] 1.52		[9] .02	[8] .14	[8] .25		[8] 2.01
United Eastern	January 1917 to May 1926.	do	Maximum, 35 to 45.	1.760		[9] .136	.414	.415	.178	[10] 2.923
Champion	1927	do	6 to 50	1.340					.423	[11] 1.763
Do	Aug. 1, 1930, to Dec. 31, 1930.	Sublevel, inclined; 40 percent sorted as waste.	do	.794	.056	.198	.103	.031	.020	1.203
Pilares, Mexico	1929	Horizontal and inclined [12]		.839	.191	.176	.134	.248	.007	1.570
Morning	1928	Stringer-set and fill	6 to 30; 15 average	1.650	.142	.130	.114	.322	.055	2.371
Hecla	1928	do	40 maximum	1.020	.030	.216	.113	.357	.101	1.895

[1] Based on tons of sorted ore; 60 percent of stope rock rejected as waste. Includes supervision and surface expense directly chargeable to mining.
[2] Based on tons of sorted ore hoisted.
[3] Based on tons of crude ore broken.
[4] Per ton hoisted; large proportion of broken rock left in stope as fill.
[5] Costs are average of cut-and-fill and square-set stoping; inclined cut-and-fill stoping is used in narrow ore and square-setting in wide ore.
[6] 30 percent of ore mined by square-set stoping, 70 percent by cut-and-fill stoping.
[7] Mostly cut-and-fill stoping; some open stoping.
[8] Costs cover cut-and-fill stope operations only; about 20 percent of stope rock sorted out as waste in stopes.
[9] Power only.
[10] Includes tramming cost.
[11] Includes tramming, timbering, track, compressor, and drill expense.
[12] 59.4 percent horizontal, 17.6 percent inclined, 17.7 percent square-setting, 3.7 percent top slicing, and 1.6 percent shrinkage.

TABLE 10.—*Direct stoping costs at mines employing cut-and-fill stoping*—Continued

Mine	Year	Variation of cut-and-fill method	Width of ore, feet	Direct stoping costs per ton of ore mined						Total
				Labor	Supervision	Compressed air, drills, and steel	Explosives	Timber	Other supplies	
La Colorada, Mexico.	Jan. 1 to June 30, 1930.	Horizontal and inclined stopes with pillars.[13]	Wide	$0.375	$0.131	$0.188	$0.216	$0.198	$0.095	$1.203
Do	Jan. 1 to June 30, 1931.	Horizontal cut-and-fill stopes with pillars.[14]	do	.476		.100	.163	.014	.005	.758
Campbell:										
Stope 68	May 1926 to January 1930.	Inclined with double-lead sets	50 to 250	.620		.215	.135	.165		1.135
Stope 90	January 1928 to January 1930.	Inclined with single-lead sets	do	.70		.20	.10	.13		1.13
Stope 112	June 1929 to January 1930.	Inclined semishrinkage	do	.81		.35	.20	.26		1.62
Magma	1926	Inclined cut-and-fill	15 to 60	1.490	.257	.414	.150	.470	.058	2.840

[13] Stoping cost only; 40 percent inclined, 14 percent horizontal, 12 percent square-set, 15 percent shrinkage stoping, and 20 percent from development.
[14] Cut-and-fill stoping only.

TABLE 11.—*Stoping man-hours per ton of ore hoisted at mines employing cut-and-fill stoping*

Mine	Year	Width of ore (feet)	Variation of cut-and-fill method	Man-hours per ton of ore hoisted					Tons per man-shift, all stope labor
				Breaking	Shoveling	Filling	Timbering	Total	
Block P	1929	1 to 4	Horizontal; considerable resuing	1.638	1.812		0.306	3.956	2.0
Gold Springs	1931	0.66 to 0.83 average	Resuing	1.24	0.81	[1] 9.84		[4] 6.69	[2] 1.4
McIntyre	1932	10.0 average	50 percent horizontal cut-and-fill, balance square-set and shrinkage.	1.52		0.14		1.66	4.82
Do	do	do	Cut-and-fill only					.89	9.00
Mutatambre, Cuba	1928	More than minimum stoping width.	Horizontal cut-and-fill, 70 percent; square setting, 30 percent.	.60	.02	.67		2.28	3.51
Taxtutlan, Mexico	July 1930 to June 1931	5.0 average	Horizontal	1.068	1.768	1.036	.330	4.208	1.86
Pecos	1929		Horizontal cut-and-fill and square-setting.[3]	.774	.865	.808	.864	2.806	2.70
Do	1929		do	.917	1.016	.451	1.022	3.406	2.31
Champion	Aug. 1 to Dec. 31, 1930	0 to 50	Sublevel, inclined	.55	[4] .73	.21		1.49	5.39
Hecla	1928	40 maximum	Stringer-set-and-fill	.256	.451		.016	1.318	6.07
United Verde	1930(?)	Wide	Horizontal cut-and-fill	.196	.547	.400		1.148	6.97
Magma	1928	16 to 60	Inclined					1.706	4.88
Eagle	1930(?)	Wide	Horizontal, with square-set aisles; pillars mined with stringer sets.	.28	.41	.18	.37	1.24	6.45

[1] Sorting.
[2] Man-hour per ton of rock broken, 0.874; tons of rock broken per man-shift, 9.15.
[3] 52 percent horizontal cut-and-fill stoping, 37 percent square-set stoping, and 11 percent shrinkage stoping and pillar mining.
[4] Includes sorting an average of 40 percent of rock broken.

TABLE 12.—*Consumption of explosives in cut-and-fill stopes*

Mine	Kind and strength of explosive used	Explosive used per ton of ore broken, pounds
Block P	Special gelatin dynamite, 30-percent	2.508
Matahambre	Gelatin dynamite, 30-percent	.24
McIntyre	Gelatin dynamite, 40-percent	1.00
Ground Hog	(?)	.87
Teziutlan	Gelatin dynamite, 40-percent	.847
Pecos	Gelatin dynamite, 35- and 40-percent	[1] .750 [2] .650
Champion	Gelatin dynamite, 40-percent	.60
Morning	Gelatin dynamite, 25-percent	.249
	Gelatin dynamite, 35-percent	.273
	Gelatin dynamite, 40 percent	.071
	Total	.593
Hecla	Gelatin dynamite, 30-percent	.533
La Colorada	Gelatin dynamite, 85 percent 40-percent strength Gelatin dynamite, 15 percent 60-percent strength	1.37
Magma	(?)	.70
United Verde	Gelatin dynamite, 60-percent	[3] .65
Eagle	Gelatin dynamite, 45-percent, and ammonia dynamite, 40-percent.	[3] .53

[1] 1929. [2] 1928. [3] Cut-and-fill stopes only.

TABLE 13.—*Consumption of timber in cut-and-fill stopes*

Mine	Consumption of timber per ton of ore mined: Framed timber, board feet	Consumption of timber per ton of ore mined: Round timber and lagging	Mine	Consumption of timber per ton of ore mined: Framed timber, board feet	Consumption of timber per ton of ore mined: Round timber and lagging
Matahambre	0.802	[1] 0.125	Hecla	11.207	[5] 1.234
Teziutlan	5.647		La Colorada	7.10	
Pecos		[2] 8.9 [3] 2.6	Magma	[6] 17.00	
Champion		[4] 8.1	United Verde	[7] 6.06	
Morning		[4] 12.7	Eagle	[7] 7.3	

[1] Piece of prop timber.
[2] 1929; linear feet.
[3] 1928; linear feet. Half of output from cut-and-fill stoping and half from square-setting.
[4] Board feet.
[5] Linear feet.
[6] Total for all stopes and includes maintenance.
[7] Cut-and-fill stopes only.

STULLED STOPES

As already stated there is some question as to whether the method of stoping in which a systematic scheme of stull timbering is used to support roof and walls should be classified as a distinct stoping method. Stull timbers are used in conjunction with other mining methods to support local patches of weak ground in most mines, but in such instances they are incidental to stoping. Sometimes stulls are spaced at regular intervals throughout the stopes and serve as the principal or only support of the walls or back and incidentally as supports for working platforms.

Although regular stull timbering is used widely in working narrow veins of fairly strong ore with moderately strong walls, recent mining literature contains few descriptions of this method, probably because it is used principally in small mines.

The method may be employed in thin tabular deposits dipping at all angles from flat to vertical.

EXAMPLES OF STULLED STOPING

CONGLOMERATE MINE, CALUMET, MICH.

The mining methods employed in the Conglomerate lode have been described by Vivian.[56]

The Calumet Conglomerate ore body occurs in a felsitic conglomerate. At Calumet the bed is 10 to 12 feet thick near the surface, widening out in the central part of the property to about 20 feet at the eighty-first level, which is 8,100 feet below the surface measured on the dip, and narrowing again in the lower levels. The dip ranges from 38° at the surface to 36° at a vertical depth of 1 mile.

The conglomerate is interstratified with volcanic rocks; thick beds of trap lie above and beneath it. The lode consists of felsite and quartz-porphyry pebbles cemented together with quartz, calcite, and native copper. Within the conglomerate occur lenticular layers of sandstone which, although barren of copper themselves, usually lie between the commercially mineralized parts of the lode and must be broken with the vein material. The entire hanging-wall trap is checkered with slips and jointing places, most of which are tight and dry and run at angles of 30° to 70° with the bedding. The footwall of the lode is the highly fragmental, amygdaloidal capping of the underlying trap. In general, it is very friable and contains a large proportion of soft minerals. When the weight of the hanging wall is concentrated upon an unmined pillar of lode material, which is very hard, the pressure of the hard pillar on the weaker footwall causes the latter to burst upward into the opening. This condition had an important bearing on the decision to adopt a retreating system of mining. The lode had been mined to great depths over wide areas, and the weight of the superincumbent mass of broken rock over the mine workings is directly responsible for air blasts and rock bursts.

The advancing system of mining was used to a depth of 6,000 feet on the dip, or about 3,500 feet vertically. From this elevation to the present bottom stopes at a vertical depth of about 5,500 feet the retreating system has been used entirely. The bottom levels are developed from inclined shafts sunk from the eighty-first haulage level. Levels are 100 feet apart measured on the dip, and on each level drifts are driven to the ore or section limits where stoping begins and retreats toward the shaft.

Figure 44, *A*, is a plan on the plane of the lode showing the general scheme of mining and the relative positions of the retreating stopes on several levels. Figure 44, *B*, shows the method of carrying the stope faces and the systematic arrangement of stull timbers in a single stope.

In the thicker parts of the lode square-set timbering was employed, and individual stopes were 200 feet long. Below 5,500 on the dip the number of stopes over 20 feet thick decreased rapidly, and

[56] Vivian, Harry, Deep Mining Methods, Conglomerate Mine of the Calumet & Hecla Consolidated Copper Co.: Inf. Circ. 6526, Bureau of Mines, 1931, 20 pp.

on this account and because of the excessive cost of framed square-sets a gradual transition was made to stull timbering, which is now used entirely for temporary support in all stopes. The stope length was cut to 100 feet at the same time. Standard practice now calls for four rows of stulls 18 inches to 2 feet in diameter, set in batteries of two alongside each other with approximately a 7-foot clear space between near timbers horizontally and on the dip. These double batteries cover about 40 percent of the height of the stope, and no matter how firm the hanging wall may appear to be, double stulls are installed to a height of 40 feet above the level. Above this, the stulls are placed singly, the distance between centers increasing according to the nature of the ground in each particular stope. Experience has shown that during the time required to complete a stope (about 100 days) the maximum thrust of the hanging wall manifests itself at a point about one-third the total height of the completed stope, or just below the fourth double row above the level. Row 4 is just below the center of the stope and must hold for about 60 days after it is placed. Each successive row above this point is required to stand for a shorter time and is located under hanging wall which receives less pressure due to the support afforded by the caved material in the stope immediately above. Frequently the upper 20 to 25 feet of the stope require no timber, or at most only a few odd stulls.

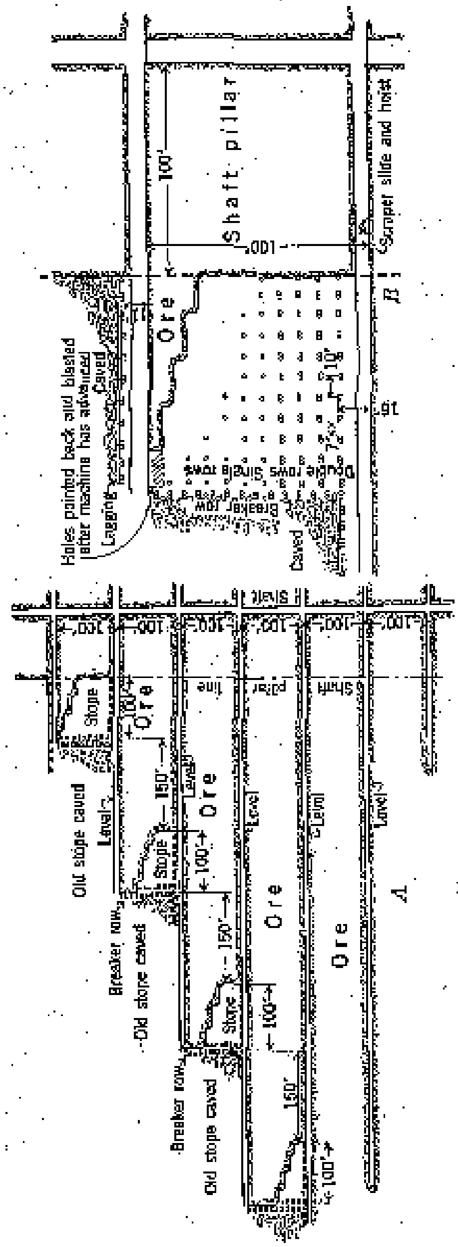

FIGURE 44.—Stull stoping, Conglomerate mine, Michigan: A, Plan on plane of lode showing position of stopes; B, method of carrying stope faces and arrangement of stull timbers.

Three and sometimes four machines are operated simultaneously in each stope. The first or "cutting-out" slice of each stope is started at the end of the stope nearest the shaft and is extended toward the previously completed stope, which is usually well caved by the time the slice reaches it. Each succeeding slice, which is about 6 feet high, is started from the caved-stope end and driven toward the shaft; thus the short starting raises which would be required if the slice were started at the other end are unnecessary, and the dangerous pillar projection at the caved end just before completion of the slice is eliminated.

Just before each stope has been mined to the levels above a double row of "breaker" stulls is placed from bottom to top, within about 3 feet of the solid end of the stope. The row consists of heavy logs at least 2 feet in diameter, set with a clear space of 3 feet between pairs, thus leaving a safe area 3 feet wide into which the hanging wall cannot cave until the new stope has been sliced out.

A complete mining operation on each side of a shaft consists of a series of four retreating stopes on four successive levels, each stope leading the one below by 150 feet. This lead is considered the minimum distance over which the floor arch should remain intact during the active life of the stope above.

After the stopes are mined out the hanging wall crushes the timbers gradually and finally breaks, filling the stope behind the breaker stulls. No attempt is made to force hanging wall to collapse by blasting out the stulls, but the time of collapse is regulated by increasing or decreasing the number of stulls placed during active stoping.

The system described is virtually a longwall retreating system. By its use rock bursts have become rare and are looked for only in the last or next to the last stope at the shaft pillar.

Mounted 3½-inch Leyner-type machines are used in stoping. Placing holes in the stopes is left largely to the discretion of the miner, supervised by the shift boss, and the arrangement varies considerably, owing to the changeable nature of the face due to pressure, slabbing, or jointing.

The conglomerate rock tends to break into slab-shaped pieces, so that in spite of the relatively flat dip of the lode most of the stope rock runs to the level by gravity. About 15 percent of the rock from the upper stope faces must be scraped dwon to the level. The same scraper is used to scrape the rock from the floor of the level up a scraper slide and into the mine cars.

The cost of stoping in 1927 was as follows:[89]

	Cost per ton of ore milled[90]
Labor	$1.605
Supplies	.543
Power	.111
Total	[91] 2.259

During 1927 the consumption of explosives in stopes was 0.711 pound per ton of rock milled. In January and February 1931 the average breaking cost was $0.626 per ton and the average stoping wage $5.04 per shift.

[89] Crane, W. R., Work cited.
[90] Items include timbering, tramming, track, compressor, and drill expense.
[91] Deducting the tramming cost, which is given as $0.847 per ton, the direct stoping cost was $1.012 per ton.

During 1930, 872,834 tons were mined and hoisted, of which 831,-870 tons were from stopes and 40,964 tons from development. Labor performance in stopes was as follows:

Labor performance in stopes, Conglomerate mine, 1930

Occupation:	*Man-hours per ton*	*Tons per man-shift*
Breaking	0.386	20.72
Timbering	.481	16.63
Mucking	.397	20.14
Total	1.264	6.33

Consumption of explosives was 0.75 pound of 40-percent dynamite per ton of ore and consumption of timber 6.19 linear feet per ton.

WITWATERSRAND, AFRICA

Stulled stopes have been employed extensively on the Central Rand, both in underhand and overhand stopes. The practice recently was discussed briefly by Beringer.[32] Figure 45, taken from

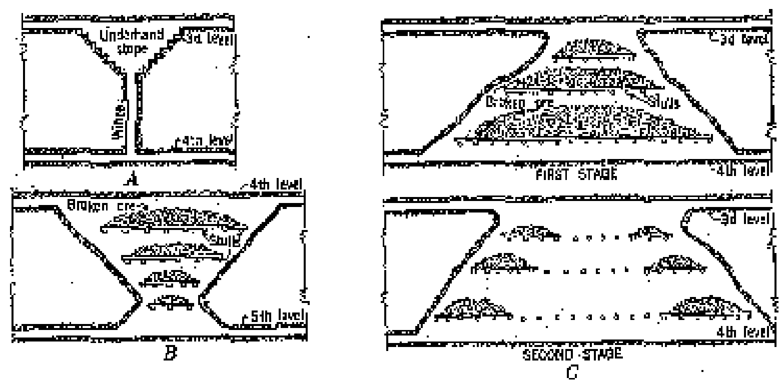

FIGURE 45.—Evolution of stoping practice at a Witwatersrand mine, Africa (after Bernard Beringer, Eng. and Min. Jour.): *A*, Early system, open underhand stope; *B*, underhand stoping onto stulls; *C*, overhand stoping onto stulls.

Beringer's article, shows the manner of carrying the stope faces and stoping onto stull timbers. According to Beringer the underhand stoping system shown in figure 45, *B*, was an improvement on the earlier method of underhand stoping without timber. (See fig. 45, *A*.) The disadvantages of the earlier method were: (1) Transportation of the ore from the face to the level below by gravity was interfered with by the benches from which it had to be shoveled down the stope faces; (2) misfires are possible because many of the holes are wet, and the misfires are not seen readily; (3) a fall of hanging wall may roll down and kill the drillers; (4) ventilation is poor because, when the winze becomes full of broken ore there is no through current of air; (5) inconvenience is caused by accumulated ore at the points.

[32] Beringer, Bernard, The Witwatersrand; Evolution of Mining Practices Especially in the Central Rand: Eng. and Min. Jour., vol. 131, 1931, pp. 322–324.

By the stulled stoping method shown in figure 45, *B*, ventilation was improved to avoid delay in reentering the stopes after blasting, and support to the hanging wall was strengthened. The face is carried at 45° from the horizontal, as in the open stopes, and stulls are placed across the stope at 20-foot intervals. Broken ore is allowed to accumulate on these, thus supporting a larger area of hanging wall and affording overhead cover for the drillers. In heavy ground stulls may be strengthened with filled cribs. Accessibility and ventilation were improved, safety was increased, and drawing of ore was accelerated by the change to stulled stopes.

A greater improvement came when underhand stoping was abolished, where dips were 45° or steeper, and overhand stoping introduced. With overhand stoping onto stulls, as shown in figure 45, *C*, mining is more economical, no shoveling is required, stope faces are left clean by the blast and "bootlegs" can be examined readily, safety is increased because a fall of hanging wall can hurt only the man directly beneath and a fall from the face can hurt no one, and the air current is directed up the stope faces, insuring perfect ventilation. This method is applicable to deposits up to 12 feet in width which dip 45° or more. The stulls are placed 20 feet above each other and should be kept well-loaded with broken ore to afford standing room for the drillers and support a larger area of hanging wall. As the stope faces advance the stulls are drawn toward the faces, and the accumulated ore drops through to the chutes.

WRIGHT-HARGREAVES MINE, KIRKLAND LAKE, ONTARIO

Stull stoping was introduced at the Wright-Hargreaves mine in 1929, and certain details of practice employed at Butte, Mont., were applied. The method is similar to that shown in figure 7, page 12, and is used in some of the narrower stopes where there are bad patches of gouge and wall rock which slough off it not well supported. The broken ore is not allowed to accumulate on the stulls but is diverted immediately to the chutes, wing slides being used when necessary to eliminate hand shoveling. Slabs of waste were left in the stopes on stull timbers as shown. By this system it was possible to stope with less overbreak of the vein in narrow ore and less dilution than when shrinkage stoping was employed, and a higher-grade product resulted. The stopes are carried through from level to level without leaving any floor pillars. When a stope has been worked to the level above it is filled with development waste. When stopes are to be carried above the level the bottom of the level is timbered with double heavy stulls and floored over, and stoping is then resumed above.

Stoping costs for 1930, taken from the annual company report, were $1.926. This was the average cost in stulled stopes, shrinkage stopes, and a few square-set stopes.

SQUARE-SET STOPING

Square-set stoping originated on the Comstock lode in Nevada in 1860; it was devised to mine a wide body of ore which was so soft and crumbling that pillars could not be used to support the roof. The method was so successful that later it came into general use in the Western States and was frequently employed where there was no real need for it and where other and cheaper methods would have served as well.

It is still used widely, especially in the West, and finds its best application in the mining of moderate- or high-grade bodies of weak ore with weak walls, where neither the ore nor the walls will stand unsupported for an appreciable time without permanent support even over small spans and where caving and subsidence of the overlying strata must be prevented. The system is applicable to tabular, lenticular, and irregular deposits of small or great width dipping at angles from flat to vertical; it is often employed for mining crown or level pillars and between caved or filled stopes which have been mined by other stoping methods.

Square-setting usually is adopted as a last resort but has a special field for mining ore bodies not minable by other methods. The system is very flexible and may be altered readily to meet changing ground conditions.

Square-set stopes may be mined with or without contemporaneous filling, but today square-set stoping is applied principally under conditions where the ground is so heavy that prompt filling is required for permanent support and the square-set timbering serves only to hold the back and walls during extraction of the ore from relatively small blocks or slices.

The square-set stopes may be mined in horizontal slices or "floors", with inclined faces, or in vertical slices. Horizontal or rill sections usually are employed in wide ore bodies where the greatest pressure is from the walls. When the greatest pressure is from above the sections usually are carried up vertically. In weak ore bodies the tendency is to use vertical rather than horizontal slices. Loose or running ground generally is mined in vertical slices, beginning at the top and proceeding downward. Several methods of attack may be used in the same vein to fit local ground conditions.

The examples in the following pages typify conditions under which square-set stoping is employed and the principal variations of the method. Reference is made to the discussion of Gardner and Vanderburg [93] for further details of square-setting practice.

SQUARE-SET STOPING, USING HORIZONTAL SECTIONS

ARGONAUT MINE, JACKSON, CALIF.

Mining methods of the Argonaut mine have been discussed by Vanderburg.[94] General conditions favorable to mining at this mine are partly offset by the following: (1) Increasing cost of operation incident to the great depth to which mining has progressed; (2) heavy, swelling ground encountered and consequent large outlay for maintenance to keep the ground open; and (3) comparatively low grade of the ore. The Argonaut vein occupies the fissure of a reverse fault and cuts through beds of greenstone, schist, and slate. It strikes about N. 20° W. and has an average dip of 63° northeast.

[93] Gardner, E. D., and Vanderburg, William O., Square-Set System of Mining: Inf. Circ. 6691, Bureau of Mines, 1933, 73 pp.

[94] Vanderburg, William O., Mining Methods and Costs at the Argonaut Mine, Amador County, Calif.: Inf. Circ. 6311, Bureau of Mines, 1930, 14 pp.

As depth is attained the vein tends to become flatter. The maximum length of the ore shoot is 1,100 feet, the maximum width 65 feet, and the average width mined about 20 feet. The vein is persistent and has been followed from the apex down to the 5,400-foot level (at

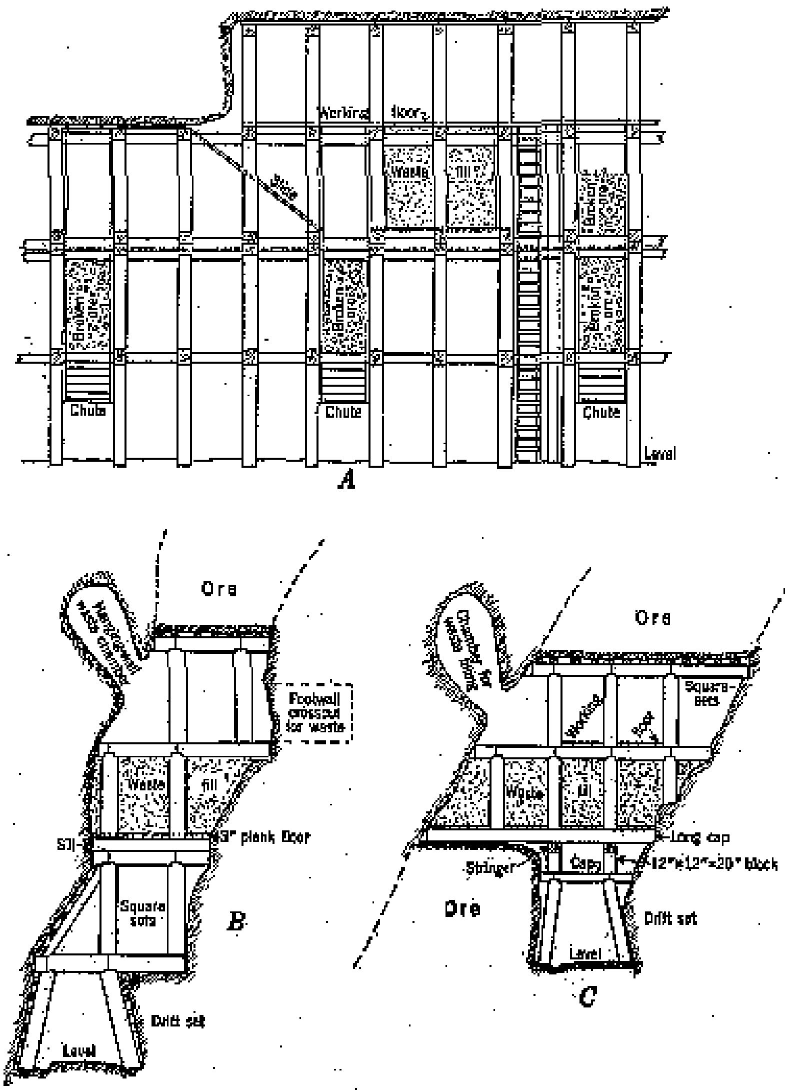

FIGURE 46.—Square-set stoping, Argonaut mine, California: *A*, Vertical longitudinal projection; *B*, vertical cross section showing square-sets directly on drift sets; *C*, vertical cross section showing square-sets on long caps and stringers.

the end of 1929) measured on the dip. The quartz filling pinches and swells along both strike and dip but never dies out entirely.

The vein is accompanied on one or both walls by gouge ranging from several inches to several feet in thickness. The walls are soft and fractured so that sloughing will occur if the openings are not supported promptly. The vein itself is traversed by planes of weakness so that lagging of the backs of all openings is required to prevent caving. The rock masses are more or less mobile, and their weight is transferred from point to point as their equilibrium is disturbed by new openings. Timber alone will not provide adequate support over a large area, and as stoping progresses timbering must be followed promptly by waste filling.

Figure 46 shows the system of stoping and stope timbering at the Argonaut mine. The level interval is 100 feet down to the 3,000-foot level and 150 feet from there to the bottom. Drifts are extended in ore along the footwall on each level. These drifts are 6 by 8 feet in cross section and are timbered with regular drift sets. (See fig. 46, *B* and *C*.) When a drift has been extended several hundred feet from the shaft a raise is driven to the level above. Stopes are silled out on the first floor above the drift, and the sills are timbered either with regular square-sets resting directly on the drift sets (fig. 46, *B*) or with regular sets on long caps and stringers (fig. 46, *C*). Flooring is laid on the stringers to hold the subsequent waste filling.

Stoping is carried upward in sections 100 feet long or as long as the distance between manways. Several sections usually are mined simultaneously, although one section may be several sets lower or higher than the adjoining ones. Virtually all the vein material in the ore shoot is mined without sorting, and no pillars are left. Generally only enough ground is removed at one time to make room for standing one set of timber. After a section has been mined one set high and across the vein for the full length of the section another floor is begun. Filling with waste is carried on along with stoping, so that only one floor is open at a time. Manways are carried up as the stope advances, and chutes are built in every fourth drift set or at 25-foot intervals. Whenever possible, the broken ore is diverted into the chutes by plank wing slides built in the square-sets. Any ore that cannot be handled by gravity in this manner is shoveled into the chutes by hand.

Square-set posts are framed at one end only and are of round timber 14 to 18 inches in diameter and 7 feet in over-all length. Caps are 12 by 12 inches by 5 feet 4 inches long; girts are 8 to 12 inches in diameter, 4 feet long, and unframed. A square-set is 7 feet 10 inches high and 5 feet 4 inches by 5 feet from center to center of posts. The standard set contains 209 cubic feet, or 16.72 tons of ore. Filling is obtained by driving inclined chambers into the hanging wall or, if the hanging wall is weak and tends to cave, from crosscuts into the footwall.

Direct stoping costs during September 1929 were as follows:

Direct stoping cost, Argonaut mine, September 1929

	Cost per ton of ore hoisted		Cost per ton of ore hoisted
Labor	$1.740	Timber	$0.335
Supervision	.052	Other supplies	.057
Compressed air, drills, and steel	.101	Total direct-stoping cost	2.433
Explosives	.148		

Wage rates were as follows: Miners $4.50, muckers $4, timbermen $4.75, and tool nippers $4.50.

Ore breaking and timbering required 2.054 man-hours per ton of ore, waste breaking and filling 0.347 man-hour, and shoveling 0.475 man-hour, a total of 2.876 man-hours of stope labor per ton, equivalent to 2.78 tons per man-shift. The consumption of explosive was 1.01 pounds of 30-percent gelatin dynamite per ton of ore. The consumption of timber amounted to 7.05 board feet of sawed stock and 1.08 linear feet of round timber and lagging per ton.

PAGE MINE, PAGE, IDAHO

Mining methods at the Page mine have been described by Berg.[85] The ore occurs in two distinct fissure veins in quartzite. They strike S. 70° to 80° W. and dip S. 40° to 60°. There are no well-defined walls, and the enclosing quartzites are broken and shattered to produce a zone extending 20 to 30 feet into the walls of the vein proper; this causes heavy, swelling ground and necessitates the use of closely filled square-set stopes for mining the ore. Some sorting is done underground, but most of the ore breaks so fine that the material either is rejected for filling or sent out as ore. The veins pinch on both strike and dip to mere gouge and swell at times to 20 feet in thickness. Small, irregular patches and bodies of ore occasionally are found running back into the hanging wall of the vein, but these are confined to the fractured zone.

Breast stoping with light drifters is preferred to overhand stoping with stoper drills, because it controls breaking better and eliminates heavy steel losses and stoper machine repairs.

Stopes are carried in blocks about 250 feet long and taken in one lift 300 feet high. Ore passes are 35 feet apart and are cribbed. Occasionally chutes are so flat, due to irregularities in dip of the veins, that water flooding is required to make the broken ore run.

Standard square-set timbering with cap-butting framing is employed. Caps average 12 inches in diameter and posts 9 inches in diameter, and girts are of 6- by 8-inch sawed timber. The ground is so heavy that the timber is placed as soon as room is made for a set. It is desirable to keep the stopes filled to within three floors or less from the back. Crosscuts are driven into the footwall to obtain waste filling material; they are begun 5 by 7 feet in section and 25 to 30 feet from the stope are enlarged to 8 by 12 feet. Waste is dragged from these crosscuts into the stopes by single-drum tugger hoists.

Direct stoping costs during 1928 are given by Berg, as follows:

Direct stoping cost, Page mine, 1928

	Cost per ton of ore hoisted		Cost per ton of ore hoisted
Labor	$1.43	Timber	$0.26
Supervision	.15	Other supplies	.04
Compressed air, power, drills, and steel	.16	Total direct stoping cost	2.15
Explosives	.11		

Wage rates were: Timbermen $6, timbermen helpers $5.25, miners $5.50, and shovelers $5.

[85] Berg, J. E., Mining Methods at the Page Mine of the Federal Mining & Smelting Co., Page, Idaho: Inf. Circ. 6372, Bureau of Mines, 1930, 8 pp.

PECOS MINE, TERERRO, N. MEX.

Cut-and-fill stoping at the Pecos mine was discussed in the section on cut-and-fill stoping. It was stated that in 1929 cut-and-fill and square-set stoping were employed about equally. The same general scheme of operations applies to both methods, except that in square-set stoping manways, extraction raises, and gob lines are provided by lacing off square-sets. Square-sets are of round timber and are 5 feet square by 6 feet, 8 inches high in the stopes and 8 feet high on the sill floor.

Direct stoping cost per ton in square-set stopes only was as follows in 1929:

Direct stoping cost, Pecos mine, square-set stoping only, 1929

	Cost per ton of ore mined		Cost per ton of ore mined
Breaking, labor	$0.62	Mining supplies	$0.12
Shoveling, labor	.57	Explosives	.15
Timbering, labor	.79	Timber	.40
Filling, labor	.18		
Total labor	2.16	Total direct stoping cost	2.83

The total cost of $2.83 per ton for square-set stoping compares with $2.01 per ton for cut-and-fill stoping at this mine.

BLACK ROCK MINE, BUTTE, MONT.

Mining methods at the Black Rock mine have been described by McGilvra and Healy.[96]

The Black Rock mine is on the eastern extremity of the Rainbow vein, which has a general east and west strike and dips 80° to the south. The vein ranges from a few feet to 120 feet in width, and stoping widths are 1 set to 12 sets (6 to 70 feet). Most of the production comes from this vein, although other veins averaging about 6 feet in width have commercial importance. The ore ranges from a soft, crumbly sphalerite to hard, flinty material composed of quartz and sphalerite. The country rock is granite, which is intensely altered near the vein. The vein structure presents a banded arrangement of ore and waste parallel to the walls. In general, the ore and wall rocks are not strong enough to support themselves, even over small spans, and artificial support is required as soon as possible after an opening is made.

The level interval is 100 feet to the 1,900-foot level and 150 feet from there down to the bottom stoping level, which is at 3,900 feet.

Square-set stoping is employed throughout the mine; it gives maximum support to the walls and back and fills the need for a flexible system of mining. Waste filling is kept as close to the stope face as possible. Often bulkheads are carried through from one level to the next as stoping progresses to give necessary additional support.

Variations of square-set stoping are necessary to suit the different conditions encountered. These conditions are: (1) Ore and walls

[96] McGilvra, D. B., and Healy, A. J., Methods of Mining at the Black Rock Mine, Butte & Superior Mining Co., Butte District, Mont.: Inf. Circ. 6370, Bureau of Mines, 1930, 16 pp.

strong enough to permit carrying the stope upward with square-sets and following closely with waste filling; (2) loose and running ground that would cave if worked upward, which requires underhand stoping by carrying the square-sets downward; and (3) ground intermediate between that of conditions (1) and (2), which is mined by top slicing with square-set timbers.

Standard square-sets are 5 feet 10 inches by 5 feet square in plan and 8 feet high and are of the cap-butting type. They are of round timber 10 to 12 inches in diameter. Caps and posts are framed alike and interchangeable.

Drifts are driven along the footwall and timbered with framed sets with 8-foot posts. Stopes are silled on the first floor above the level in overhand stoping, and chutes are installed at 30-foot intervals along the drift. Stopes are carried up in short blocks 4 to 10 sets long with a chute and manway at the end which serves later as a supply raise for the next block. Where the vein is wide the hanging-wall half is stoped first and the footwall half later. Two-inch lagging is used to retain the waste filling on the end of the stope adjoining ore in place.

Usually only enough ground is blasted out at a time to make room for a single set of timber. Waste between bands of ore usually is blasted separately and dropped directly into the gob. Where it is impracticable to blast the waste separately the round is blasted onto the floor, which is covered with plank, and the waste is sorted out by hand.

Stopes more than two sets wide ordinarily require bulkheads in addition to the square-sets and waste filling. Cribbed bulkheads usually are sufficient, and when the ground over a bulkhead has been mined the bulkhead is moved up a floor and the space below filled.

In underhand square-setting a raise first is put through from level to level. From the raise a cut is taken on the floor below the sill. Booms are placed as shown in figure 47, *A*, to catch up the sill timbers. These booms are of 5- by 10-inch timber and 13 feet long. There are two methods of holding the timbers above while the ground is being taken out for the set below: (1) If down-weight is excessive, booms are used for every set, and two pairs of booms are employed, the upper pair being moved down as mining progresses; (2) if side pressure is excessive, a diagonal stull will hold the cap in place while the ground is being worked out for posts. After the first set is in, the ground for the set directly below is taken out. Mining progresses about 50 feet downward in this manner. At this point an intermediate drift is driven along the footwall as a tramming sublevel for tramming the ore back to the chute raise. (See fig. 47, *B*.) Starting at the top again another set is mined down alongside the first one on the hanging-wall side. This procedure is followed until a line of sets is mined from foot to hanging wall and from the upper sill to the intermediate level. The next lines of sets are mined in the same way. As stoping progresses waste filling is put in and held in place by small poles or broken lagging. One set in each line always is kept open as a temporary chute and manway. The ore below the intermediate level is stoped in the same manner as that above. The upper sill usually is abandoned before or during mining below. Access to the stopes is gained through footwall laterals and short crosscuts.

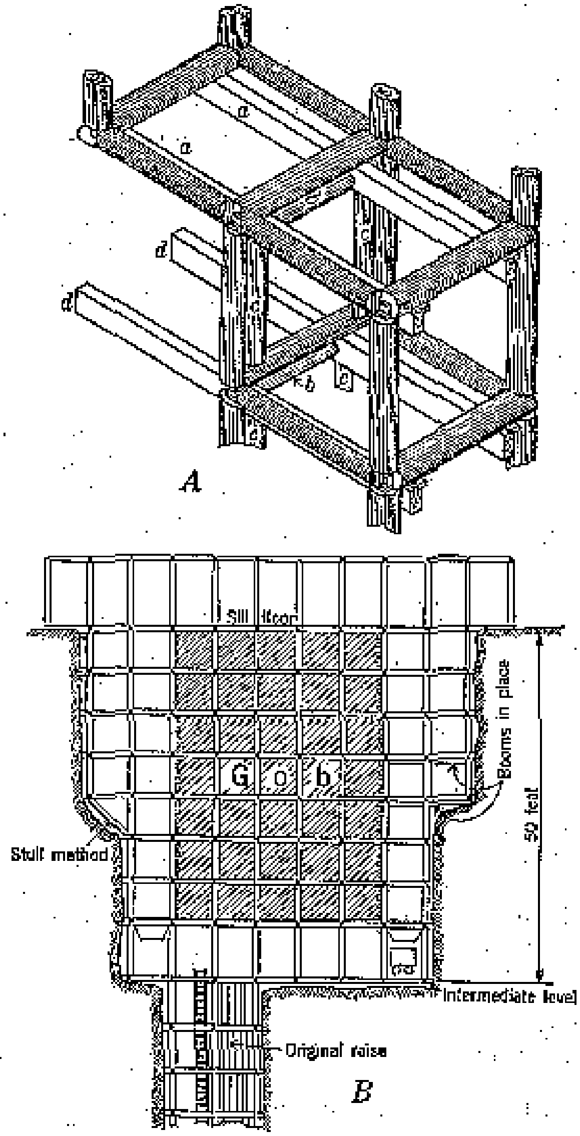

FIGURE 47.—Underhand square-set stoping, Black Rock mine, Montana: A, Method of using booms—a, upper booms, b, braces between booms, c, short posts under booms, d, lower booms; B, general section through stope.

In top slicing with square-sets the slices are mined 1 to 3 floors high. When more than one floor is to be mined a drift is run to the end of the ore block on the footwall at the lower floor and timbered with square-sets. The floor above then is mined back to the ore chute, starting at the end of the block. When all the ore above the slice floor has been mined out a new slice is begun below and mined in the same manner. No filling is employed, and although the timbers are not blasted out the upper slices soon close in and fill themselves.

SILVER KING COALITION MINE, PARK CITY, UTAH

Stoping methods at the Silver King Coalition mine have been described by Dailey.[57] Both lode deposits and bedded replacement deposits occur in this mine, the latter furnishing 95 percent of the ore. Bedded deposits usually are lenticular in outline, with irregularly lobed margins. Some lobes or arms extend hundreds of feet from the main deposits. Bedded deposits have been continuously productive for more than 10,000 feet along their strike. The lode deposits are found in fissures that appear to be the channels through which the mineralizing solutions circulated and formed the replacement deposits in the limestone beds. Ore minerals are galena, pyrite, sphalerite, tetrahedrite, and their oxidized products.

In the Silver King section of the mine a stringer-set-and-fill system (using timbering similar to that in the stringer-set-and-fill system described in the section on cut-and-fill stoping) is employed. The sets consist of unframed 8- by 8-inch caps 10 feet long supported on three round posts 8 to 10 inches in diameter and 6 feet 6 inches long. Round girts 4 feet 4 inches long either are spiked to the posts or held in position by small blocks nailed to the posts under the girts. Parts of the larger stopes are filled with waste as stoping advances to support the hanging wall, all available timbers being removed before filling so that they can be reused.

In the Silver Hill section of the mine, where the ground is heavier, the stringer sets are framed. The posts and caps are of 8- by 8-inch and the girts of 4- by 8-inch timbers. In flat, heavy ore bodies, where it is not practicable to advance the stope as one continuous face, the ore body is mined in sections 60 feet wide.

Broken ore is moved down the dip of the stopes by hand shoveling, wheelbarrows, or cars. Between 15 and 20 percent of the material broken is sorted out underground and returned to the stopes for filling. If enough waste is not supplied from this source waste drifts and raises that have merit as prospects or as ventilation and haulage openings are driven.

Direct stoping costs during the year 1929 are given as follows:

Direct stoping costs, Silver King Coalition mine, 1929

	Cost per ton of ore hoisted[58]		Cost per ton of ore hoisted[58]
Labor	$2.454	Timber	$0.249
Supervision	.130	Other supplies	.024
Compressed air, drills, and steel	.472	Total	3.502
Explosives	.173		

The wage scale was as follows: Miners $4.50, machinemen $4.75, timbermen $4.75, timbermen helpers $4.25, shovelers $4.25, and tool nippers $4.50.

[57] Dailey, M. J., Mining Methods and Costs of the Silver King Coalition Mine Co., Park City, Utah: Inf. Circ. 6371, Bureau of Mines, 1930, 12 pp.
[58] Of the rock broken, 20 percent is sorted out and returned to stopes for filling.

Labor performance in stopes, Silver King Coalition mine, 1929

Occupation:	Man-hours per ton	Tons per man-shift
Breaking	0.934	8.56
Timbering	.683	11.71
Shoveling and sorting	2.154	3.71
Total	3.771	2.12

SQUARE-SET STOPING, USING RILL STOPES

A rilled section in square-setting permits waste filling to be run in the stope where necessary, with a minimum of hand shoveling. Moreover, fewer chutes need be maintained through the fill to receive the broken ore. Where much sorting is to be done horizontal sections usually are preferred.

GROUND HOG MINE, VANADIUM, N. MEX.

Cut-and-fill stoping at this mine already has been discussed under Cut-and-Fill Stoping. It was stated that cut-and-fill stoping is employed where the vein is narrow and square-set stoping where it is wide.

Where square-set stoping is used the sill floor is taken out the full width of the ore and timbered with square-sets. The first floor above then is stoped and timbered, after which the raises are widened and timbered one set wide across the vein and up to the level above. Starting at the center raise of a 100-foot block on the second floor above the sill, three lines of sets are stoped from wall to wall on each side of the raise. On the floor above two lines of sets are taken in the same manner, and on the next floor above that one line of sets is taken on each side of the raise. This forms a pyramidal stope with the raise in the center. Waste is run in through the raise to fill the stope as completely as possible; then a floor is laid on the fill. Stoping is continued on each side of the raise until the toe of the rill is within 10 feet of the next chute, after which the last two sets are carried flat to furnish a shoveling platform where sorting can be done.

PARK UTAH MINE, PARK CITY, UTAH

Except for a small amount of bedded ore in the Wasatch limestone all the ore at the Park Utah mine occurs in fissures.[69] The width of the fissures ranges from 3 to 80 feet and the dip from 40° to 55°. The ore shoots are lenticular and range in length up to a maximum of 900 feet. The fissures are banded with alternating layers of sulphides and altered limestone. The bands constitute planes of weakness so that the ore, which is of medium hardness, breaks well. The walls of the veins usually are weak and will slough if left unsupported. Where the hanging wall is quartzite the rock is brecciated for about 50 feet from the vein and must be well supported to prevent caving. Faults cut the veins at acute angles, forming wedge-shaped masses of rock which cave readily into the stopes if not

[69] Hewitt, E. A., Mining Methods and Costs at the Park Utah Mine, Park City, Utah: Inf. Circ. 6290, Bureau of Mines, 1930, 18 pp.

supported. As a rule there is a clear line of demarcation between the vein and the hanging wall, whereas the footwall is poorly defined. Two classes of ore are mined—siliceous silver ore which is shipped directly to the smelter and lead-zinc-silver sulphide ore which is milled.

Square-set stoping with filling is employed in all stopes more than 12 feet wide.

The square-set mining system employed at the Park Utah mine is an adaptation of rill stoping and was planned primarily to permit the waste filling to flow to its destination with a minimum of handling. The first section is 100 feet or 20 sets long, 6 sets high, and the full width of the vein, which may be 12 to 80 feet. (See fig. 48.) Succeeding sections adjacent to the first are 50 feet or 10 sets long and 6 sets high. The sill floor is begun from the top of the footwall drift. When the section represented by blocks *A–W* and *A–E* (fig. 48) is mined 6 sets high, stoping is begun in blocks *B–W* *B–E*, *A2–W*, and *A2–E* simultaneously by horizontal slices one set high. The square-sets on each slice are placed in position at the

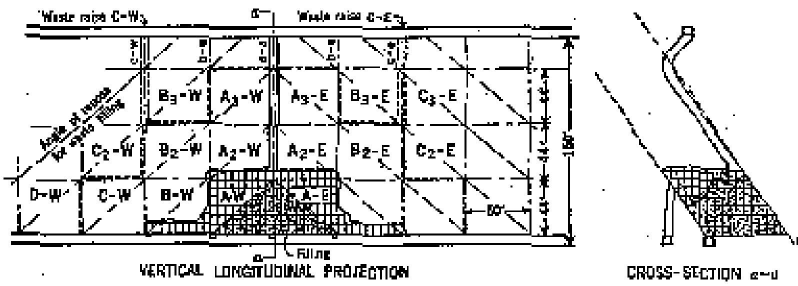

FIGURE 48.—Square-set stoping system, Park Utah mine, Utah.

footwall first. The inclined lines in the figure represent the top of the filling at its angle of repose (about 40°) at various stages of mining. The ore is handled in the stopes by scrapers. Waste filling is run in as the square-set sections are being carried up so that mining and filling are continuous operations. Waste is derived principally from development work and run into the stopes from the hanging-wall drift on the level above. As the stope advances to the position shown by the dotted lines additional waste raises *C–W* and *C–E* are driven. The floor pillar left under the level above is 10 feet thick and is mined in a separate operation by vertical slices across the vein. Before the floor pillar is mined the stope below is filled to within one set of the back between waste raises *C–E* and *C–W*, the waste being leveled off with scrapers when necessary. Horses of waste in the ore body are drilled and blasted into the stope fill. Ore passes and manways are carried up at intervals of 50 feet, or every tenth set.

Square-sets are framed with butting caps. Posts are of 9- by 10-inch timber, caps are 10 by 10 inches, and girts are of 6- by 10-inch timber. A square-set is 5 by 5 feet in plan by 7 feet 4 inches high, center to center, and contains 183 cubic feet.

A square-set stoping crew usually consists of six or eight men, divided equally on two shifts.

The daily routine of operations of such a crew is as follows:

Daily routine of operations in square-set stopes

	Miner	Timberman	Shoveler	Time (minutes)
1	Barring down	Laying floor	Rigging scraper and tail-block.	30
2	Setting-up machine	Getting timber	Cleaning up scattered muck	45
3	Drilling	Standing and blocking timber.	Scraping out ore	180
4	Tearing down and removing machine.	Lagging sets and raising chute.	do	60
5	Getting powder and loading holes.	do	Removing scraper and tail-block.	60
6	Blasting	Blasting	Blasting	

For scraping in square-set stopes the scraper hoist is mounted on a truck and moved parallel with the walls of the stope over a 3-inch plank floor. The scraper is dragged in any line of sets from the foot to the hanging wall. Plank wing slides are built in the sets to direct the broken ore into the chutes or to the scraper runways. Scraping has reduced stoping costs 30 percent compared to handwork.

The direct stoping cost for 1928 was as follows:

Direct stoping costs, Park Utah mine, 1928

	Cost per ton of ore hoisted		Cost per ton of ore hoisted
Labor	$1.437	Timber	$0.423
Supervision	.060	Other supplies	.202
Compressed air, power, drills, and steel	.192	Total	2.426
Explosives	.112		

Basic wage rates were: Miners $5.25, muckers $4.75, nippers $5.25, scraper operators $5.25, and timbermen $5.25.

Square-set stoping was contracted for at $9 to $12 per square-set. Contractors were required to drill, blast, stand, and block timber, place the mining floor, rig up scrapers, and scrape the ore into the chutes. The stope was filled on company account.

Labor performance in the stopes was as follows:

Labor performance in stopes, Park Utah mine, 1928

Occupation:	Man-hours per ton	Tons per man-shift
Breaking	0.74	10.8
Timbering and filling	.83	9.6
Mucking (scraping)	.36	22.2
Total	1.93	4.14

Consumption of timber was 16.46 board feet per ton of ore, including that used in stoping, development, and maintenance.

BAWDWIN MINE, UPPER BURMA

Mining methods at the Bawdwin mine have been described and discussed by Calhoun.[1] The main ore fissure runs through a wide band of rhyolite tuff exposed by erosion of the overlying sediments. The lode has been formed by metasomatic replacement of the rhyolite tuff along the fissures. The Chinaman lode has a well-defined hanging wall but no well-defined footwall. On some levels there is solid sulphide ore 50 feet wide for 1,000 feet or more along the strike, and in places the ore is 140 feet wide. A faulted portion of the lode, known as the "Shan", is much narrower.

The ore is an intimate mixture of galena, sphalerite, and in many places of chalcopyrite also. The Chinese worked the mine from the surface to 50 feet below the 171-foot level for silver alone, and a large remnant of the upper part of the ore body is mixed with old filled stopes and workings. This remnant must be preserved for future mining, although much of it is not of commercial grade today.

The first method of stoping tried was flat-back square-set stoping

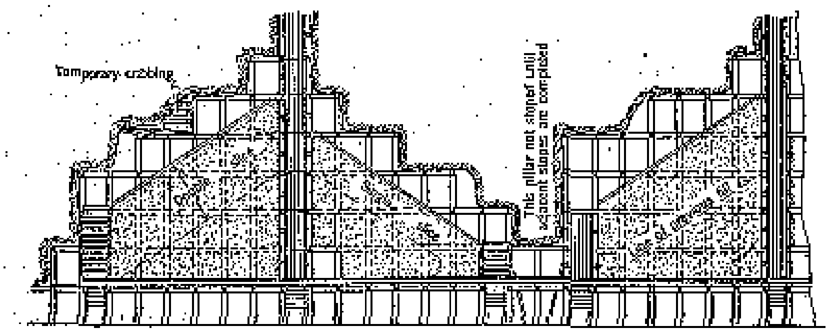

Figure 49.—Square-set stoping, Bawdin mine, Burma.

with long, wide stopes. Later a narrow Gilman-slice rill stope was tried but abandoned. As the upper levels must be kept intact, mill-hole, top-slicing, and caving systems are not applicable. The system finally adopted was a combination square-set rill system with stopes of variable width to suit local conditions. Development drifts are driven outside the ore body in the hard footwall rock, from which crosscuts are driven to and through the ore at 100-foot intervals. From these crosscuts continuous raises are put up from level to level and finally to the surface. When stoping is begun, auxiliary crosscuts are driven halfway between the original crosscuts, which are connected to the main extraction drift with curves of 25-foot radius. The 50-foot blocks on either side, except for a 12-foot pillar directly over the crosscut, comprise the stopes. The rill stopes apex at the main passes and slope down to the extraction crosscuts. (See fig. 49.) Stopes are three or four sets wide. Where the vein is wide a longitudinal section is mined first along the hanging wall; when this is completed second and third sections are mined alongside, retreating toward the footwall. Square-sets are 5½ feet square and 7 feet

[1] Calhoun, A. B., Mining Methods at Bawdwin Mine: Trans. Am. Inst. Min. and Met. Eng. (Issued with Mining and Metallurgy), August 1923, 40 pp.

4 inches high. The only timber available is hardwood, into which nails cannot be driven unless holes are drilled first. Lagging is of 4-inch bamboo 11 feet long. Timber is shipped in by rail at the following rates:

	Rupees	Value in American money[2]
Mine logs, per ton (50 cubic feet)	41	$11.89
Local sawed timber per ton	90	26.10
8-inch by 2-inch by 6-foot lagging, per 100 pieces	74	21.46
8-inch by 3-inch by 7-foot 4-inch lining boards, per 100 pieces	120	34.80

Cost of framing timber

	Hand	Machine
Stope post	R. 0.563—$0.163	R. 0.292—$0.085
Stope cap	R. .563— .163	R. .292— .085
Stope girt	R. .313— .091	R. .292— .085

Manways are on the hanging-wall sides of the chutes; thus access is given at any time to either stope from the manways or chute without blocking or interference. As all ore is dropped to the lowest adit levels any system of continuous raises can be used as an ore pass providing it is not used for passing waste to a stope below. However, it has been found cheaper to run continuous untimbered ore passes from level to level in the footwall, leaving the main passes for waste only. Waste filling is obtained from the surface by quarrying or milholing around the tops of the continuous main passes situated at intervals of 100 feet along the strike of the ore body, so that filling can be passed directly into the stopes without any tramming.

The total average cost per ton of ore is 12.73 rupees ($3.69).

The direct stoping cost was not segregated in Calhoun's paper but is estimated from the detailed costs as nearly as possible on the same basis as the costs given in the Bureau of Mines information circulars cited in this bulletin. Costs originally were given in rupees and are recalculated at 29 cents per rupee.

Estimated direct stoping costs, Bawdwin mine

Ore breaking:	Cost per ton of ore	Filling:	Cost per ton of ore
Miners and shovelers	$0.533	Miners and shovelers	$0.142
Supervision	.166	Supervision	.032
Explosives	.228	Explosives	.008
Timber	.317	Steel	.003
Compressed air and steel	.024	Sundry supplies	.030
Sundry supplies	.045	Lighting	.035
Lighting	.115	General expenses	.018
General expenses	.101	Total filling	[4] .264
Total ore breaking	[3] 1.529	Total direct stoping cost	[5] 1.793

[2] 1 rupee equals 29 cents.

[3] Total ore-breaking cost (including engineering $0.010, repairs and maintenance $0.280, hoisting $0.024, mine ventilation $0.011, and assaying and sampling $0.042) amounted to $1.890.

Part of these items may be chargeable to direct stoping; thus the costs of drill maintenance and repairs are not stated separately and should be added to $1.529, and part of the charge for maintenance and repairs doubtless is directly chargeable to stoping. On the other hand, part of the charges for miners and shovelers, sundry supplies, and general expenses may apply to general mine maintenance.

[4] If repairs and maintenance, $0.150, and hoisting, $0.007, are added, this figure becomes $0.430.

[5] The total direct stoping cost at this mine, figured on the same basis as for the mines discussed in the information circulars, is probably between $1.793 and the total of $1.896 plus $0.430 ($2.326), as reported in Calhoun's paper.

Stopemen were paid on a bonus system based upon the number of sets of timbered stope ground taken out; one set in ordinary ground contains 270 cubic feet or 27 long tons of ore.

Labor performance in stopes was as follows:

Occupation:	*Man-hours per long ton*	*Long tons per man-shift (8 hours)*
Miner	5.16	1.55
Shoveler	1.95	4.10
Total	7.11	1.10

Consumption of explosive was 0.36 pound of 50-percent gelignite per long ton and consumption of timber 0.38 cubic foot of logs, 0.33 cubic foot of sawed timber, 0.30 cubic foot of 8- by 2-inch lagging, and 0.27 cubic foot of 8- by 3-inch lagging, a total of 1.28 cubic feet of timber per long ton of ore.

SQUARE-SET STOPING, USING VERTICAL SECTIONS

The manner of applying vertical slices in loose ground at the Black Rock mine, together with other square-set practices, has been described.

BUNKER HILL & SULLIVAN MINE, KELLOGG, IDAHO

The stoping methods at the Bunker Hill & Sullivan mine have been described by Brown.[6]

According to Brown, the square-set system of mining employed at this mine is suited exactly to the irregular nature of the ore bodies. The ore is too high in grade to risk loss, and the square-set method permits not only very clean mining but also complete extraction at minimum cost. Mining costs probably could be reduced by applying shrinkage methods to some of the ore bodies, but this would result in dilution because of the poorly defined, soft walls and the leaving of important masses of ore in the walls. The stoping system now employed was developed about 1915. The ore in the hanging wall is stoped at the peak of the stope with a nearly vertical working face, as shown in figure 50 and described later. The older flat-back stope resulted in unavoidable heavy falls of ore and waste, danger to the workmen, and certain losses of mineral. The present form of stope with the inverted V cross-section has resulted in a greatly reduced accident rate and virtual elimination of caving of stopes.

The ore bodies occur in a severely faulted zone of quartzite and have been much disturbed by later faults. There are two types of veins: The Bunker Hill type, which consists of wide, irregular masses of galena in a siderite and quartz gangue, and the Jersey fissure, which may range from a few to 40 feet in width, dip 45° to 50°, and traverse hard quartzite. In the Bunker Hill type, which furnishes 80 percent of the ore, there is ordinarily one fairly well defined wall but never two.

The hard quartzite in which the Jersey veins occur allows these ore bodies to be mined by cut-and-fill stoping. Square-set stoping with waste filling is employed for mining the large, irregular masses

[6] Brown, C. E., Mining Methods of the Bunker Hill & Sullivan Mining and Concentrating Co., Kellogg, Idaho: Inf. Circ. 6407, Bureau of Mines, 1931, 9 pp.

of Bunker Hill type ore. The ore bodies range from 300 to 1,000 feet in length and 30 to 125 feet in width. Stoping has been continuous from the outcrop, 3,800 feet above sea level, to the lowest level (1929) 400 feet above sea level and 6,000 feet down the dip.

Although there is seldom a definite hanging wall, the rock over the ore body is considered the hanging wall. This is invariably heavy, and closely filled square-sets must be employed to support it safely. As rapidly as the ore is removed square-sets are erected and then lagged tightly and filled as soon as convenient.

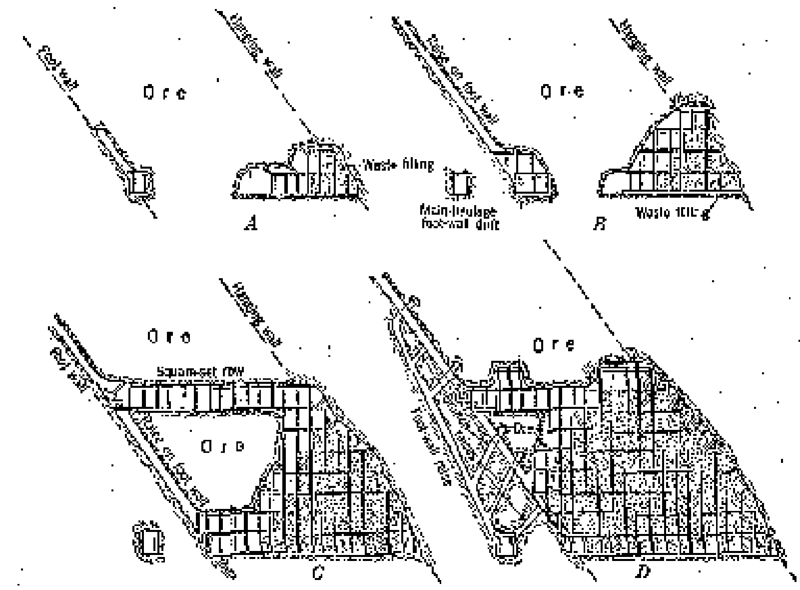

FIGURE 50.—Square-set stoping system, Bunker Hill & Sullivan mine, Idaho: *A*, Stoping begun at hanging wall and raise started on footwall; *B*, stoping advancing toward footwall, raise on footwall completed, and main haulage drift advancing in footwall; *C*, sill floor completely developed and top of stope connected to raise; *D*, first lift of stope nearly completed and footwall and branch raise from haulage drift completed.

Square-set timbers of the post-butting type consist of 12-inch round posts, 8- by 10-inch caps, and 8- by 10-inch ties. The ties are unframed, stronger than the caps, and placed parallel to the walls with the caps at right angles.

Figure 50 shows four stages in the mining of a stope. The vein is cut by the crosscut from the shaft, and a drift is run on the footwall to the extremities of the ore body. When the ore widens so that all of it cannot be included in the drift, crosscuts are driven at intervals to the limits of the ore on the heavy hanging-wall side.

Stoping is begun on the hanging-wall side of the vein, and the ore is silled out on the level. Sills covered with slabs are laid, and square-set timbers are placed. A new floor is begun above the sill floor, the first row of square-sets is extended into the hanging wall to obtain filling for the sill floor, and stoping then progresses toward the footwall. A steeply arched back is maintained at all times. Near the center of the ore shoot a raise is driven on the footwall to the level above and when finished is timbered to provide a waste pass and a timber pass with a manway between.

A second drift is driven 20 or 30 feet in the footwall and extended as stoping progresses to provide the main haulageway from the stope. Footwall branch raises are driven from this haulageway to tap the ore on the footwall side of the stope as shown in figure 50, *D*.

When the waste pass and manway raise have reached the level above a connection is made between it and the hanging-wall side of the stope. This connection is timbered with a row of square-sets (fig. 50, *C*), track is laid, and waste is drawn from the raise and distributed in the stopes with hand-trammed cars. The stope is worked so that the highest part is opposite the raise and slopes downward toward the ends of the stope. Waste for filling is derived from development work, supplemented when necessary by waste from waste raises which are driven into faulted zones where caving can be employed to break the waste.

Direct stoping costs during 1928 were as follows:

Stoping costs, Bunker Hill & Sullivan mine, 1928

	Cost per ton of ore		Cost per ton of ore
Labor	$1.377	Timber	$0.315
Supervision	.068	Other supplies	.164
Compressed air, drills, and steel	.194	Total	2.272
Explosives	.154		

TINTIC STANDARD MINE, TINTIC DISTRICT, UTAH

The major ore bodies of the Tintic Standard mine are in limestone on or near its contact with faulted faces of quartzite—a firm, hard, massive rock.[7] Faulting has been extensive along these faces. Postmineral faulting has brecciated the ore into fine, sandy material, intermixed with masses of heavy lead ore. The hanging-wall rocks are shales and limestones. The shales have been altered to a soft, clayey material, and the limestone is altered and brecciated. Individual ore bodies adjacent to the quartzite fault faces dip from 45° to vertical, whereas those distant from the quartzite are tabular replacements of flat-lying limestone beds which range from 6 to 200 feet in height.

Two principal classes of ore are mined—a high-grade lead-silver ore and a siliceous silver ore. Most of the ore bodies have indefinite boundaries, and the grade diminishes gradually around the margins. Rapid changes from lead to siliceous silver ore make stope control essential.

Levels are established at vertical intervals of 100 feet. In preparation for stoping a vertical square-set raise is driven from the footwall at the level from which the stope is to be started to the level above. Stoping is begun from the raise, which is used as a waste pass for filling the stope. Every stope (not stope section) must have at least one raise.

The first stope section is two or three sets wide and extends from the raise to the hanging wall. It is mined floor by floor as high

[7] Wade, James W., Mining Methods and Costs at Tintic Standard Mine, Tintic District, Utah: Inf. Circ. 6360, Bureau of Mines, 1930, 21 pp.

as safety permits and is then filled, leaving manways on the outside of the section at convenient places. These manways afford points of attack when the next section is begun and provide means of access to any floor of the stope. The first section is completed to the hanging wall, or mined eight floors high, and filled before the second section, two sets wide, is started from the sill on one side. While the second section is being filled a third is begun on the other side of the first section. By alternating from one side to the other and filling one side while the other is being stoped a regular output is maintained from the stope.

After a number of sections have been stoped a filled stope with a vertical face of ore exists on the footwall end. This ore is mined in the same fashion, starting a two-set section from the raise and advancing up the footwall. A footwall drift from which raises are driven to the eighth floor affords an outlet for the ore above this floor.

Virtually all stopes produce three types of ore in addition to waste, and the three types are mined and shipped separately.

Due to irregularity in ground conditions few stope sections can be mined floor by floor and filled. If the ground will not stand while a complete floor is mined, the practice is varied by using vertical instead of horizontal cuts, working from the top down by floors or from the top down by vertical cuts, or using combinations of these methods.

Square-set timbers consist of 9- by 10-inch posts, 10- by 10-inch caps, 6- by 10-inch braces, and 6- by 10-inch sills. Caps are placed parallel to the footwall with their ends against the gob of the adjoining section so that settling of the fill will not disconnect the timbers at the tenons. Before ground is removed to make room for a set along the side of a filled stope a stringer is placed against the gob posts and braced diagonally.

Stoping costs during November 1929 were as follows:

Direct stoping costs, Tintic Standard mine, November 1929

[Cost per ton of ore hoisted]

	Breaking and timbering	Filling	Total
Labor	2.109	0.291	2.400
Supervision	.223	.031	.254
Compressed air, drills, and steel	.521		.521
Explosives	.099		.099
Timber	.691		.691
Other supplies	.049	.010	.059
Total	3.692	.332	4.024

Maintenance cost, reported under mining in Information Circular 6360, was $0.949 additional, but probably only a small part is chargeable against stoping.

The wage scale was as follows: Special timbermen, $6; special timbermen helpers, $5.75; timbermen, $5.50; timbermen helpers, $5.25; miners, $5.50; shovelers, $5; and tool nippers, $5.50.

Labor performance in stopes was as follows:

Occupation:	Man-hours per ton	Tons per man-shift
Breaking	0.733	10.71
Timbering (stoping)	.685	11.68
Timbering (filling)	.191	41.88
Shoveling (stoping)	.827	9.67
Shoveling (filling)	.294	27.20
General (stoping)	.449	18.82
General (filling)	.062	128.24
Total	3.241	2.47

The consumption of explosive in stopes only was 0.4184 pound per ton of ore, of which about half was 25-percent and half 40-percent gelatin dynamite. The consumption of timber in stopes was 19.743 board feet per ton of ore, and 4.286 board feet per ton additional were used for mine maintenance.

UNITED VERDE EXTENSION MINE, JEROME, ARIZ.

D'Arcy[5] has described the conditions and mining practice at the United Verde Extension mine. The ore bodies occur on the hanging-wall side of the Verde fault, which dips about 59° northeast and has a vertical downthrow of approximately 1,600 feet. A large lens of high-grade copper ore extends from a point between the 1,200- and 1,300-foot levels to the 1,500-foot level, below which it gradually narrows. On the 1,400-foot level this lens is oval, with a maximum length of 500 feet and a maximum width of 300 feet; virtually all of this lens is clean, high-grade ore. The enclosing rocks are quartz porphyry, diorite, and greenstone, overlain by conglomerate and lava.

Because of the heavy, massive nature of the main ore body, the richness of the ore, and the necessity for mining it so that no blocks of sulphide ore will move and generate heat only square-set stoping by stopes tightly filled with waste has been tried in the main sulphide lens. With this method complete extraction of the ore has been obtained with virtually no dilution, and careful prospecting of the walls has resulted in the discovery of many small, rich lenses of ore that otherwise would have been missed.

Levels are established at vertical intervals of 100 feet in the main ore body. A typical stope is shown in figure 51. Stope sections usually are 3 sets wide in fairly solid ore and 2 sets wide in heavier ground. If the ore is broken very badly it sometimes is removed in slices only one set wide. Slices are taken 100 feet high, and the length of a slice usually ranges from 10 to 20 sets. Ore chutes are placed in about every fourth set, and alternate chutes have manways beside them. If no weight develops on the timbers after one floor is removed another floor is mined, and sometimes several more, before filling with waste. This reduces the cost of mining, as most of the ore rolls to the chutes and a large part of the filling can be run into place by gravity.

After one section has been finished it is filled entirely, except for the chutes and manways which will be required for entrance to and

[5] D'Arcy, Richard L., Mining Practice and Methods at the United Verde Extension Mining Company, Jerome, Ariz.: Inf. Circ. 6250, Bureau of Mines, 1930, 11 pp.

mining of the next section. The chutes also serve as fillholes for running waste into the section. By having a fillhole in about every sixth set and mining several floors before filling little shoveling of waste fill is required.

All stope timber is standardized. In the heavier sulphide stopes nearly all timber is 10- by 10-inch Oregon fir, but in the lighter ground 8- by 8-inch Oregon fir is used. In the heaviest sill-floor gangways all timber is 12 by 12 inches in section.

A system has been employed successfully for mining badly broken pillars by stoping up through the center with small square-set cuts, then tying across the top with timber stringers and slicing the sides downward.

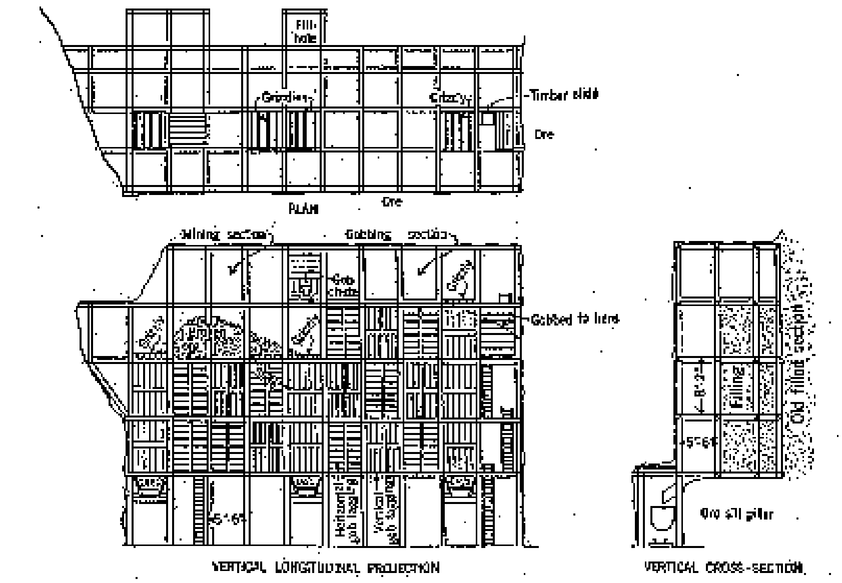

FIGURE 51.—Square-set stoping system, United Verde Extension mine, Arizona.

Stoping costs for 1928 are given as follows:

Stoping costs, United Verde Extension mine, 1928

	Cost per ton of ore
Extraction	$1.710
Rock drills	.152
Compressed air	.092
Waste pit	.005
Total	1.959

Efficiency data during 1928 were as follows:

Efficiency data, United Verde Extension mine, 1928

Ore mined per stope-shift ... tons	4.84
Man-hours per ton in stopes	1.65
Mine timbers and handling cost per 1,000 board feet	$36.24
Mine timbers and handling cost per ton of ore	$0.67
Mine timbers ... board feet per ton	18.53
Powder in stopes ... pound per ton	.39
Fuse in stopes ... feet per ton	1.93

EAGLE MINE, GILMAN, COLO.

The use of square-set aisles along each side of cut-and-fill stopes and of square-set stoping for mining the weak ore at the top of the ore bodies, as described by Borcherdt, has been referred to under Cut-and-Fill Stoping.

The blanket ore body directly under the capping and pyritic chimneys are mined by square-setting. In the flat deposit stopes are laid out in checkerboard fashion. Individual stopes range from 3 by 5 to 4 by 6 sets in floor area and from 3 to 6 sets in height. They are filled from waste raises in the back.

The pyritic ore of the chimney ore bodies is too heavy and weak to stand unsupported over more than 10 or 12 feet, and usually it is cemented so poorly that timber sets must be placed as soon as room is available for them. In some places the pyrite is in the form of unconsolidated sand, and underhand mining is the only method practicable. In the best mineralized areas the ore is 100 to 135 feet thick. Stopes are carried 2 sets wide and from 5 to 10 sets long and from the base to the top of the ore. The chimneys are mined by vertical slices. The following is taken from Borcherdt's discussion:

A two-set raise through the ore is the initial step in stope preparation. This raise connects with a filling transfer level above and with an extraction crosscut below. One compartment is used for an ore chute and waste pass and the other for a manway, pipe lines, and materials. Overhand square-set stopes are started at the base of the ore or on the second set above the drawing level, where the base of the ore drops below this level. If the ore back is sufficiently strong, as many as five tiers of sets will be placed before filling is introduced, and much of the ore will be broken onto slides to direct it to the chute. In weaker ground no more than two tiers are kept open. The stope is filled to within one set of the back, and a one-set cut is then made, after which the stope is again filled to the base of the upper sets.

As the stopes are small, filling and breaking are not carried on simultaneously but in rotation. Stopes are lagged both inside and outside of the sets, the inside lagging retaining the fill. Broken ore is used to fill the space between the outside lagging and the standing wall, thus preventing spalling of the ore wall and loosening of the blocking. The outside lagging and this broken ore pack are recovered in mining the adjacent stope. Much of the ore body is so loose that it must be worked down from the top, underhand square-setting being employed in small rectangular blocks. Normally, several tiers of sets are placed before waste filling is introduced. Occasionally, however, the stope is filled after each slice. As underhand or overhand stopes are completed, the raise is extended, to the depth of two sets, into the ore that is to be mined next. This extension or "wing" forms the extraction raise for the new stope adjacent to that worked out. The original two-set raise of the completed stope is filled with waste and abandoned.

Waste for filling iron-ore stopes comes from development and general caving stations. Caving from the back above stopes is not practiced in the chimney areas. The general caving stations are placed in the porphyry and shale of the capping. Where the capping is good caving ground, a two-compartment vertical raise is driven 25 feet above the drawing level. A 50° inclined raise is then driven from the vertical raise into the capping and branched, and short drifts are turned off. The ground is core-drilled with jointed-rod drills, and the holes are heavily loaded and blasted simultaneously. In good caving ground this is sufficient to start the caving. Where the capping is firmer, the back has to be undercut for a width of 60 to 80 feet by shrinkage stoping before the desired self-caving will start.

Data were given on nine cut-and-fill stopes in the section under Cut-and-Fill Stoping. Similar figures follow on 11 square-set stopes in zinc ore.

Data on 11 square-set stopes, Eagle mine

Production per machineman shift	tons	25.7
Production per timberman shift	do	13.5
Production per mucker shift	do	13.5
Production per fill-labor shift	do	35.1
Production per man-shift, all stope labor	do	4.64
Timber per ton of ore	board feet	13.9
Powder per ton of ore	pounds	.67

Comparison of these figures with those for cut-and-fill stopes shows that production per man-shift for all classes of stope labor was less in square-set stopes than in cut-and-fill stopes. Production per man-shift, all stope labor, was 6.4 tons in cut-and-fill stopes compared with 4.64 tons in square-set stopes. Timber consumption per ton in cut-and-fill stopes was a trifle more than half that in square-set stopes.

RECOVERY OF PILLARS BY SQUARE-SETTING

Pillars left between filled stopes usually have taken weight before they can be mined. In some instances they are completely crushed. Most pillars between finished stopes are mined by taking vertical slices; if the pillars are badly crushed vertical slices comprising only a few sets each are mined downward, as at the United Verde Extension. Crown or level pillars may be mined either by inclined or vertical sections. At the Homestake mine 60-foot shrinkage stopes are taken across wide ore bodies, leaving 40-foot pillars between each pair of stopes; 25-foot crown pillars are also left at the top of the shrinkage stopes immediately below the level above. These pillars are mined by square-setting. The method is described in the section on Shrinkage Stoping.

UNITED VERDE MINE, JEROME, ARIZ.

As stated under cut-and-fill stoping 27 percent of the ore stoped in the United Verde mine during 1929 came from square-set stopes.

Square-set stoping is employed for extracting the vertical pillars between completed cut-and-fill stopes and for mining crown pillars at the top of cut-and-fill stopes; the method is employed also in some sections in the main stopes where the ground is heavy. In the vertical pillars, which are 30 or 40 feet wide, the square-set sections are necessarily small, rarely exceeding 35 sets in area, whereas in the main stopes there may be as many as 250 sets on a floor. Sets are 5 feet 6 inches square by 7 feet 2 inches high. The usual square-set-and-fill practice is followed in horizontal square-setting. No more than two floors are opened at a time, and filling is kept as close as possible behind the shovelers. An inclined square-set system is used principally for mining level pillars. An inclined stope is begun from a raise in the center, usually by taking out four rows of sets on the second floor and two on the third floor before any filling is introduced. After the incline is established stoping is begun on each new cut at the bottom and progresses upward to the crest of the stope. The main advantages of the inclined system are that it increases the rate of output from a given area at a lower cost per ton and that there is less weight on the brow than with horizontal stoping.

Labor performed in square-set stopes is given by Quale as follows:

	Man-hours per ton of ore stoped	*Tons stoped per man-shift*
Breaking	0.383	20.9
Shoveling	.372	9.2
Timbering and filling	.606	13.2
Total	1.361	[9] 4.3

SUMMARY OF SQUARE-SET STOPING

1. *Applicability.*—Square-set stoping is applicable to the mining of ore deposits of all sizes, shapes, dips, and degrees of irregularity, where the ore and walls are too weak to stand unsupported even over small spans for more than a very short time and where it is necessary to prevent caving and subsidence of the overlying and surrounding ground. It is applied best to the mining of high-grade ore where complete extraction of the ore body is desirable. The method seldom is used today unless the ground is so weak that the stopes must be filled as well as timbered, either along with the stoping or shortly after relatively small slices of ground are removed.

2. *Flexibility.*—Square-set stoping is very flexible and permits following stringers and offshoots from the main ore body and mining irregular lobes and patches of ore with minimum dilution by waste and loss of ore. Bands or irregular inclusions of waste can be left in place, if large enough, or broken and the waste sorted out from the ore and gobbed in the stope. The method can be adapted to sudden changes in dip of the deposit and to following sinuosities or abrupt offsets along the strike and dip.

Square-set stoping usually is done overhand (from the bottom upward) but also may be employed for mining from the level downward (underhand square-setting). It can be, and frequently is, employed as an auxiliary method with other methods of stoping; it may be a regular part of the stoping system in combination with other methods or be introduced here and there to meet local conditions requiring its use in parts of stopes mined by other methods. As in cut-and-fill stoping the rate of production in a given stope is restricted, particularly in small ore bodies, and each face is limited in the amount of ore it can contribute per day due to the necessity of breaking only a limited amount of ground at a time and timbering this before the next round is broken. The ore is removed as fast as it is broken; therefore a large amount of capital need not be tied up in broken ore lying in the stopes.

3. *Recovery.*—The method commends itself because a high percentage of the total ore in the deposit can be recovered, especially where the ore is high in grade and the cost of recovery is secondary to complete extraction—in other words, where the value of the ore far exceeds the cost of recovering it. By square-setting and close filling, high-grade sections of an ore body below or adjacent to lower-grade ore often can be mined, as at the Bawdwin mine, without disturbing and increasing the difficulty of extracting the lower-

[9] This compares with 8.97 tons per man-shift in cut-and-fill stoping. Consumption of explosives was 0.60 pound per ton of ore and of timber 9.91 board feet per ton.

grade ore at some future time. Such lower-grade material, which may not be mined profitably during the earlier life of the property, thus may be preserved as a reserve for the future, when improvements in mining, ore dressing, or metallurgical technology or more favorable market conditions may make its extraction profitable.

4. *Development.*—Development in preparation for square-set stoping—that is, stope development—is comparatively simple but usually requires at least one through-raise to the level above for each stope or stope section to introduce waste filling, to improve ventilation in the stope, and to afford a means for lowering timbers and supplies to the stopes instead of hoisting them up from below. Where ore bodies are large and the ground is strong enough to permit large stopes to be mined the amount of raising required per ton of ore may be small. In very weak ground, where it is necessary to stope small sections at a time and each section requires a through-raise in advance of stoping, the ratio of raise footage to tons mined is high. No figures are available on the footage of stope development, as distinguished from and segregated from total development per ton stoped in square-set mining. However, simple calculations show that the ratio of stope-development footage to tons mined may vary widely. Thus, in a vein 25 feet wide, in heavy ground requiring mining by a series of transverse stopes 20 feet wide with a waste-fill raise for each 20-foot stope, about 30 to 35 tons of ore would be mined per foot of raise development. If one raise would do for each pair of stopes the quantity mined would be 60 or 70 tons. On the other hand, where a single raise serves a stope 100 feet long by 50 feet wide about 400 tons of ore would be mined per foot of raise development. With respect to ratio of stope development to ore mined, square-set stoping compares favorably with other stoping methods.

5. *Cost of square-set stoping.*—Stoping costs at a number of mines employing square-set stoping are shown in table 14. The mines listed are all in ore bodies of good stoping width, and most of them are in what would ordinarily be classed as wide ore bodies. Stoping costs in table 14 should be compared with those in tables 7 and 10 for shrinkage stoping and cut-and-fill stoping at mines working in deposits of similar width. In table 14 it should be noted that (1) the Argonaut mine is in heavier ground and narrower ore than the other mines; (2) considerable sorting (20 percent) is required in the stopes at the Pecos mine which increases the cost; (3) at the Tintic Standard mine virtually all stopes produce three different types of ore besides waste, necessitating selective breaking and sorting which add appreciably to the stoping cost; and heavy ground and settling fill add to the timbering cost; (4) at the Silver King Coalition mine 15 to 20 percent of the ground broken is sorted out underground and returned to the stopes as waste for filling; and (5) labor at the Bawdwin mine is cheap coolie labor with European supervision, and other conditions are quite different from those at American mines.

Table 15 summarizes data on man-hours per ton in stoping operations, and tables 16 and 17 give, respectively, explosives and timber consumption per ton mined at several mines employing square-set stoping.

TABLE 14.—*Stoping costs at mines employing square-set stoping*

Mine	Year	Variation of square-set method	Width of ore, feet	Direct stoping costs per ton of ore produced						
				Labor	Supervision	Compressed air, drills, and steel	Explosives	Timber	Other supplies	Total
Argonaut	September 1929	Horizontal floors	20, average	$1.746	$0.052	$0.101	$0.148	$0.336	$0.057	$2.436
United Verde Extension	1929	Vertical slices 2 or 3 sets wide and 10 to 20 sets long.	Wide, maximum 300			.214 [1]1.745				1.959
Bunker Hill & Sullivan	1928	Inverted V-shaped back; nearly vertical stope faces.	30 to 125	1.377	.058	.194	.154	.316	.064	2.272
Page	1929	Horizontal breast stopes	Up to 20	1.43	.16	.16	.11	.20	.04	2.16
Pecos	1929	Horizontal in sections 30 feet long	Narrow to more than 30 feet.	2.16		(2)	.15	.40	.12	2.83
Park Utah	1929	Fill sections	3 to 80	1.427	.060	.192	.112	.423	.202	2.420
Tintic Standard	November 1929	Vertical sections 2 sets wide and up to 20 sets long.	Up to 300	2.400	.254	.321	.009	.001	.059	4.034
Silver King Coalition	1929	Stringer-set-and-fill and regular framed square-sets.	Wide	2.454	.130	.472	.178	.249	.024	3.503
Bawdwin	Before 1924	Fill sections 50 feet long by 3 or 4 sets wide.	Up to 140, average 50	.792	.199	[3].097	.234	.317	.225	1.793

[1] All other stoping costs.
[2] Not segregated.
[3] Air and steel only.

TABLE 15.—*Man-hours per ton stoping at mines employing square-set stoping*

Mine	Year	Width of ore, feet	Variation of square-set method	Stoping man-hours per ton of ore mined: Breaking	Shoveling	Filling	Timbering	Total	Tons per man-shift, all stope labor
Argonaut	September 1929	Average, 20	Horizontal					[1] 2.051	
					0.475	0.347		.822	
United Verde Extension	1929	Wide, maximum 300	Mined up in slices 2 or 3 sets wide and 10 to 20 sets long.					2.870	2.83
								6.603	4.84
United Verde [2]	1930 (?)	Wide	Inclined stepped faces	0.883	.872	0.600		1.801	4.50
Park Utah	1928	3 to 30	Rill sections	.74	.36	.83		1.93	4.14
Tintic Standard	1920	Up to 300	Vertical sections 3 sets wide by up to 20 sets long.	.733	.527	.350	.870	2.702	
								[3] 3.450	
Silver King Coalition	1928	Wide	Stringer-set-and-fill and regular framed square-sets.					8.241	2.47
				.934	[4] 2.454		.083	3.771	2.12
Eagle [2]		do	Horizontal	.313	.382	.239	.802	1.723	4.64
Bawdwin	Before 1923	Up to 140, average 50	Rill sections 50 feet long and 3 or 4 sets wide	[5] 5.16	1.95			7.11	1.10

[1] Breaking and timbering.
[2] Square-set stopes only.
[3] Stoping, general.
[4] Shoveling and sorting.
[5] Timbering and breaking (all miners in stopes).

TABLE 16.—*Consumption of explosives in square-set stopes*

Mine	Kind and strength of explosive used	Explosive used per ton ore broken, pounds
Argonaut	Gelatin, 30 percent	[1] 1.010
United Verde Extension	(?)	.390
Pecos		[2] .730 [2] .850
United Verde	Gelatin, 50 percent	[3] .800
Tintic Standard	Gelatin; one-half 25 percent, one-half 40 percent	.418
Eagle	Gelatin, 45 percent and ammonia dynamite, 40 percent	[3] .870
Bawdwin	Gelignite, 60 percent	.300

[1] Includes some powder used in development.
[2] Average of square-set and cut-and-fill stopes; 1929 and 1928, respectively.
[3] Square-set stopes only.

TABLE 17.—*Consumption of timber in square-set stopes*

Mine	Consumption of timber per ton of ore mined		Mine	Consumption of timber per ton of ore mined	
	Framed timber, board feet	Round timber and lagging, linear feet		Framed timber, board feet	Round timber and lagging, linear feet
Argonaut	7.06	1.08	Tintic Standard	10.74	
United Verde Extension	18.68		Black Rock	12.00	2.6
United Verde	9.01		Eagle [2]	13.00	
Park Utah [1]	10.40		Bawdwin	15.30	

[1] Includes timber for stoping, development, and maintenance.
[2] Square-set stopes only.

ADVANTAGES OF SQUARE-SET STOPING

The advantages of square-set stoping are summarized as follows:

1. Weak ore with weak walls can be mined without resultant caving and subsidence of the overlying and surrounding rocks.

2. The method is flexible and can be employed on all dips in large or small ore bodies. Irregularities in the ore bodies can be followed and mined, and waste inclusions can be left in place or sorted out and used for filling in the stopes.

3. High percentage of extraction and clean ore may be obtained.

DISADVANTAGES OF SQUARE-SET STOPING

The disadvantages of square-set stoping are summarized as follows:

1. Square-set stoping is a high-cost method and therefore is not applicable to low-grade ore which will not stand a high stoping cost.

2. It is a slow method of stoping, and production cannot be forced to any extent. Slow working results in a long life for individual stopes which gives time for development of pressure, air slacking, and attendant difficulties.

3. Handling large quantities of timber and waste filling complicates and increases the costs of underground transportation.

4. The accident rate for square-set stoping is higher than for any other method of stoping. This is doubtless due partly to the fact that it is employed in bad ground but also to the necessity of handling heavy timbers and using sharp tools.

5. The large amount of timber presents a distinct fire hazard.

BLOCK CAVING

In the block-caving method of mining natural forces are utilized to the highest degree in breaking the ore, filling the mine openings, and transferring the broken ore to the haulage levels. The sublevel-caving and top-slicing methods utilize these forces to a smaller degree and are intermediate between the supported-stope mining methods and the block-caving method. For this reason it might be logical to discuss top slicing and sublevel caving first and conclude with a discussion of block caving. On the other hand, block caving embodies most perfectly and completely the principles of mining by caving and is the method that comes to mind first when caving methods are mentioned.

Block caving is applicable to the mining of deposits of considerable vertical height and horizontal area which will readily cave when the support is removed by undercutting and which after caving will break fine enough to pass through the extraction raises as the caved mass is drawn downward. This condition occurs when the ore body is traversed by closely spaced seams, fracture and joint planes, or invisible planes of weakness, which trend in several directions or which occur in two or more systems of such planes. However, if the ore packs considerably after breaking up, results may be unsatisfactory. Block caving is used principally in mining low-grade ores, where the inevitable loss of some of the ore and some dilution with capping and wall rock are not serious objections. The adaptation of block caving to the mining of large, low-grade, so-called "porphyry-copper" deposits marked a distinct departure from previous conceptions of mining by small-scale selective methods and toward the wholesale extraction of mineral-bearing rock by nonselective methods.

Although block caving has been perfected at the porphyry-copper mines it had previously been employed elsewhere. The authors are unable to state where it was used first, but it was employed in Michigan iron mines several years before it was introduced at western copper mines. The caving system was used at an early date in the Kimberley (South Africa) mines by Charles Henroten.[20]

MICHIGAN IRON MINES

Figure 52 shows the details of one of the earliest block-caving systems used in Michigan. The deposit was divided into blocks 200 feet long. Drifts *A–A* were run 20 to 40 feet back in the hanging wall and parallel thereto on levels about 100 feet apart vertically. After all the ore above the upper level had been worked out and

[20] Jennings, Sydney J., Discussion of Underground Mining Systems: Trans. Am. Inst. Min. and Met. Eng., vol. 52, 1916, p. 422.

the surface caved thereto, crossdrifts *B–B* were driven to the footwall and raises *D–D* run nearly to the level above. The tops of these raises were connected by drift *E*, and the entire slice bounded by the dotted lines was stoped out to detach the block on each end. During these operations drifts *C–C* were run to the footwall, leaving pillars *N–N* between them. From the footwall these pillars gradually were slabbed back toward the main drift and finally drilled and blasted simultaneously. After 3 to 12 months the ore block caved.

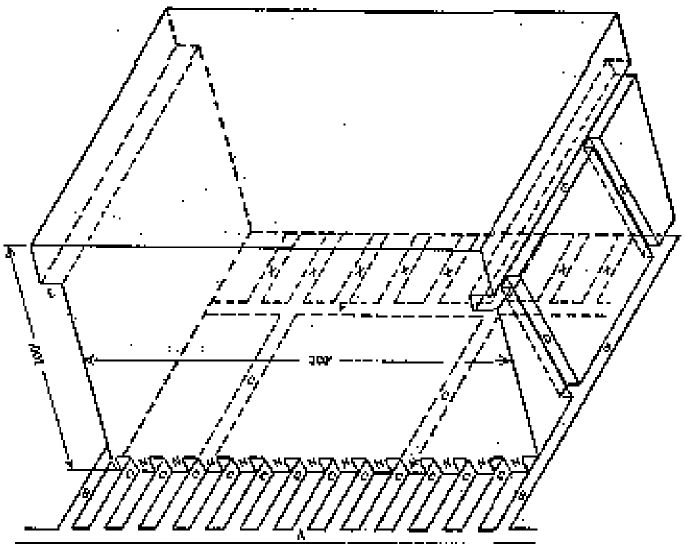

FIGURE 52.—Early block-caving system in Michigan iron mine.

Two of the crossdrifts *C–C* were then reopened by forepoling and connected by a drift, *F*, parallel to and near the footwall. From the latter, crossdrifts *X–X* were opened to the footwall on 20-foot centers. The caved ore slid down the footwall into these drifts and was shoveled into cars. When all the ore that would cave had been cleaned out another drift was opened parallel to the footwall and farther back toward the hanging wall, and the operation was repeated. This method was applicable to hard ore only.

The caving system at the Pewabic mine on the Menominee range has been described by Wilson [11] and Myers.[12]

The following is abstracted from Myers' discussion: The block-caving system was used in a block of ore about 250 feet wide that stands nearly vertical between slate walls which are uniform in strike and dip. A horizontal sandstone capping 100 feet thick overlies the ore formation and forms a strong arch over it.

Development of a block for caving consists of driving footwall and hanging-wall drifts the length of the block, which is about 250 feet square. Crosscuts and drifts are driven north from the hang-

[11] Wilson, J. B., Caving Systems of Mining in America: Eng. and Min. Jour., vol. 94, 1912, p. 245.

[12] Myers, A. J., The Block-Caving System Used at the Pewabic Mine: Proc. Lake Superior Min. Inst., vol. 21, 1916–17, pp. 25–28.

ing-wall drift and east from the main crosscuts. Stopes are then begun above the footwall drift and the crosscut at the west end of the block, and a back pillar 6 feet thick is left above the drift and crosscut to hold the broken ore on which the men work while they are raising the stopes. Raises are put through the back pillars on 30-foot centers for ore passes and manways. The stopes are raised to within 20 feet of the level above, and the pillars above the drift and crosscut are blasted out. The broken ore then is cleaned from the stopes, leaving a block of ore 250 feet square and 100 to 125 feet high cut off on all but the hanging-wall side and standing on solid pillars 22 feet square. Meanwhile, a main haulage drift has been driven in the hanging-wall slate about 40 feet from the ore. (See fig. 53.)

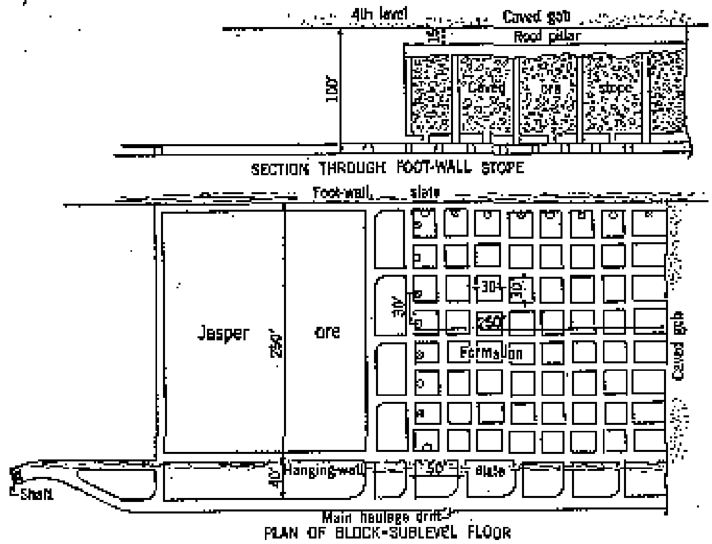

FIGURE 53.—Block caving, Pewabic mine, Michigan.

The pillars are then weakened by drilling and blasting off the corners; some of the pillars will crush when 6 feet in diameter, and others must be blasted out entirely. This weakening of the pillars is begun on the side of the block farthest from the main haulage drift and proceeds until the width of the formation is covered. The block when down has dropped about 8 feet and has broken finely enough to be shoveled into cars.

Meanwhile crosscuts have been driven from the main haulageway on 50-foot centers to the hanging wall of the block, and timbered drifts then are forepoled through the broken ore to the footwall side where drifts are driven off at right angles. Drawing of the ore is begun here and continued at one point until the sandstone or surface capping moving with the ore makes it too low in iron content. Drawing then proceeds backward toward the haulage drift until all the ore is extracted. Only enough crosscuts are driven to supply the required output, thus only a small portion of the caved block is supported on timbers at one time.

The system at the Tobin mine differed in that the stope floor was begun above the haulage level. The following is abstracted from Roberts'[19] discussion.

The levels at the Tobin mine are 125 feet apart vertically, and the main haulage drift follows the hanging wall closely.

In the block to be caved parallel crosscuts 24 feet from center to center are driven from the main drift and as nearly as possible at right angles thereto. These are driven to the footwall, where they are connected by a small drift for ventilation. Chute raises are put up along the crosscuts alternately on the right and left sides at intervals of 15 feet to a sublevel 25 feet above. (See fig. 54, *A*.) On the sublevel a drift is run parallel to and about 15 feet from the hanging wall the entire length of the block. From this drift crosscuts are driven to the footwall directly above the crosscuts on the haulage level. Opposite each crosscut on the sublevel a raise inclined 45° is put up from the subdrift to the hanging wall so as

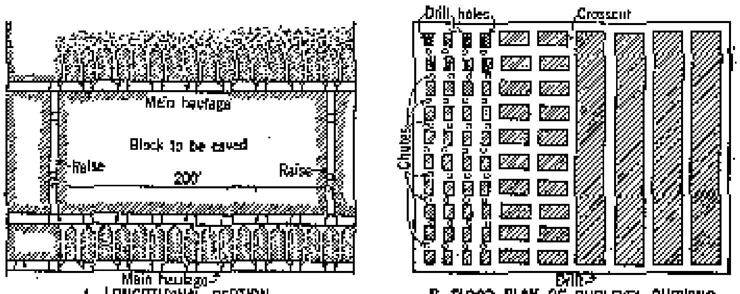

FIGURE 54.—Block-caving system, Tobin mine, Michigan.

to leave additional back above the main haulage drift. The crosscuts on the sublevel are connected every 15 feet by drifts over the lines of chute raises, leaving pillars about 10 by 16 feet. (See fig. 54, *B*.) These pillars are slabbed and then drilled and blasted, beginning at the pillars farthest from the manway. Holes also are blasted around the tops of the raises to funnel them out.

The ground must be weakened at the ends of the block so that it will cave square with the pillar. Raises are put up from the end crosscuts at varying intervals and connected by crosscuts 25 and 50 feet above the sublevel. After all necessary raising, drifting, and crosscutting have been completed the holes in the sublevel pillars are blasted simultaneously, undercutting the entire block, which settles down and breaks so that it can be drawn through the funneled raises with only occasional blasting of masses that lodge in the chutes. The caved ore is drawn uniformly throughout so that it will settle down evenly and prevent excessive dilution of the ore with caved waste rock from the level above.

Roberts stated in his article that "for this method of handling the block-caving system, as far as known to us, we are the originators."

[19] Roberts, Fred C., Block Caving and Substope System at the Tobin Mine, Crystal Falls, Mich.: Proc. Lake Superior Mining Inst., vol. 16, 1911, pp. 218–220.

WESTERN COPPER MINES

The first application of block caving to the mining of copper ore was in 1906 at the Ohio Copper mine at Bingham, Utah. At this time a chute-caving method was being used at the Utah Copper mine, but it was considered that control of the movement of the broken ore was not positive enough for the system to be applicable to the Ohio Copper ore body.

A block-caving method using branched raises adapted from the practice at the Ohio Copper mine was installed at the Inspiration Copper Co. in 1911. Contemporaneous with the work being done at the Inspiration mine operations were begun at the Ray mine, now belonging to the Nevada Consolidated Copper Co. The original method at Ray consisted of mining shrinkage stopes to the overburden above the ore, leaving alternate pillars which were cut off at the bottom and caved after the shrinkage stopes were completed. Both the shrinkage and pillar stopes were drawn at the same time. The present practice is an improvement on this first method.

Later the Miami Copper Co. adopted a caving method similar to that in use at the adjoining Inspiration property. The Nevada Consolidated introduced a caving method for mining the Ruth ore body at Ruth, Nev., in 1915. Beginning in 1922 the Morenci branch of the Phelps Dodge Corporation developed a caving system for mining a large body of low-grade ore at Morenci. The Copper Queen branch of the Phelps Dodge Corporation began mining low-grade ore bodies by a caving system in 1925. The Andes Copper Co. began mining by a caving system in 1926 and the Consolidated Copper mines in 1928.

A block-caving system has been gradually evolved from shrinkage and pillar caving at the Braden mine, Sewell, Chile; the cost of stoping by this change has been reduced from 43.4 to 20.5 cents per ton. In 1932, 85 percent of the production of the mine was from block caving. The method used at Braden more nearly corresponds to block caving in the iron mines than in the western copper mines.

The block-caving method is used for mining molybdenum ore at the Climax mine, Climax, Colo.; this has been evolved from a shrinkage and pillar caving method described under the section on Shrinkage Stoping; caving of the initial block in a new section is induced by blasting.

Block caving has been successfully adapted to mining blocky limestone at the Crestmore mine of the Riverside Cement Co. The method was evolved from the Inspiration and Ray systems. The draw points are arranged at Crestmore to permit the coarse material coming to the grizzlies to be readily blockholed. Until this development it was not considered practical to mine a material that broke so coarsely by a caving method.

Although the general principles are the same the methods used at the important mines have been evolved to fit particular conditions. Considerable variation exists in the manner of accomplishing the desired results.

The undercut block-caving method is characterized by an intensive production of ore from a relatively small area which permits standardization of nearly all operations and efficient working of the mine. The principal improvement in the method, which has had the greatest influence in reducing mining costs, has been the progressive increase in the height of the blocks from the initial 30 feet at the Ohio Copper to over 300 feet at some of the modern mines.

Operations in modern copper mines using a block-caving method of mining are conducted on three levels—the level on which the ore is undercut, the grizzly level on which the ore is drawn from the stopes, and the haulage level on which the ore is transported to the shaft.

The ore from the grizzlies is dropped through a regular system of raises, usually but not always branched, to the haulage level.

Caving is induced by undercutting supplemented by either cutting off the block entirely from the surrounding ground or weakening it along its boundaries (usually the latter) to assist the caving action and confine it to the block mined.

The ore is drawn through openings uniformly spaced below the block. For perfect drawing the draw points should be spaced closely and regularly, but a compromise must be made between close spacing, which increases ore recovery and reduces dilution, and wider spacing, with resulting economy in preparatory costs and operating repairs. The character of the ground is the controlling factor in drawing practice.

In the original Inspiration ore body and at the Miami and Ruth mines the ore breaks finely and tends to pack and, when drawn, pipes vertically, with little spread beyond the draw points. This condition makes it necessary to space the draw points as close together as economy and the necessity for maintaining supporting pillars will permit. At Morenci, Ray, and the Copper Queen, where the ore is coarser and harder, the draw points are larger and spaced farther apart.

After undercutting the ore is drawn slowly at first, until it begins to cave freely and afterward at the rate that best suits the nature of the ground and the requirements for ore. Whatever the rate of drawing it should be continuous to avoid packing of the ore. The ore should be drawn uniformly, particularly during the early stages of the operation, to assure a satisfactory recovery. The contact between the ore and the broken overlying capping should be an even plane, whether horizontal or inclined. If the early drawing is not done uniformly the mass of ore may cave along some plane of weakness, causing chimneys or cavities that may extend into the capping and allow it to mix with the ore. After the main mass of ore has been thoroughly broken and the block has caved to the surface, the result of subsequent irregular drawing is not as serious as it would have been earlier.

Dilution is caused by the irregular movement of ore and capping toward the drawing points forming pipes of waste, which may reach a draw point in advance of the top boundary of the ore, or by waste being drawn in from the sides or ends of a block. Dilution may also be caused by the general infiltration of fine capping down through coarsely broken ore. Moreover, in all caving there is a gradual mixing of the ore and capping as the ore is drawn down.

If properly drawn a large part of the ore comes to the chutes clean. After dilution starts the proportion of waste increases and that of the ore decreases.

In mining a block of ore by a caving method three things must be constantly watched and controlled.

1. The mining must be done in such a manner that excessive weight does not develop on the grizzly level. This is generally controlled by the rate at which the ore is pulled.

2. The ore should be broken sufficiently in caving to be handled subsequently without further breaking. As a rule, the slower the drawing rate the more the ore is broken; also the slower the drawing rate the more weight is likely to develop on the grizzly level. Generally a balance is maintained between the two to get a maximum efficiency.

3. The ore must be drawn in such a manner as to get a maximum recovery and minimum dilution with waste.

The point at which drawing of a block ceases depends upon the minimum grade of ore that can be handled and upon the value of ore that yields the best financial returns. If the plant capacity is large, drawing may continue until near the point of no profit. If not, to obtain adequate profits drawing must be stopped soon after dilution appears. The method is elastic in that by the sacrifice, for example, of 15 or 20 percent of the ore, the grade can be maintained at very nearly its original figure.

All phases of undercut block caving are interrelated. The development workings are laid out as a whole; changes in one set of workings must be met by corresponding changes in others.

Stopes usually are undercut either by running a checkerboard system of drifts and crosscuts and blasting the pillars or by belling out the finger raises to intersect, or by a combination of both methods. At Ray the cut-off workings consist of a series of low shrinkage stopes with pillars between; the cut-off is completed by blasting the pillars. In hard ore a system that will insure a complete undercut should be used; otherwise the back may not cave. Moreover, a small area of unbroken ore under a stope may transmit excessive pressure from the broken ore to the workings below.

The principal variation in the application of the method is in the manner of undercutting the blocks. Considerable variation also exists in the method of weakening the boundaries; in some cases, especially for the initial block in hard ground, the block is entirely cut off by narrow shrinkage stopes.

Gardner has recently discussed block caving as it is practiced in the copper mines of the Western States.[14] His discussion covers various aspects of block caving including geological factors affecting the practicability of the caving system, surface subsidence, principles of mining by the block-caving method, mechanics of caving, size of blocks and panels, development for block caving, methods of confining stopes to predetermined boundaries, methods of undercutting, drawing practice, ore recovery and dilution, and costs. The reader is referred to this for certain details of block caving

[14] Gardner, E. D., Undercut Block Caving Method of Mining in Western Copper Mines: Inf. Circ. 6350, Bureau of Mines, 1930, 44 pp.

which are not covered by the method of treatment employed in the present bulletin. Mitke [18] has discussed the caving method in several important articles.

The descriptions of the following mines illustrate the method, with the principal variations.

OHIO COPPER MINE, BINGHAM, UTAH

The method employed at the Ohio Copper mine was described in 1914 by Allen.[19] Although in this method sublevels were used for development, undercutting, and drawing the caved ore, it differed from previous sublevel-caving methods and from the method now termed the "sublevel method." The ore was caved in blocks of small height compared to those now caved in a single lift. The ore body was a mass of shattered quartzite containing copper sulphides disseminated throughout, and the walls were of the same formation. It was opened for a width of 400 feet, a length of 450 feet, and a depth of 1,300 feet and dipped at an angle of 50°.

The different levels were opened by drifts and crosscuts cutting the ore up into blocks 200 feet square. Between the main levels, on sublevels 30 feet apart vertically, they were subdivided into blocks approximately 30 by 50 by 25 feet. The finger raises radiated from the vertical raises. The sublevels were blasted down from the tops of these raises, the ore falling into the chutes and thence to the haulage level via the raise system. The top sublevel was mined first, then successively lower sublevels were mined and caved.

During October 1911, 56,811 tons were mined at an operating cost of 28.06 cents per ton, including development and equipment expense, or 21.97 cents per ton for actual mining. If the entire working force of 110 men is considered as producing ore, 17 tons were delivered to the bins per man per day. If the men engaged in the actual breaking of ore only are considered the output was 63 tons per man per day. In April 1913 the cost, including development and equipment, was 22.2 cents per ton.

RAY MINES, RAY, ARIZ.

The methods employed at Ray in 1929 have been described by Thomas.[17] The ore body is irregular both in plan and section and occurs in the form of two flat lobes. The main axis is 7,000 feet long, and the average width is 1,500 feet. The ore body is 40 to 400 feet thick and averages 120 feet. It is undulating but has a general dip of about 10 percent. The capping is 40 to 600 feet thick, averaging 250 feet.

A system of major faults with considerable fault breccia extends through the ore body. Another major fault runs at right angles to

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[18] Mitke, Charles A., Comparison of Branch Raise and Combined Shrinkage and Caving Methods: Tech. Pub. 186, Am. Inst. Min. and Met. Eng., 1928, 11 pp.; Caving Methods in General: Eng. and Min. Jour., vol. 125, Mar. 3, 1928, pp. 364–368; Caving with Branch Raises: Eng. and Min. Jour., vol. 125, Apr. 7, 1928, pp. 569–573; Combined Shrinkage and Caving Methods of Mining: Eng. and Min. Jour., vol. 125, May 12, 1928, pp. 764–767, and May 26, 1928, pp. 853–856; Stope Control, Dilution, and Recovery with Caving Methods: Eng. and Min. Jour., vol. 126, August 18, 1928, pp. 246–252.

[19] Allen, Carl A., Methods and Economics in Mining: Trans. Am. Inst. Min. and Met. Eng., vol. 49, 1914, pp. 396–398.

[17] Thomas, Robert W., Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.: Inf. Circ. 6167, Bureau of Mines, 1929, 27 pp.

this system near the west end, and a number of minor faults occur. The ore is a mineralized quartz-sericite schist, with subordinate quantities of porphyry, which, however, are mostly outside the ore zone. Several sets of planes of weakness occur, the most pronounced generally being with the schistosity. A second set runs at right angles to the schistosity and parallels its strike. Others run in different directions.

As a rule the ore breaks coarse, but in some places it breaks quite readily. As it falls into the caves it breaks in larger blocks than at most mines that employ the caving system. Parts of the ore body which contain a large proportion of sericite cave readily, but the ore is apt to pack if not drawn immediately and continuously after it is undercut. In such ground preliminary development must be well in advance of mining to allow the ore to dry out as much as possible.

Four variations of the caving system are used at Ray—the hand-tramming method, motor method, modified-motor method, and sublevel method. These variations differ mainly in the position of the haulage level below the bottom of the ore and in the method of handling the broken ore and are determined by the height of the ore above the haulage level and its thickness. There is no essential difference in the actual stoping of the ore, but preliminary development differs in each system. The sublevel system is preferred and is used in all types of ground where the ore above the haulage level is 150 feet or more thick.

The original system, as adapted from that in the Boston mine, consisted essentially of alternate shrinkage stopes and pillars. All stopes were carried up to the capping, and the pillars then were cut off at the base and caved. The shrinkage stopes were on 25-foot centers and driven as wide as ground conditions would permit.

The following description applies to the sublevel system, which is used where possible. A panel 200 feet wide usually is developed by four motor-haulage drifts on 50-foot centers and 40 feet below the grizzly level. (See fig. 55.) These drifts are timbered, and pony sets are erected over every fourth and fifth drift set. Chutes are installed, and raises 4 feet in diameter are driven to a height of 40 feet above the rail. The raises are inclined to connect with the sublevel draw laterals on 25-foot centers. The sublevel development consists of small, timbered, fringe drifts around the boundaries of the stoping block and small, untimbered laterals across each line of raises from the haulage level. These laterals are 25 feet apart and at right angles to the haulage drifts. A 13-inch grizzly is installed over the top of each storage raise.

End cut-off shrinkage stopes are driven at each end of the block in two lifts and carried to the capping. In the modified-motor and other systems they are usually carried in only one lift. Cut-off stopes are, of course, unnecessary on sides of the block adjoining previously mined blocks.

In undercutting, shrinkage stopes are carried at 25-foot intervals across the block. They are begun by inclined raises from draw sets installed over the grizzlies and carried up about 37 feet, as shown in figure 55.

All drilling in cut-off, undercut, and pillar stopes is done with hand-rotated stopers. The back of the stopes is kept level lengthwise and slightly arched in cross-section. Drill rounds consist of rows of 4 holes 6½ feet deep at intervals of 3 or 4 feet along the stope. The final round in undercut and pillar stopes consists of 6

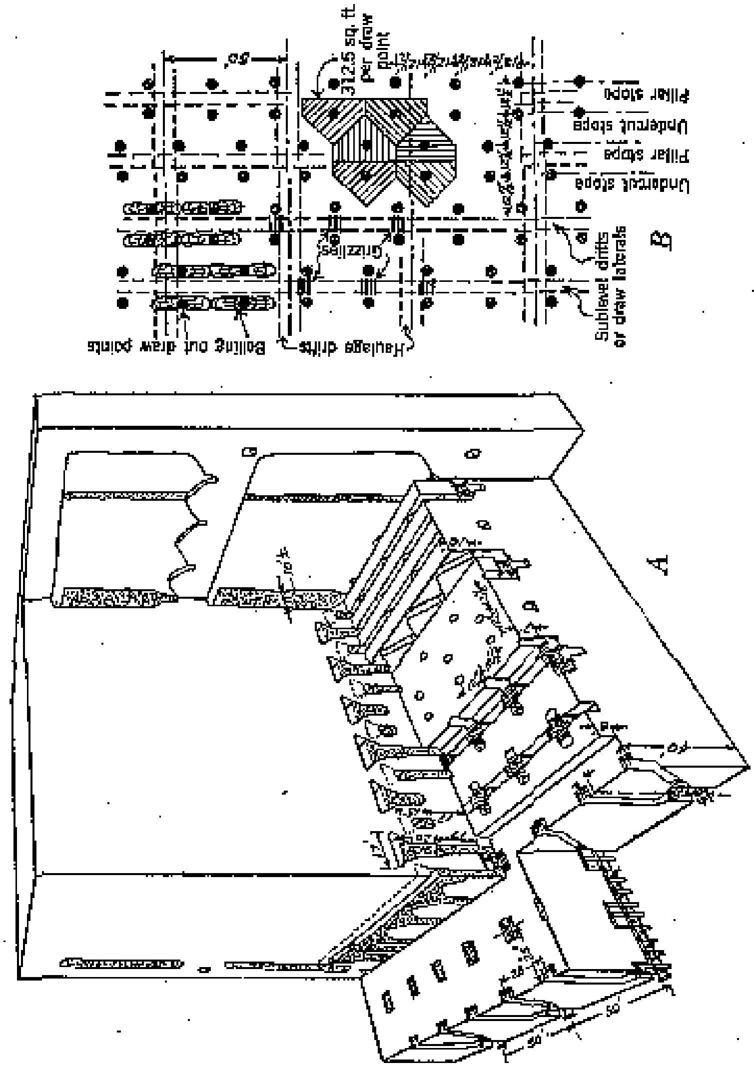


FIGURE 55.—Sublevel undercut block-caving method at Ray mines, Arizona: A, General drawing of block; B, drawpoint spacing.

holes to a row, the outside holes sloping outward 45°. As it is essential that the final round in the pillar stopes shall break through into the undercut stopes the holes are not loaded until they have been checked and approved by the engineer in charge.

Cribbed stope manways are carried up at the ends and middle of the stopes. Ordinarily 5 or 10 men work in each stope. A round is drilled and blasted over half the length of the stope on each of two shifts, while the swell is being drawn in the other half.

It is not practicable to work a whole panel as a unit on account of the expense of keeping the laterals open, so the block at the far end of the panel is worked first, then the next one in order, retreating toward the fringe drift.

Men designated as chute tappers draw the ore from the draw sets into the storage raises. It is often necessary to blast chunks of ore, and if the ore is hung up any distance from the draw point this is done with blasting sticks so that the tappers need not enter the chutes. No blasting is permitted on the grizzlies, hammers being used to break boulders too large to pass.

The method of undercutting used at Ray was adopted: (1) To provide pyramids or cones of solid rock above the drawing level, which support the broken ground and keep the direct weight from coming on the draw sets, and (2) to break the ground approximately 45 feet above the draw point. This broken ore acts as a cushion and as it is drawn off allows more thorough breaking of the main mass of ore before the latter reaches the draw points.

A uniform draw, particularly during the early stages of drawing, is essential for satisfactory recovery and for preventing excessive chimneying and consequent dilution of the ore with capping. The estimated dilution after more than 40,000,000 tons are drawn is about 10 percent.

The total cost of preparatory work for a block of ore containing 560,000 tons, including haulage drifts, subdrifts, raises, undercutting, shrinkage stopes, chutes, grizzlies, etc., is $81,939, or $0.146 per ton, divided as follows:

Estimated cost of preparatory work for a block of ore containing 560,000 tons, Ray mines

	Cost per ton of ore in block							
	Labor (1)	Timber (2)	Explosives (3)	Machine drills (4)	Haulage and hoisting (5)	Assaying, ventilation, sampling (6)	Miscellaneous mine expenses (7)	Total
Supply drifts, 325 feet	$0.0023	$0.0011	$0.0004	$0.0005	$0.0003	$0.0001	$0.0001	$0.0048
Grizzly drifts, 1,456 feet	.0049		.0017	.0015	.0009	.0002	.0003	.0095
Grizzlies, 50	.0052	.0091	.0005	.0004	.0002			.0174
Shrinkage stopes and undercutting	.0309	.0064	.0167	.0180	.0050	.0011	.0016	.0777
Total above grizzly level	.0463	.0146	.0193	.0204	.0064	.0014	.0020	.1091
Haulage drifts, 700 feet	.0092	.0064	.0012	.0019	.0012	.0005	.0004	.0188
Pony sets and chutes	.0039	.0036	.0002	.0010	.0005		.0002	.0075
Haulage raises, 1,648 feet	.0030		.0021	.0018	.0011	.0002	.0003	.0085
Total below grizzly level	.0161	.0092	.0035	.0046	.0028	.0005	.0009	.0365
Grand total	.0614	.0235	.0218	.0250	.0092	.0019	.0029	.1460

Direct stoping costs (excluding haulage, hoisting, and miscellaneous expense) during 1928 were as follows:

Direct stoping cost, Ray mines, 1928

	Cost per ton of ore						
	Labor	Timber	Explosives	Power	Machine shop	Miscellaneous	Total
Preparatory excavation	\$0.0681	\$0.0339	\$0.0070	\$0.0014	\$0.0010	\$0.0022	\$0.1030
Stoping	.0711	.0092	.0247	.0050	.0014	.0059	.1165
Drawing ore	.0840	.0120	.0190	.0001	.0001	.0006	.1267
Total	.2232	.0550	.0507	.0065	.0027	.0087	.3468

Following are statistics covering labor and consumption of explosives and timber per ton of ore during 1927 and 1928:

Year	Tons mined	Man-hour per ton		Explosives, pound per ton	Timber, board feet per ton
		Stoping only	Total mine pay roll		
1927	2,551,913	0.0767	0.748	0.2523	1.79
1928	3,243,160	.0692	.650	.3198	1.88

INSPIRATION MINE, ARIZONA

Numerous articles have been published on the development of the undercut-block-caving system at the Inspiration mine.

One of the earlier articles was written by Lehman [18] and contained a discussion of experimental tests on ore drawing, as well as a description of the early practice at this mine.

The following discussion is based principally upon a more recent article by Stoddard.[19]

The practice employed at Inspiration is an adaptation of the method developed by Felix McDonald at the Ohio Copper mine.

The mine is operated as three separate divisions; The Inspiration, Live Oak, and Keystone. The Inspiration division was the original one. The Live Oak and Keystone divisions cover a length of 4,000 feet. On the east end the ore outcrops, and on the west end it is covered by 500 feet of capping, the bottom being 900 feet lower.

The system employed in the Inspiration division differs from that in the other divisions chiefly in the spacing of grizzlies and draw points, the point of draw control, and the method of undercutting. In the Inspiration system the grizzly drifts and grizzlies are farther apart, the broken ore is drawn from square-sets above the grizzly level, and the block is undercut by a series of drifts on the undercutting level between which the pillars are slabbed and then drilled and blasted out. In the Live Oak division the ore is drawn from chutes over the grizzlies, and where the ground is strong enough it is undercut by connecting the tops of the finger raises and blasting out the pillars between rows of raises.

[18] Lehman, George R., Ore-Drawing Tests and the Resulting Mining Method of Inspiration Consolidated Copper Co.: Trans. Am. Inst. Min. Eng., vol. 55, 1916, pp. 218–231.

[19] Stoddard, Alfred C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.: Inf. Circ. 6169, Bureau of Mines, 1929, 23 pp.

When mining operations were begun there was some doubt as to the feasibility of caving a 200- to 300-foot back of ore, and the first mining was done with 75-foot backs. Later, however, the full height of the ore was taken in one lift.

Inspiration division system.—There have been numerous changes in the horizontal interval between haulage-level and grizzly-level drifts and in the spacing of main raises, branches, and finger raises to suit varying conditions. The general scheme, however, is shown in figure 56.

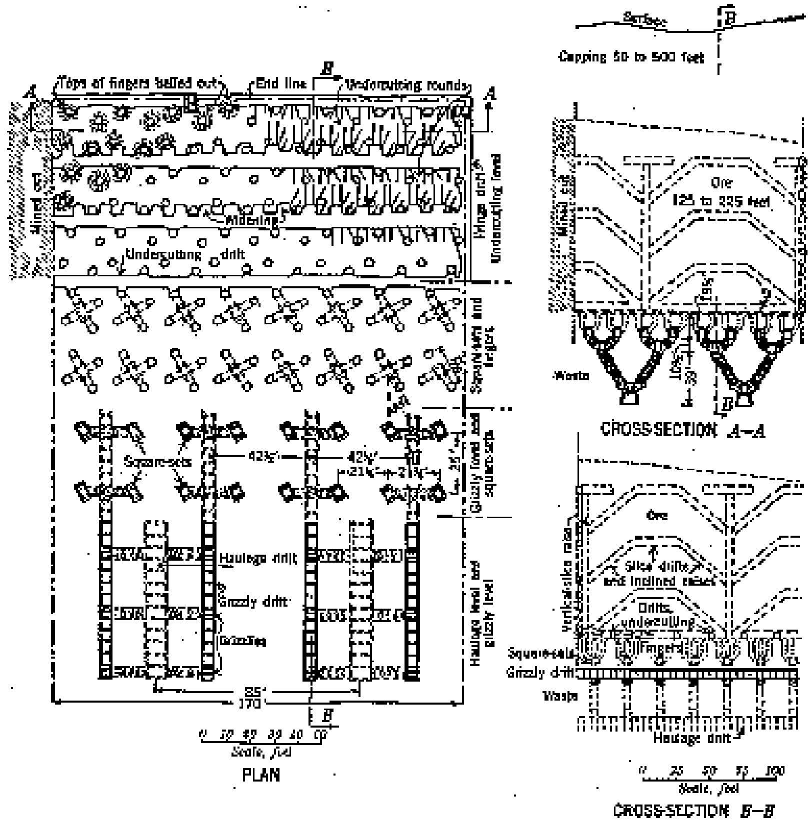


FIGURE 56.—"Square-set" block-caving system, Inspiration mine, Arizona.

The essential features of the system are: (1) A series of main haulage laterals under the ore body 20 to 200 feet apart (usually about 100 feet), along which pony sets and loading chutes are installed at intervals of 15 to 75 feet (usually 25 feet); (2) a system of raises driven from the pony sets in planes at right angles to the haulage drifts and at an inclination of about 50°; (3) a series of grizzly drifts intersecting the tops of these raises and parallel to and 25 feet on either side of the center of the haulage drifts, or 50 feet apart (the grizzly level being about 30 feet below the undercutting level); (4) two raises driven from the grizzly level at the top of each main raise, one in each direction at right angles to the drift (these raises being driven on a 40° to 42° slope for the first

18 feet, at the end of which a square-set is built in each raise; (5) four finger raises driven from each square-set, one from each side, to the undercutting level; (6) a series of drifts on the undercutting level, parallel to the grizzly drifts below and spaced on 25-foot centers, connecting the tops of the finger raises; (7) slabbing and blasting out of the pillars between the undercutting level drifts; and (8) drawing of the caved ore.

This system is termed locally the "square-set" caving method.

Live Oak division system.—The system employed in the Live Oak division is locally termed the "grizzly-control" caving method, and its essential features are shown in figure 57. One important disad-

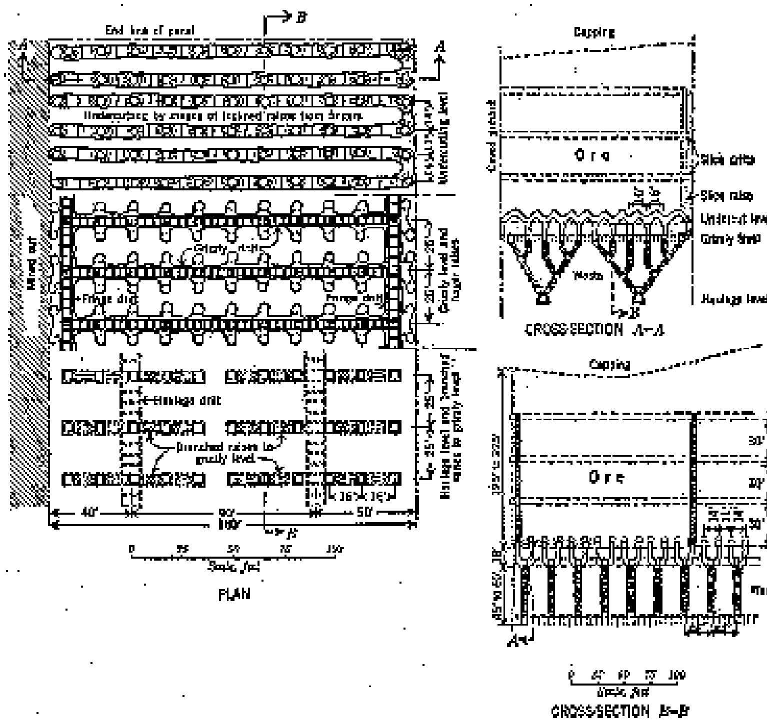


FIGURE 57.—"Grizzly control" block-caving method, Inspiration mine.

vantage of the system just described was that men had to go above the level into the square-sets to draw the ore, which makes supervision of drawing difficult. Moreover, the square-set itself is not easy to inspect and repair.

Ground conditions in the Live Oak division were entirely different from those in the Inspiration division. When the ground caved it broke into small pieces which packed together readily, so that when the ore was drawn "piping" occurred. The pipes were small in section and seemed to rise vertically, and many areas were imperfectly drawn. It was evident that closer spacing of draw points was required to overcome this.

To obtain closer spacing the raises were put up at right angles to the grizzly drift at 12½-foot intervals along the drifts. The usual square-sets were put in but with this difference: From one set four finger raises were put up as before, and from the next set only two. The grizzly drifts were still 50 feet apart. It was then found that good drawing could be done from the grizzly level with this method, but the grizzly drifts were too far apart. The next step was to drive the grizzly drifts 25 feet apart with grizzlies 12½ feet apart. In soft ground a number of draw chutes could be installed over one grizzly, aiding extraction materially if regular drawing was practiced. With this system the grizzly drifts may be either parallel or at right angles to the haulage drifts. If ground conditions permit the undercutting is done directly from the raises. The number of draw holes from a grizzly ranges from 2 to 10.

At the Inspiration mine it is unnecessary for each block to be cut off by boundary shrinkage stopes next to solid ore. Instead, the boundary is weakened by raises at the corners of each block and at intermediate points along the boundary, and drifts are driven from them at 30- to 40-foot vertical intervals. These drifts are driven around the block, and in some instances a stope round is blasted from the backs of these drifts when the ground requires further weakening.

During 1928 the stoping costs were as follows:

Direct stoping costs, Inspiration mine, 1928

	Cost per ton of ore hoisted
Labor	$0.146
Supervision	.007
Compressed air, drills, and steel	.005
Explosives	.017
Timber	.031
Other supplies	.002
Total	.208

MIAMI MINE, ARIZONA

The block-caving system used at the Miami mine has been discussed in several articles; the one by Maclennan [20] is abstracted in the following paragraphs.

In 1924, with the high-grade ore bodies virtually exhausted, it was decided to develop the low-grade ore body, which at that time was estimated to contain 36,000,000 tons of ore assaying 1.06 percent of copper, covered an area of 50 acres, and had an average thickness of 206 feet overlain by an average of 320 feet of barren capping. As developed to date (1930) the ore body is estimated to contain 108,461,700 tons assaying 0.88 percent of copper, of which 0.79 percent occurs as sulphide. The present dimensions are 3,500 feet east and west by 2,700 feet maximum width north and south, with an average thickness of 325 feet. The barren capping ranges from 250 to 500 feet in thickness.

The ore is a thoroughly fractured, mineralized schist, ranging from a hard, silicified schist to a soft, kaolinized schist. From a mining standpoint it is a free-caving ore body after it has been thoroughly dried by drainage and ventilation.

[20] Maclennan, F. W., Miami Copper Co. Method of Mining Low-Grade Orebody: Trans. Am. Inst. Min. and Met. Eng., 1930 Year Book, pp. 39-88.

In the earliest caving systems at Miami the entire width of the ore body was undercut and caved, beginning at one end and retreating the length of the ore body; this was followed closely by drawing of the ore. In drawing, an endeavor was made to maintain a plane of contact between the broken ore and the capping, with a dip away from the direction of retreat at an angle of 40° to 60° from the horizontal. Caving over widths of 500 or 600 feet resulted in excessive weight being thrown on the extraction openings, correspondingly high maintenance costs, and interference with orderly drawing of ore.

Later experience indicated that a width of 150 feet caused satisfactory caving of ore with moderate maintenance costs, and it became standard practice to cave and draw alternate panels 150 feet wide across the entire ore body. A year or so later, when the waste rock which had settled into these panels had consolidated, the pillar panels were caved and drawn back across the ore body. This method was well adapted to the comparatively low lifts then in use, but it was felt that it would be unsatisfactory with lifts as high as 300 feet or more, which were essential to low mining cost.

To maintain an angle of contact between the broken ore and capping of 40° to 60° in an ore column 300 feet high it would be necessary to maintain a length of 300 feet or more of extraction levels. In this system, using the inclined plane of contact and a great thickness of ore, there would be great danger of dilution.

It was decided that a system of mining utilizing individual stopes was best adapted to mining in one lift an ore body 300 feet or more thick. Advantages claimed for this method are summarized as follows:

1. The block to be caved is confined to a definite area surrounded on all four sides by solid ground in the original stopes and by consolidated fill (partly or wholly) in the pillar stopes. Undoubtedly the support of the individual stope is much better, hence maintenance costs are lower, and repair work interferes less with drawing of ore.

2. The order of mining the stopes is such that pillar stopes are not mined until the waste fill along any boundary has been consolidated for several months. During this period the fill becomes a substantial support. The stopes are mined in the order shown in figure 58, *A*. If the contact between broken ore and waste at the retreating working face of the panel (in the old panel system) is classified as waste the panels would have 21,450 feet of waste boundary compared with 17,700 feet of waste boundary in the individual stope system.

Therefore, there should be less dilution from the waste boundaries in the individual stope system. In the stope method 25 stopes are mined in solid ground with no waste boundaries, 14 stopes with 25 percent of waste boundary, 15 with 50 percent of waste boundary, and 14 with 75 percent of waste boundary; 8 stopes are entirely surrounded by waste.

3. In the individual stope system the ore is drawn down evenly over the entire stope area, resulting in a horizontal plane of contact between the broken ore and capping. For high lifts the horizontal plane of contact is unquestionably superior to the inclined plane of contact.

4. With the same area of ore body the stope system provides a greater number of working places, and the work is standardized more easily, resulting in a higher rate of production.

5. In the stope method production can be distributed more conveniently over the area so as to deliver the ore to the various drifts on the haulage level.

The first individual stopes were 150 by 300 feet. Later the stopes were 150 by 150 feet, although a few were 150 by 200 feet to conform to spacing of certain haulage drifts.

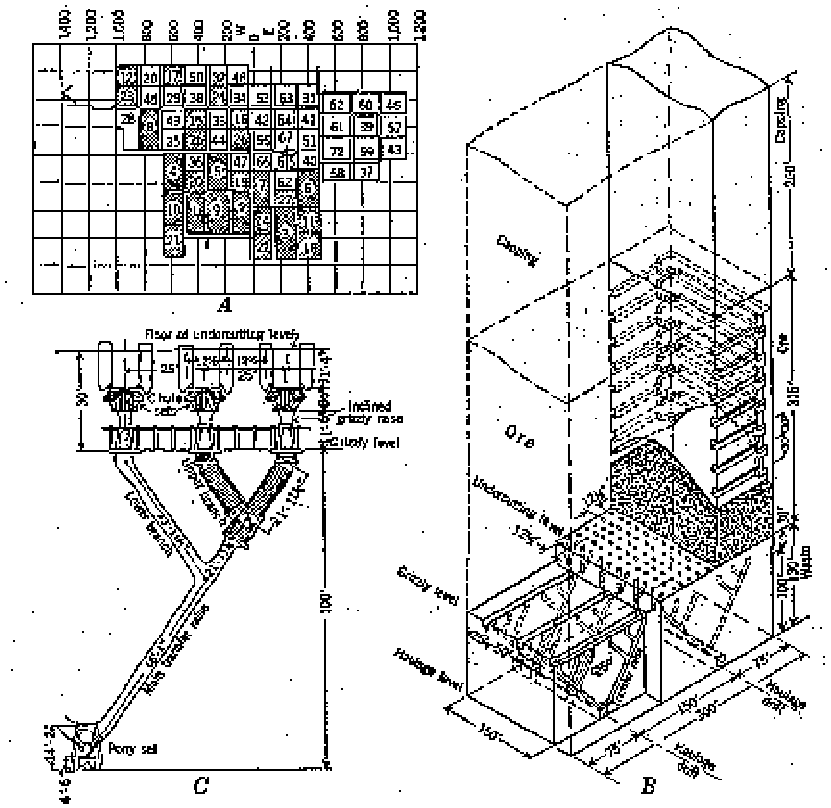


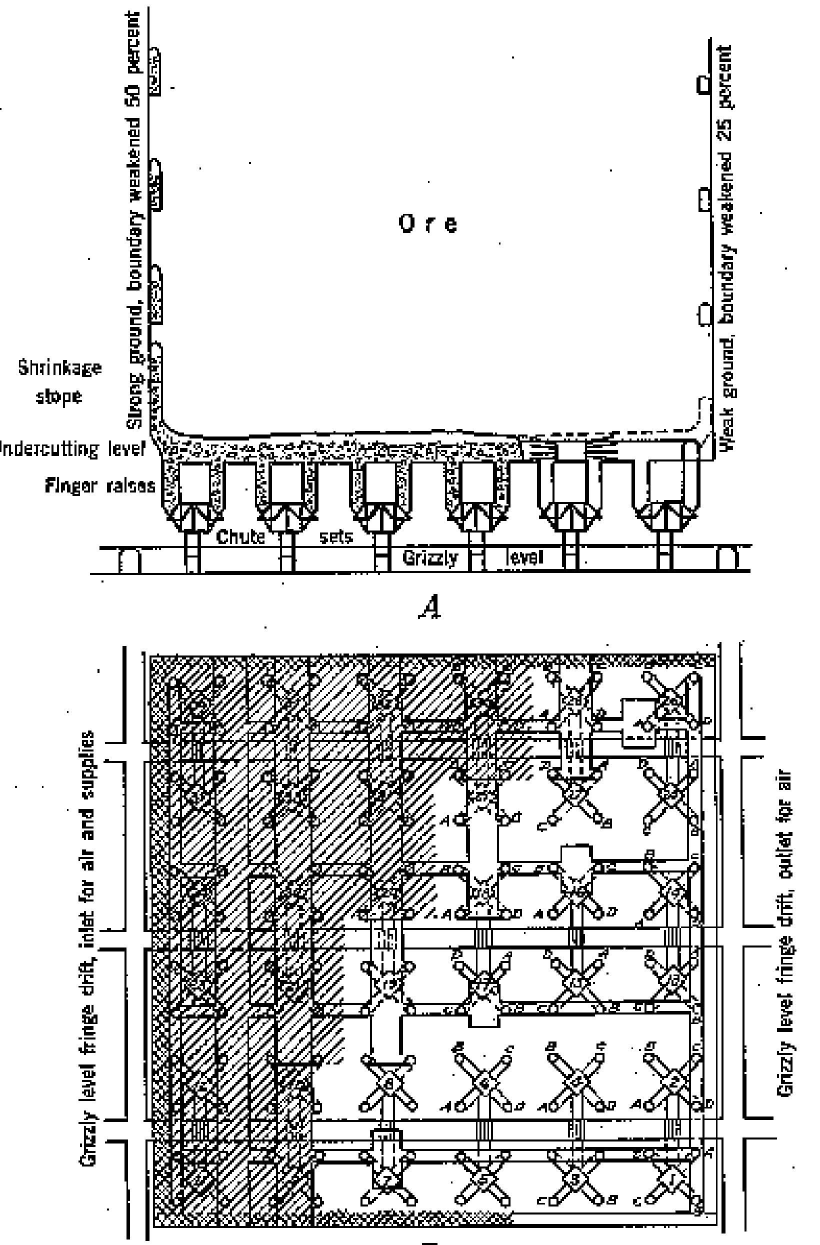
FIGURE 58.—Undercut block caving, individual stope system, Miami mine, Arizona: *A*, Plan of 510-foot boundary caving level showing order of mining stopes; *B*, isometric drawing of 150- by 300-foot stope; *C*, detail of ore transfer raise system.

The ore is undercut below the entire area of the stope and allowed to cave by its own weight. If left to itself it might arch over to the center and stop caving, or it might follow slips and cave beyond its vertical boundaries. A satisfactory method of isolating a caving area with low lifts is by a narrow shrinkage stope around the boundary. With a high lift, however, this method might cause the block to settle down like a plug or piston and crush the mine workings below, and the broken ore would arrive at the chutes in large blocks unsuitable for drawing. It is desirable to have the back of the stope hang up and come under strain, so that it will become highly fractured and cave down slowly in small pieces and will cave to, but not beyond, its vertical boundaries. To accomplish this boundary caving drifts are driven around the stope at suitable vertical intervals, and raises are put up at the four corners. The boundary drifts are 7½ feet high and were originally at 30-foot vertical intervals. In hard ground the boundaries may be weakened further by drilling and shooting in the backs of the drifts. More recently, the practice has been to space the drifts 45 feet apart vertically, thus weakening the boundary plane 33⅓ percent.

Figure 58, *B*, showing the individual stope system, is an isometric drawing of a 150- by 300-foot stope; it also applies to the present 150- by 150-foot stopes. The Miami ore breaks up fine and tends to pack, and when drawn it pipes up vertically. Draw points therefore had to be closely spaced, and a 12½-foot interval in both directions was adopted, giving a spacing of 50-foot centers for the grizzly drifts and 25-foot centers for the grizzlies or tops of the transfer-raise branches.

Details of the transfer and grizzly raises are shown in figure 58, *C*. The grizzly drifts are driven at right angles to the haulage-level drifts 100 feet below and are vertically over the transfer raises, which are driven at both ends from the boundary or fringe drifts. During actual mining the grizzly levels are the base of operations for undercutting and drawing of ore. Grizzlies are installed in openings over the transfer raises. Grizzly raises are driven on both sides of, and at right angles to, the grizzly drifts. These are inclined 42° for 14 feet and then are run vertically for 10 feet. The inclined section is small (3½ by 3½ feet) to resist crushing, and the upper part of the vertical section is enlarged to accommodate the chute set, which is oriented as shown in figure 58, *C*, to give the correct spacing of the draw points above. Four finger raises 3½ feet in diameter are driven from the chute sets and inclined to a point 8.85 feet horizontally from the center of the set, thence vertically to a horizontal plane 30 feet above the floor of the grizzly level, which they intersect at points 12½ feet apart.

As soon as enough finger raises have been put up the undercutting level is begun from the tops of these raises. Figure 59 shows the undercutting level and its relation to the grizzly level, chute sets, and draw points. The chute sets are numbered from 1 to 36, and the draw points are lettered. The undercutting level is opened by four small drifts driven parallel to the grizzly drifts through every third line of draw points; these are 37½ feet apart and equidistant from each side of the central grizzly drift. They are connected at both ends by fringe drifts. Undercutting is begun by putting up a shrinkage stope 2 or 3 rounds from one of the fringe drifts and is usually carried along the side boundaries and the other end of the stope as undercutting of the main body progresses. The main body is undercut from large drifts 8 feet wide, which are driven at right angles to the small drifts. This work is carried back diagonally as shown hatched on the plan (fig. 59, *B*). The sides and backs of the drifts are drilled and blasted to complete the undercut. The tops of the finger raises are funneled as undercutting progresses.



B

FIGURE 60.—Undercutting and grizzly levels, Miami mine, showing method of undercutting and relation of grizzly and undercutting levels: A, Composite section; B, composite plan.

Drawing of ore usually does not start until the stope has been completely undercut. In drawing there are two main objectives: (1) To draw a maximum tonnage of ore with minimum dilution by waste capping; and (2) to regulate the drawing to avoid, if possible, or relieve damaging weight on the extraction openings below.

Drawing is under the supervision of three stope engineers on day shift. They inspect the stopes daily and issue ore-drawing orders to the bosses and chute sealers on the three shifts. Drawing is done by crews of two men each, a chute blaster and a chute tapper. The blaster draws the ore from the chutes in the chute set, and the tapper keeps the grizzly clear on the grizzly level. Each crew usually draws 12 to 15 finger raises per shift, blasting 5 to 8 times and drawing approximately 400 tons per shift. Normally 50 tons are drawn daily from each chute listed on the draw orders. The aim is to draw this amount in rotation from each chute in the stope, but this routine is necessarily varied because of the appearance of waste capping in a chute, necessity for repairs, weight on timbers, etc.

Drawing is continued from each chute until the grade of the ore drops to a point where it is not profitable enough. Usually this does not occur until 100 percent of the expected tonnage has been drawn.

The tonnage, grade, and copper extraction from 13 stopes which had been completely drawn before 1930 are given below.

Extraction tonnage and grade

	Expectancy		Mined		Percentage extraction		
	Tons	Copper, percent	Tons	Copper, percent	Tonnage, percent	Grade, percent	Copper, percent
Total of 13 stopes	11,039,070	1.0260	12,710,378	0.8124	115.15	58.93	102.40
Best original stope	[illegible]	1.0368	1,216,521	[illegible]	121.28	97.14	117.81
Best pillar stope	311,560	1.0840	697,827	.9358	121.38	67.88	108.63
Poorest original stope	1,071,645	.8701	1,053,153	.7768	99.25	80.48	87.94
Poorest pillar stope	1,098,818	1.1067	1,025,052	.6605	93.33	61.29	76.86

Stoping cost per ton, as given in Maclennan's paper for the 4-year period October 1, 1925 to September 30, 1929, during which 16,556,296 tons were mined, was $0.13628, and development cost was $0.10000—a total of $0.23628. Haulage, hoisting, ventilation, general underground, engineering, sampling, mine-surface, and mine-accident expense added to this figure give a total direct mining cost of $0.39937 per ton.

It is not entirely clear from the figures presented what the direct stoping cost was on a basis comparable to that used at other mines described in this bulletin.

From a table in the paper it appears that 10 cents per ton for development covers the driving of haulage-level drifts, transfer raises, grizzly-level drifts, finger raises, and boundary drifts and raises; the installation of haulage-level chute sets, grizzlies, and grizzly chute sets; the driving of finger raises and undercutting drifts; undercut-level mining; and the drilling and blasting of boundary drifts and boundary corner raises. This figure then covers all work preliminary to drawing, and the conclusion is that $0.13628 covers drawing and maintenance of extraction drifts and raises, chutes, square-sets, etc., during drawing. According to the table the cost per ton of ore extracted for development up to the chute sets above the grizzly level was $0.0905, and the cost per ton for driving the finger raises and all work on the undercutting level was $0.0299, a total of $0.1204 per ton (from 1925 to 1928, inclusive). From the figures in this table Maclennan concludes "that the uniform charge for development of 10 cents per ton is ample."

HUMBOLDT MINE, MORENCI, ARIZ.

Block caving at the Humboldt mine, Morenci, Ariz., has been described by Mosier and Sherman.[21]

The country rock at Morenci is a quartz-monzonite porphyry and a series of sedimentary rocks—limestone, shale, and quartzite—which are generally metamorphosed. The ore bodies of the Humboldt mine are in the main porphyry area. The ore body being mined in 1929 was in zones of fracturing and had no definite walls. Irregular fracturing occurred in three directions, but the fracture planes had been more or less recemented with quartz and pyrite. The planes of weakness were 2 to 18 inches apart.

The ore reserves lie principally in one mass in the porphyry along two intersecting fault zones over a maximum length of 2,000 feet and a maximum width of 600 feet. The vertical range is about 1,000 feet. Much of the porphyry wall rock carries at least 0.4 percent of copper. It is generally stronger than the ore physically and carries less kaolin and sericite. The grade of the ore reserves in 1929 was about 1.90 percent of copper (total), of which 0.20 percent was acid-soluble copper.

Although the ore was somewhat harder than at most other porphyry mines in the Southwest, its caving was made possible by the major faults with their sympathetic sheeting, by the jointing throughout the ore body, by the minute fracturing of the ore in all directions, and by the alteration of the ore.

Figure 60 shows block caving at the Humboldt mine. In this system the undercutting level is 20 feet above the grizzly level, and the latter was connected to the haulage level 60 feet lower by branch raises.

The ore body was divided into stoping panels 112 feet wide which extended on the strike of the ore from one side to the other. As mining proceeded from one end of the panel it could be followed by another stoping operation in an adjoining panel after the caved waste capping had settled in the finished stope. The ore from each panel was delivered to two haulage drifts 56 feet apart which paralled the panel. Chute raises were driven to the grizzly level at 28-foot intervals along the haulage drifts. The grizzly drifts were at right angles to the haulage drifts and across the stope panels. These were also 28 feet apart and directly over lines of raises from the haulage-level chutes. As laid out originally the grizzlies were 14 feet apart, but when this paper was written they were symmetrically but unevenly spaced to suit the finger raises, and the grizzlies were set in

[21] Mosier, McHenry, and Sherman, Gerald, Mining Practice at Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz.: Inf. Circ. 6107, Bureau of Mines, 1929, 34 pp.

pairs so that the centers of certain pairs were 14 feet apart. Two short finger raises were connected with each grizzly by openings 4 by 6 feet in section cut in the sides of the drifts. On the undercut level the finger raises were 14 feet apart in one direction and 18 feet 8 inches apart in the other direction, and they were funneled out until the rims of the funnels met. After the necessary finger raises were driven, shrinkage stopes 6 feet wide were carried up around the panel boundaries high enough to cut the panel free from the surrounding

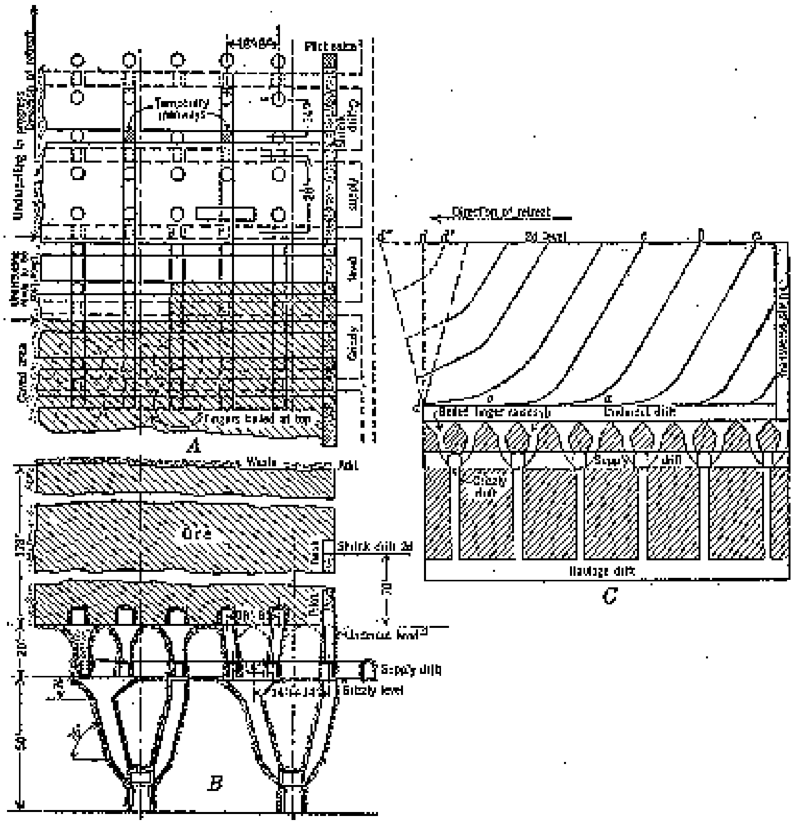


FIGURE 60.—Standard block-caving system, Humboldt mine, Morenci, Ariz.: *A*, Plan of undercutting level and grizzly level (dash lines); *B*, vertical cross section; *C*, typical section showing progressive stages of caving.

ground and to direct the line of shearing to the top of the ore along the desired planes. In soft ground these stopes were generally carried up 35 feet. At this height it was usually possible to leave sections 50 to 60 feet long midway between the pilot raises (fig. 60) and to shrink above this point only in sections 40 to 50 feet on each side of the pilot raises.

In beginning the undercut, one or two drifts were run lengthwise of the panel across several grizzly lines. Undercut drifts 4 by 6 feet in section connected the tops of finger raises, both lengthwise and across the panel, forming a gridiron pattern of drifts with pillars about 10 by 14 feet in plan between them. The backs of the drifts were drilled 6 feet deep in fan-shaped rings of stoper holes at 4-foot intervals. These holes were fired simultaneously with holes drilled in the pillars. Two or three pillars usually were blasted at a time, beginning in the heaviest ground and progressing across the panel.

When a block 150 feet high was mined, the quantity of ore actually drilled and blasted, including the necessary development work, was less than 10 percent of the total ore removed.

When undercutting had been completed, drawing of the caved ore began. The ore was drawn slowly at first until it started to cave freely. In 1928 the average stoping lift was 164 feet, whereas on the next lower stoping horizon it increased to 270 feet. An increase in height above 250 feet reduces the stoping cost very little and increases the time required for the ore to reach the mill after it has been broken. This delay in drawing may be detrimental to ores that oxidize rapidly, when concentration is by gravity or flotation methods.

The proper rate of drawing depends upon the rate at which the ore caves; if it is drawn very rapidly, too large a space is left between the broken ore and the unbroken back, and the ore is apt to break in large masses. If it is drawn too slowly, the broken ore will partly support the back and delay caving.

If a "pipe" of cap rock reaches a draw point in advance of the upper boundary of the ore the finger is sealed, and ore is drawn from the surrounding fingers until the pipe is broken.

During the initial stages of drawing, the angle of retreat is 60°; later it is reduced to 50° or less. Figure 60, *C*, shows the progressive stages of caving, looking at right angles to the line of retreat.

The estimated cost of preparatory work for caving a block of ore containing 120,000 tons, exclusive of tramming, hoisting, sanitation, ventilation, sampling and assaying, and general expense, is as follows:

Estimated cost of preparatory work for block of 120,000 tons, Humboldt mine

	Cost per ton of ore					
	Labor	Drills and tools	Explosives	Timber	Other supplies	Total
Supply drifts	$0.0048	$0.0015	$0.0012	$0.0021	$0.0000	$0.0103
Grizzly drifts	.0108	.0045	.0035	.0053	.0003	.0244
Grizzlies	.0139	.0030	.0014	.0210	.0005	.0400
Fingers	.0071	.0095	.0061		.0009	.0235
Pilot raises	.0021	.0009	.0009	.0005	.0001	.0045
Shrink drifts	.0064	.0018	.0014		.0001	.0097
Subtotal	.0418	.0224	.0145	.0292	.0024	.1100
Shrinkage stopes	.0075	.0091	.0060		.0012	.0238
Undercutting	.0233	.0220	.0165		.0010	.0640
Subtotal	.0308	.0311	.0243		.0022	.0884
Total above grizzly level	.0726	.0532	.0388	.0209	.0040	.1984

These costs do not include tramming, hoisting, sanitation, ventilation, assaying and sampling, and general mine expense, which would bring the total cost above the grizzly level to $0.228 per ton.

Direct stoping costs for 1928 were as follows:

Direct stoping costs, Humboldt mine, 1928, 1,548,958 tons mined[1]

	Cost per ton mined					
	Labor	Supervision	Compressed air, drills, and steel[2]	Explosives	Timber	Total
Stope preparation:						
Drifts; driving, timbering, and tracks	$0.014			$0.004	$0.006	$0.024
Raises and grizzlies	.016			.006	.015	.035
Breaking ground and chute tapping[3]	.074			.032		.105
Stope repairs	.011				.010	.021
Undistributed		$0.040	$0.055			.095
Total, direct stoping	.114	.040	.055	.043	.031	.273

[1] General development, haulage drifts, timbering and tracks on haulage level, haulage-level chutes, and repairs not included.
[2] Includes sundry supplies.
[3] Includes undercutting and shrinkage stopes.

If haulage-level development, main raises, and chute fronts are included the total would be $0.356 per ton.

During 1928, 62.76 tons were produced per man-shift stoping (including labor on boundary shrinkage stopes, undercutting, belling fingers, chute tapping, and stope repairs but no development labor). The consumption of explosives was 0.19 pound per ton of ore and the consumption of timber 0.25 board foot per ton.

CRESTMORE MINE, CRESTMORE, CALIF.

The following brief description of the block-caving method at Crestmore is added to the discussion of caving, as the method was adapted from western copper-mining practice and is used successfully for mining a hard, blocky, crystalline rock; it had previously been considered that block caving was not feasible for mining ground of this nature. The method has been described by Robotham.[52]

The mine is in the Stanley bed of limestone, which is approximately 270 feet thick and dips 45° to 55°. When first observed the limestone appeared to be massive, but during opencut operations it was found to be traversed by minute watercourses and slightly faulted and fractured; it is hard and brittle. The deposit was water-logged and was drained from workings 96 feet below the lower mining level before stope development was begun. The underground mine is below opencut workings.

Haulage drifts for the first block to be caved were 40 feet below the grizzly level and run on 50-foot centers, with loading chutes from the grizzlies 25 feet apart. Grizzly drifts, at right angles to the haulage drifts, are 25 feet apart with draw points on 25-foot centers. The top of the undercut workings is 15 feet above the bottom of the grizzly level. The blocks are 200 by 200 feet; the height of the first lift mined was 100 feet. The blocks are cut off from virgin ground by shrinkage stopes.

[52] Robotham, C. A., Mining Limestone by a Caving Method at Crestmore Mine of the Riverside Cement Co., Crestmore, Calif.; Inf. Circ. 6795, Bureau of Mines, 1934, 20 pp.

Undercutting is done by a series of room and pillar workings run diagonally to the grizzly drifts. The pillars are midway between each pair of grizzly drifts; final undercutting is effected by blasting the pillars.

The principal departure from copper-mining practice is in putting in the draw points. The raises from the haulage level are branched just below the grizzly level; grizzlies are on either side of the drifts. The draw point is driven horizontally about 10 feet beyond the grizzlies then raised on an incline into the undercutting area. Protective cones of solid rock are left over each grizzly drift. In effect the draw points are similar to grizzly blasting chambers used in connection with shrinkage stopes.

The rock caves satisfactorily after undercutting, but most of it breaks in relatively large fragments. About one-third of the rock must be reduced in size by blockholing in order to get it through the grizzlies. It has been found that large boulders require no more drilling and blasting than fragments of intermediate size. A low-strength explosive is used for blasting boulders.

The following table shows the cost of drifting, raising, and timbering in connection with the development of 1,258,379 tons of rock and the cost of cut-off stoping, undercutting, raising, and extraction during active mining of 1,775,696 tons.

Cost per ton of development and active mining

Cost per ton	Labor	Timber	Explosives	Power and miscellaneous	Total
Development	$0.0374	$0.0062	$0.0155	$0.0044	$0.0647
Active mining	.0600	.0051	.0400	.0110	.126

Normal production costs during 1931, which are considered fairly representative of results obtained by this method of underground operation, were as follows:

Production costs per ton hoisted

	Labor	Timber	Explosives	Power and miscellaneous	Total
Drawing	$0.0939	$0.0054	$0.0250	$0.0051	$0.1343
Haulage and hoisting	.0392	.0002	.0001	.0060	.0458
General underground expense	.0254	.0001		.0092	.0350
General mine expense	.0176			.0028	.0169
Total	.1761	.0057	.0273	.0229	.2330
Hanging-wall rock					.0132
Total					.2462

In the preceding tabulations general underground expense covers mine supervision, mine illuminants, man hoist, changehouse, ventilation, mine tools, and mine drainage; general mine expenses cover expenditures for the mine office, warehouse, engineering, framing shed, and general supervision pertaining directly to underground operations.

The charges for explosives shown are for blockholing at the grizzlies.

The following tabulation for 1932 shows production costs more closely approaching normal conditions than those covering the early stages of operation.

Summary of costs, in units of labor and supplies

Year	Tons per man-shift		Explosives per ton, pound	Lumber per ton, board feet	Power, kilowatt-hours per ton
	Mining	All employees			
1932	17.3391	16.8970	0.6187	0.7500	4.3107

SOUTH AMERICAN COPPER MINES

BRADEN COPPER CO., CHILE

The mining method at the Braden Copper Co. mines, Sewell, Chile, has been described by Webb and Skinner.[23] The ore bodies lie around the periphery of an explosive vent in the form of crescent-shaped deposits, limited on the inside, where they are of high tenor, by the tuff contact. The upper limit is formed by the bottom of the oxidized zone 50 to 100 meters below the surface and the lower limit by the contact with the primary zone. The andesite porphyry surrounding the vent is intensely fractured; the width of the zone of intense shattering ranges from 100 to 600 meters. Ore bodies were formed by mineralizing solutions rising around the periphery which deposited chalcopyrite and other minerals in the irregular fractures of the andesite porphyry.

The undercut block-caving system employed at Sewell differs considerably in detail from the caving systems previously described. A branch-raise system is used to transfer the ore to the main haulage level, but the grizzly level and finger raises used at the other mines are not employed here. Instead, the ore is undercut from a series of timbered drifts, as shown in figure 61, *1*, *2*, *3*, and *4*, and the caved ore is drawn from chutes installed along the drift in alternate sets into 1-ton cars which are trammed by hand a maximum of 15 meters (49.2) feet to transfer raises. The transfer-raise system is shown in figure 61, *5*.

This block-caving system, by which over 85 percent of the ore was mined in 1932, was evolved from an earlier system in which the ore was mined by shrinkage stoping in original stopes separated by pillars which were caved and drawn with the shrinkage ore. Formerly it was considered that 50-meter (164-foot) lifts were the maximum that could be caved successfully, but later 100-meter (328-foot) lifts were successfully caved without loss of ore or excessive dilution. All development is now laid out for 100-meter lifts.

[23] Webb, J. S., and Skinner, T. W., Mining Methods and Costs at the Braden Copper Co.'s Mines, Sewell, Chile: Inf. Circ. 6565, Bureau of Mines, 1932, 13 pp.

Undercutting is begun by taking a section of a drift 6 meters (19.68 feet) long and drilling a 6-foot round above the top lagging and through the chutes, as shown in figure 61, *1*. The ore broken in this round is drawn out through the drift chutes, leaving a small open stope about 7 feet high above the drift timbers. Access to this stope is through the end chute next to the adjoining solid ground. Before the first blast an entrance chimney is completed from the

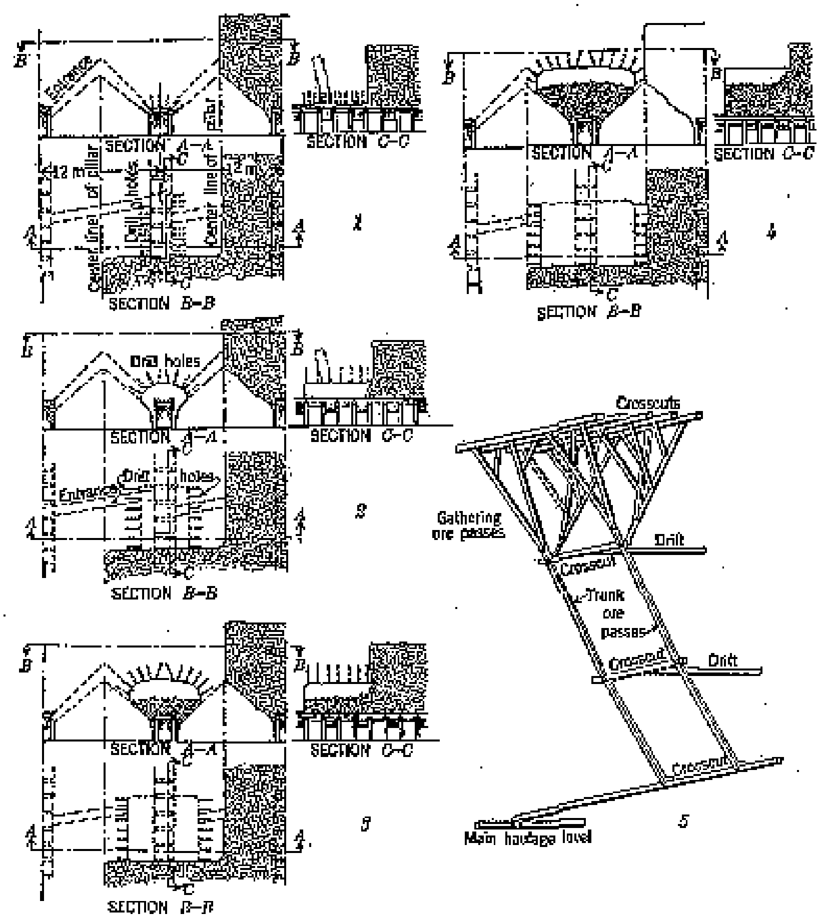


FIGURE 61.—Undercut block-caving system, Braden Copper Co., Chile: *1*, *2*, *3*, and *4*, Details of undercutting—first, second, third, and fourth rounds; *5*, ideal transfer raise and ore-pass system.

adjacent drift to the center of the pillar, as shown in the figure. This entrance is 1.5 by 1.5 meters in section, is driven on a 48° slope at an angle 8°30′ from normal to the drift, and meets a similar chimney driven from the last chute in the undercut stope. Second and third widening rounds are drilled and blasted; then the ground usually shows signs of weakening, and it becomes necessary to stull up the back before drilling and loading the final round which breaks to the adjacent cave. (See fig. 61, *4*.)

After an undercut section has been completed 300 tons of ore are drawn from each chute to allow room for further caving. These chutes are then sealed and remain closed until the next progressive slice has been undercut. By this procedure the miners never have to work against an open stope, and the part of the entrance chimney leading into the last undercut stope affords a means for observing the caving in the preceding slice. An undercut crew consists of four miners in each undercut stope, who are paid on contract at so much per meter of drift undercut.

The ore is drawn by trammers, each of which has a minimum of 10 chutes to pull in order that a daily tonnage, specified by the draw engineer, may be drawn from each chute. An average of 55 tons per trammer is drawn per 8-hour shift. Chute blasters aid drawing operations by bulldozing chunks in the chutes. Usually one chute blaster is employed to every 3.5 to 4 trammers. An automatic car counter operated by an arrangement of springs and cams was designed and installed on each 1-ton car used in drawing. The trammers must load 1 ton into the car and dump it before the automatic device will register.

The direct stoping cost during 1928, including drawing and hand tramming (which corresponds to the chute tapping and work on the grizzlies in the other caving operations described), was as follows:

Direct stoping costs per ton of ore delivered, Braden mines, 1928, 5,174,017 tons delivered

	Labor [1]	Compressed air, drills, and steel	Explosives	Timber	Other supplies	General expense	Total
Development	$0.0101	$0.0059	$0.0064	$0.0033	$0.0020	$0.0206	$0.0483
Undercutting	.0777	.0069	.0152	.0415	.0063	.0414	.1890
Hand tramming	.0393				.0027		.0420
Total	.1271	.0128	.0216	.0448	.0110	.0620	.2793

[1] Includes supervision.

ANDES COPPER MINING CO.

The caving system employed at Potrerillos has been described by Greninger.[24]

The ore is found in an intrusion of porphyry, generally classified as quartz-diorite porphyry. The surface area of the intrusion is about 2 km from north to south and has a maximum width of 1 km; the widest part is near the northern limit, and it tapers to a point at the southern end. All of the porphyry contains some copper, but a considerable part of the intrusion is too low grade to be considered ore. The intrusion has tilted the sedimentary rocks—quartzite, limestone, and sandstone—50° to 60° from the horizontal, and the ore bodies within the porphyry generally have the same dip. Of the total tonnage developed, about 35 percent is oxidized ore and 65 percent sulphide ore; fortunately there is little mixed ore, the change from oxide to sulphide being sharp.

[24] Greninger, I. L., Mine Development and Underground Construction of Andes Copper Mining Co. at Potrerillos, Chile: Trans. Am. Inst. Min. and Met. Eng., 1929, Year Book, pp. 144–166.

Exploration by churn drilling showed that at some points the oxide ores reached the surface and that at others there was barren capping. Generally there is little or no capping over the northern part of the ore body, whereas the southern sulphide ore body is covered to a vertical depth of 100 to 200 meters. Metallurgical convenience dictated that mining should be begun first in the sulphide ore at the south end of the ore body. As a matter of fact several million tons of oxide ore in this locality, while not directly overlying the sulphide ore, were within the zone that would cave as mining of the sulphide ore progressed. In mining this oxide ore the regular caving method was employed, except that no control was practiced in drawing as there was no capping over it.

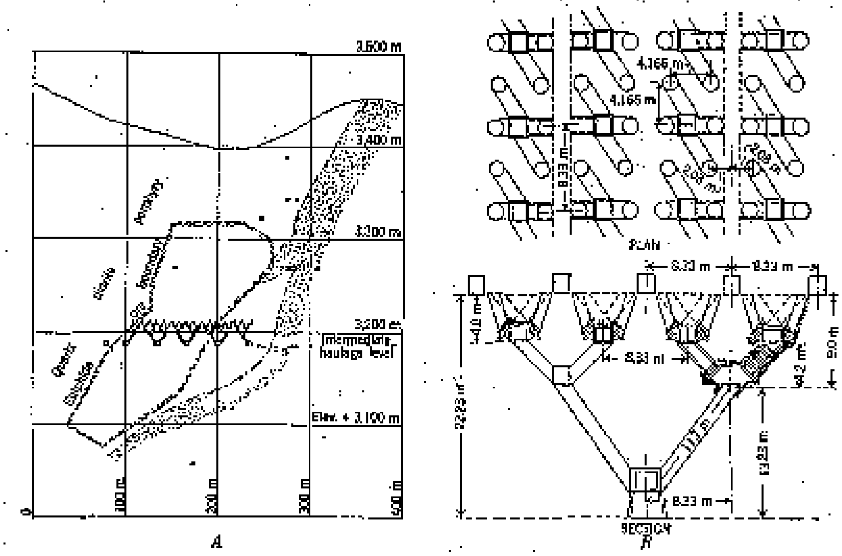


FIGURE 62.—Undercut caving, Andes Copper Mining Co., Chile: *A*, Transverse section, south sulphide ore body; *B*, plan and section showing branch-raise system.

Figure 62, *A*, is a tranverse section through the south sulphide ore body where mining of sulphide ore was begun and shows the workings on and above the intermediate haulage level. This level is about 130 meters below the top of the sulphide ore body. Haulage drifts are turned off the gathering drifts at intervals of 33.33 meters across the ore body, and main raises are put up at 6.25- and 9.37-meter intervals. Grizzly drifts 12 meters above the haulage drifts are spaced 16.66 meters center to center and parallel the haulage drifts below. Branch raises are driven at intervals of 8.33 meters along the grizzly drifts, and the block served by each branch-raise system measures 8.33 meters square. (See fig. 62, *B*.) The undercutting level is 10 meters above the grizzly level, and on it undercutting drifts are driven 8.33 meters apart and parallel to the grizzly drifts.

The block first mined extended entirely across the ore body from west to east, a distance of 200 meters, and measured 70 meters from north to south. Experience demonstrated that this block was larger than advisable, and new blocks were laid out across the ore body only 50 meters from north to south.

In each branch or drawing-off raise is a central set containing four chutes, from each of which raises are driven to the undercutting level. (See fig. 62, *B*.)

In undercutting a short crosscut is driven from the undercutting drift where the work is begun. This is driven 4.5 meters into the pillar, and undercutting holes 3 to 4 meters long are drilled from it. Undercutting advances diagonally across the block.

Haulage drifts are timbered with 10- by 10-inch Douglas-fir sets, and most of the main or transfer raises must be timbered with 6- by 12-inch cribbing. Moreover, branch raises above the grizzly level must be closely timbered. Grizzly drifts are supported by masonry lining where there is danger of crushing. Undercutting drifts generally are not timbered when they are being driven, but when undercutting and caving begin some timber support sometimes is necessary to protect the miners while the undercutting rounds are being drilled and blasted.

Ore drawing is under the direction of the geological department and is done in accordance with typewritten orders issued by the chief geologist and approved by the mine superintendent. The block from which most of the ore has been drawn so far was calculated to contain 2,858,588 tons. By April 1928 about 2,534,600 tons had been extracted, with an assay value of 90.69 percent of the calculated grade of ore in place. At that time about 165,300 tons of ore were being drawn per month.

SUMMARY OF BLOCK CAVING

1. Applicability.—Block caving is suited to large-scale, nonselective mining of large deposits of low-grade ore, of which the strength and structure are such that when a block of ore is undercut and partly isolated from the surrounding blocks it will cave and break up finely enough to be drawn off from below through raises and chutes.

2. Flexibility.—Block caving is a nonselective method by which waste inclusions within the ore body are necessarily drawn with the ore, and the grade of ore mined can be controlled only to the extent possible by regulation of the amount of capping and marginal-grade wall material drawn with the ore.

3. Recovery.—The percentage of the total ore in the deposit that can be recovered varies with conditions and is influenced by the amount of dilution that the ore can stand. Thus, if the minimum grade which it is profitable to mine is ore containing 1.25 percent of copper, drawing must stop when, owing to dilution, the grade falls below this figure, regardless of whether or not all the ore has been drawn. Some dilution always occurs, and the tonnage drawn from any block usually is greater than the estimated tonnage, whereas the average grade of the ore as drawn usually is less than that estimated. If the margins of the deposit are very irregular in outline, either the extraction or the grade of ore recovered will suffer, as it is impossible to make the cave follow an irregular boundary closely.

At Ray the dilution in drawing more than 40,000,000 tons is estimated at 10 percent. At Inspiration 110 percent of the estimated tonnage and more than 85 percent of the estimated metallic contents are recovered. At Miami operations in 13 completed stopes resulted in the recovery of 2.4 percent more copper than was estimated, but

15.15 percent more rock was drawn than the estimated tonnage, the average grade of the total ore drawn being 88.93 percent of the calculated grade.

4. *Development.*—A large amount of preliminary development and stope preparation is required before extraction begins. Besides haulage-level development, preliminary work includes some or all (depending upon the particular system employed) of the following: Transfer, branch, and finger raises; grizzly-level drifts; grizzly installation; boundary cut-offs, consisting either of stopes or drifts and raises; draw sets; and undercutting stopes or undercutting drifts. This work requires considerable time for completion, during which the money expended is tied up. Table 18 shows the amount of preparatory work exclusive of main-level development per ton of ore at three mines.

5. *Cost of block caving.*—Mining costs are low and often compare favorably with costs at open-pit mines. Low costs are due primarily to the facts that most of the ore is broken by caving instead of by drilling and blasting, the ore usually is handled by gravity between the undercutting level and the haulage level, production per man is high, little timber is required, consumption of explosives per ton is low, and the operations are conducted on a large scale. Tables 19, 20, and 21 show various cost items for block caving.

TABLE 18.—*Preparatory work per ton of ore developed, block caving*

[Exclusive of main-level development]

Mine and kind of development	Development per ton of ore: Linear foot	Development per ton of ore: Square foot	Development per ton of ore: Ton	Ore per unit of development, tons
Ray:				
Supply drifts	0.00058			[1] 1,723
Grizzly drifts	.00260			[1] 385
Shrinkage stopes			[2] 0.020	[3] 31.5
Undercutting			.020	[3] 49.8
Total	.00318		.040	[1] 314 [4] 20.4
Miami:				
Grizzly-level drifts	.00131			[1] 763
Grizzly-level raises and chutes				[8] 13,917
Boundary drifts	.00438			[1] 229
Boundary corner raises	.00057			[1] 1,754
Finger raises	.00575			[1] 174
Undercutting drifts	.00126			[1] 866
Undercut mining		0.01484		[6] 22.3
Drilling and blasting boundary drifts	.00438			[1] 229
Drilling and blasting corner raises				[7] 69,439
Total	.01700	.01484		[1] 58.8
Humboldt:				
Supply drifts	.00108			[1] 926
Grizzly drifts	.00325			[1] 307
Grizzlies				[8] 8,571
Finger raises	.00691			[1] 145
Pilot raises	.00067			[1] 1,500
Shrinkage drifts	.00133			[1] 750
Shrinkage stopes		.075		[4] 13.3
Undercutting		.083		[4] 12.0
Total	.01324	.158		[1] 75.5 [4] 6.3

[1] Per foot.
[2] Tons of broken swell drawn only.
[3] Per ton.
[4] Per ton of swell.
[5] Per set.
[6] Per square foot.
[7] Per raise.
[8] Per grizzly.

TABLE 19.—*Direct stoping costs, block caving.*

Mine and year	Variation of block caving	Cost per ton of ore drawn						
		Labor and supervision	Compressed air, drills, and steel	Power	Explosives	Timber	Other supplies	Total
Ohio (copper):								
October 1911	(1)							$0.2197
April 1913	(1)							.2220
Ray, 1928:								
Preparatory excavation		$0.0581	$0.0010	$0.0014	$0.0070	$0.0330	$0.0022	.1030
Stoping		.0711	.0016	.0050	.0247	.0092	.0050	.1165
Drawing ore		.0940	.0001	.0001	.0190	.0128	.0006	.1267
Total	(6)	.2232	.0027	.0065	.0507	.0550	.0087	.3468
Inspiration, 1928	(4)	.138	.005		.017	.031	.002	.208
Miami, Oct. 1, 1925 to Sept. 30, 1929:								
Preparatory excavation above grizzly level.								.0299
Drawing, maintenance of extraction drifts, grizzlies, and raises.								.1303
Total	(3)							.1662
Humboldt, 1928:								
Stope preparation		.020			.009	.021		.050
Breaking and drawing		.085			.034	.010		.129
Undistributed		.040	.045					.085
Total	(5)							.273
Braden, 1928:								
Development		.0101	.0039		.0064	.0033	.0020	.0257
Undercutting		.0777	.0089		.0152	.0415	.0063	.1496
Hand mucking		.0393					.0027	.0420
Undistributed								.0020
Total	(7)	.1271	.0128		.0216	.0448	.0110	.2193

1 Undercut and caved on sublevels 30 feet apart vertically; caving to finger raises.
2 Includes development and equipment.
3 Cut-off and undercutting by shrinkage stopes; caving to grizzly raises.
4 Undercut by drifts and crosscuts or by connecting tops of finger raises; caving to finger raises.
5 Boundary cut-off drifts and corner raises; individual blocks or stopes 150 feet square; undercutting by drifts and crosscuts; caving to finger raises.
6 Partial cut-off by boundary shrinkage stopes; undercutting by checkerboard system of drifts and crosscuts; caving to belled-out finger raises.
7 Undercut by widening blasts over a series of parallel timbered drifts; ore drawn through chutes into cars in these drifts.

TABLE 20.—*Man-hour per ton stoping and drawing, undercut block caving*

Mine and year	Man-hour per ton		Tons per man-shift	
	Stoping and drawing only	Total underground	Stoping and drawing only	Total underground
Ohio, October 1911	0.127	0.469	63.0	17.0
Ray:				
1927	.0787	.749	101.6	10.7
1928	.0692	.661	116.0	12.1
Miami, Oct. 1, 1925 to Sept. 30, 1929	(7)	.296	(7)	27.0
Humboldt, 1928	.128	(7)	62.6	(7)
Braden:				
1927	(7)	.609	(7)	13.2
1928	(7)	.658	(7)	12.2

TABLE 21.—*Consumption of explosives and timber, block caving*

Mine and year	Consumption of explosives, pound per ton of ore [1]	Consumption of timber, board feet per ton of ore [1]
Ray:		
1927	0.2523	1.79
1928	.3163	1.83
Miami, Oct. 1, 1925 to Sept. 30, 1926	.2225	1.04
Humboldt, 1928	.1600	.25
Braden, 1928	.1660	1.08

[1] Total underground.

SUBLEVEL CAVING

Sublevel caving is employed for mining large or moderate-size bodies of medium-soft ores (occasionally hard ores) of low or medium grade under a capping which caves in blocks which, with the gob, will arch over and hang up long enough to permit removal of the underlying ore in slices 20 to 30 feet thick. Schaus [25] has summarized the conditions for successful sublevel caving as it is practiced on the Gogebic range, as follows:

> The determining conditions to which this method is applicable are: (*a*) A dipping and pitching irregular ore body which does not lend itself to top slicing; (*b*) a medium-soft ore which breaks fine yet stands well and is not free-caving; (*c*) a hard capping which caves in medium-sized blocks without much fines and which is easily controlled.

Sublevel caving is applicable to smaller deposits than block caving, also to large ore bodies, and can be employed where block caving would be impracticable because of the tendency of the broken ore to pack and reconsolidate. It may also be employed where top slicing would be hazardous because of the tendency of the gob to hang up temporarily.

EXAMPLES OF SUBLEVEL-CAVING PRACTICE

EUREKA-ASTEROID MINE, MICHIGAN [26]

The iron formation consists of alternate layers of cherty and slaty horizons lying upon a footwall of quartzite and overlaid by siliceous slates. The formation dips 55° to 75° and is intersected by a series of diorite dikes dipping normal to the bedding. Important ore bodies occur in the troughs formed by the intersection of the footwall or impervious slate bands with the dikes. There are many faults in the Eureka mine that displace the beds 100 feet or more, sometimes 600 feet. (See fig. 63, *A*.)

The ore is a soft hematite that requires timber support in all openings. A band of soft, red slate at the top of the footwall slabs off readily. The hanging wall or capping is usually cherty iron formation, and the contact between ore and capping is very irregular.

[25] Schaus, O. M., Mining Methods and Costs at the Montreal Mine, Montreal, Wis.: Inf. Circ. 6368, Bureau of Mines, 1930, 29 pp.

[26] Schaus, O. M., Method and Cost of Mining Hematite at the Eureka-Asteroid Mine on the Gogebic Range, Gogebic County, Mich.: Inf. Circ. 6348, Bureau of Mines, 1930, 15 pp.

Two types of ore bodies occur. One consists of triangular masses (in cross section) lying at the intersection of the dikes and the footwall, and the other consists of narrow blankets 5 to 20 feet wide lying on the footwall.

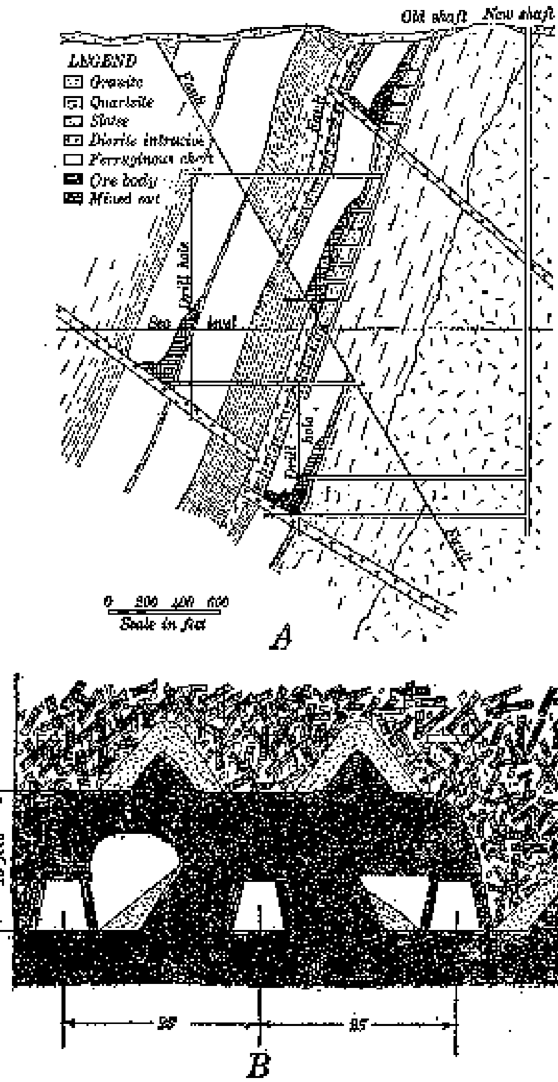


FIGURE 63.—Sublevel caving, Eureka-Asteroid mine, Michigan: *A*, Typical cross section of a Gogebic range mine; *B*, method of caving back on retreat.

Haulage levels are driven at 200-foot intervals, and sublevels are 18 to 25 feet apart vertically. Raises are put up from the haulage drifts to the top of the ore body and are spaced at intervals ranging from 200 feet in narrow ore to 50 feet in ore 100 or more feet in width. Raises have two compartments, one serving as a manway and the other as a chute. In wide ore, where raises are 50 feet apart, branch raises are put up midway between them. Ore bodies 5 to 50 feet wide are developed on the sublevels by longitudinal drifts and in the wider bodies by crosscuts or slice drifts spaced at 25-foot intervals and by one connecting drift over the line of raises.

In narrow ore, caving is begun midway between raises. If the ore is no wider than the drift the back lagging is removed between two sets of timber and a 6-foot cut drilled in the back. This is blasted, and the miners stand on the broken ore to drill the next cut, which is fanned out parallel to the drift. Before the second cut is blasted three or four back-lagging poles are replaced to hold back premature runs of gob. The second cut reaches the caved sublevel above, and the ore from both cuts is scraped to the chutes.

Where the ore is a little wider than the drift, which is usually driven on the footwall side, caving is begun on the hanging-wall side, where two long cuts are blasted, after which two cuts are blasted on the footwall side. Each cut is scraped to the raise after the blast. Ore over the back of the drift then becomes heavy and usually comes down with one blast. Runs of gob are held back by poles placed as side lagging.

Figure 10, page 16, shows the scheme of sublevel development for wide ore bodies. Figure 63, *B*, shows the method of caving back. The crosscut from the raise to the hanging wall is driven first and then caved back to within 8 feet of the raise; the scraper hoist is moved to that side of the raise and the crosscut (or "slice drift") driven to the footwall. This is caved back in the same manner, and the hoist is moved so that the 8-foot stubs and the back of ore over the connecting drift can be caved and scraped out. The miners then move to the next raise and continue slicing and caving back. Development is kept ahead of mining, so that a gang of miners on the sublevel below can be caving the hanging crosscut while the gang above is drawing the footwall crosscut. Miners receive $2.50 to $3 per foot for slice drifts and 60 to 90 cents per 60-cubic-foot car for caving (1929). Auger steel and hand-held machines are used for drilling and boxtype scrapers operated by 15-horsepower electric hoists for scraping the ore, except that where the scraping distance exceeds 100 feet 25-horsepower hoists are used.

Direct stoping costs during 1929 averaged as follows:

Direct stoping costs, Eureka-Asteroid mine, 1929

	Costs per long ton (2,240 pounds) of ore						
	Labor	Supervision	Compressed air, drills, and steel	Explosives	Timber	Other supplies	Total
Stoping	$0.240	$0.014	$0.067	$0.069		$0.002	$0.396
Timbering [1]	.190	.011	.001		$0.069	.006	.277
Total	.430	.025	.068	.069	.069	.008	.677

[1] Includes cost of all mine timbering.

Labor performance in stopes during 1929 was as follows:

	Man-hour per ton	Long tons per man-shift
Breaking (drilling and blasting)	0.194	41.23
Mucking (scraping)	.179	44.69
Timbering	.038	210.49
Other stope labor	.139	57.55
Total stoping labor	.550	14.54

The consumption of explosive in stopes was 0.382 pound of 40-percent gelatin dynamite per ton of ore, and the consumption of timber was 3.35 board feet per ton (round timber calculated to board feet).

MONTREAL MINE, WISCONSIN [27]

Figure 64, *A*, is an ideal cross section, showing the various iron formations and the mode of occurrence of the ore bodies. As a rule the outlines of the ore bodies are quite sharp. The intrusive dikes form, with the footwall, a trough that pitches eastward about 20°; these dikes are very regular and persistent. Cross faults shatter the formation and are always present near the ore bodies. There are two types of ore bodies: (1) The more important ores, which are irregular bodies lying in the pitching troughs, and (2) "chimney" bodies which usually pitch 80° to the west. All ore bodies are very irregular in outline. The ore is soft and generally drilled with auger steel, although ribs of hard, blue ore occur which require the use of mounted hammer drills. The ore is not free-caving, but it disintegrates rapidly. The enclosing rocks consist of bands of very hard, unaltered chert interspersed with bands of softer, leached material. The latter form planes of weakness parallel to the bedding. Cross-jointing planes cut the bedding almost at right angles at intervals of about 12 inches.

The lower levels of the mine are 165 feet apart measured on the dip. Wide ore bodies are developed by a footwall haulage drift, from which crosscuts are driven across the ore body at 300-foot intervals. From these crosscuts loading drifts are driven, as shown in figure 64, *B*. From the loading drifts double raises are put up to the capping on 34-foot centers. These are connected 17 feet below the top by 8-foot timber drifts; the ore body then is ready for mining. Slice drifts are driven to the foot and hanging walls, and the side and back pillars are caved back on the retreat, as shown in figure 64, *B*. The "stopes" on one side of the sublevel crosscut are large, and those on the other are small. The larger stopes take down the hogback above as follows: The side lagging is removed between two sets and a light cut blasted, care being taken to protect the pillar around the caps of the sets. The stope is then carried up, the roof being enlarged as much as possible without danger of caving before the proper height is attained. At a height of 12 feet a "doghole" is holed into the stope from the direction of retreat, and this is used as a means of entrance and exit during final drilling, which is done to fan out the opening under the hogback. (See fig. 64.) After

[27] Schaus, O. M., Mining Methods and Costs at the Montreal Mine, Montreal, Wis.: Inf. Circ. 6360, Bureau of Mines, 1930, 29 pp.

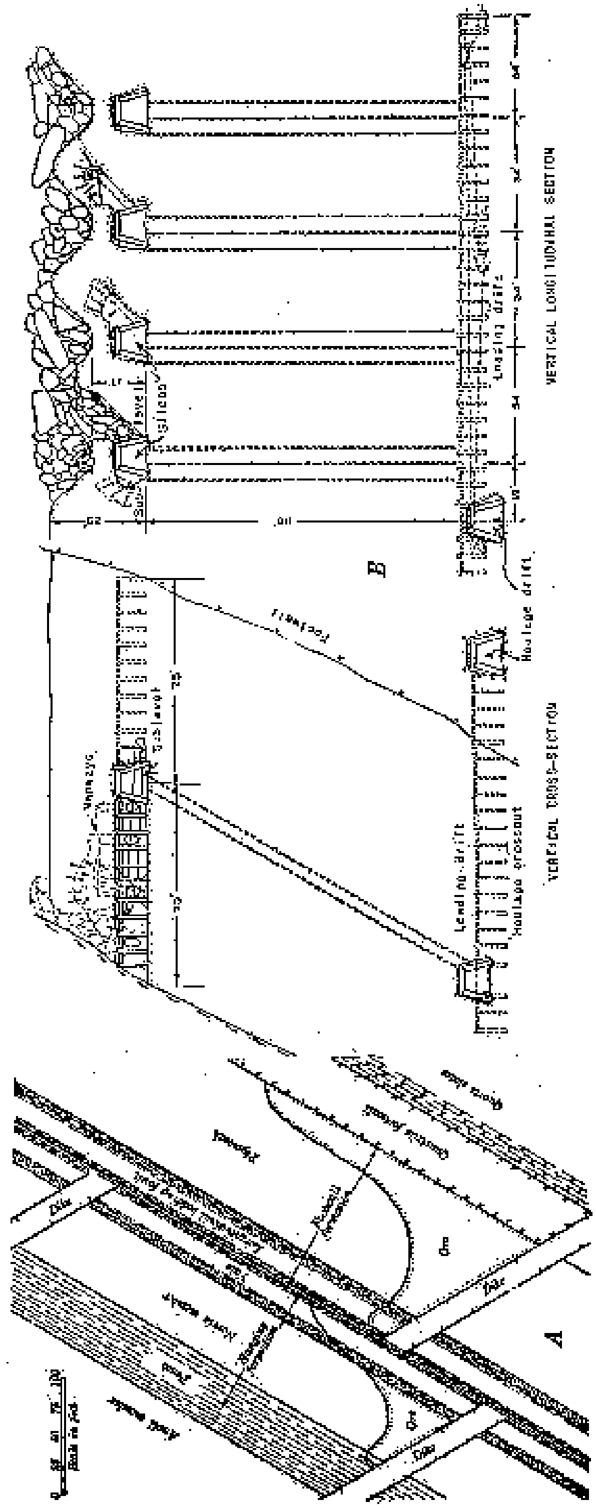


FIGURE 64.—Sublevel caving, Montreal mine, Wisconsin: A, ideal cross section of formations; B, method of developing, slicing, and caving.

the ore is drawn and the stope caved full blasting is continued in the cave between the chunks to shake out the ore mixed in during caving. The broken ore runs to the floor of the sublevel and is scraped to the raise by 42-inch hoe-type scrapers drawn by 15 horsepower slusher hoists.

Direct stoping costs during 1928 were as follows:

Direct stoping costs, Montreal mine, 1928

	Costs per long ton (2,240 pounds) of ore						
	Labor	Supervision	Air, drills, and steel	Explosives	Timber	Other supplies	Total
Stoping	$0.224	$0.036	$0.020	$0.061		$0.020	$0.360
Timbering [1]	.149			.007	$0.060	.008	.224
Total	.373	.036	.020	.068	.060	.028	.584

[1] Includes cost of all mine timbering.

Labor performance in stopes during 1928 was as follows:

Occupation:	*Man-hour per long ton*	*Long tons per man-shift*
Breaking (drilling and blasting)	0.176	45.45
Timbering	.074	108.08
Mucking (scraping)	.102	78.43
Supervision	.045	177.76
Total	.397	20.15

Consumption of explosives in stopes was 0.345 pound of 35-percent ammonia gelatin per ton of ore, and the consumption of timber in stoping, development, and maintenance was 2.76 board feet per ton (round timber calculated to board feet).

OLD DOMINION MINE, GLOBE, ARIZ.

At the Old Dominion mine a caving method was employed for mining both small and large bodies of low-grade ore.[28] As used in the smaller ore bodies this system is virtually a sublevel-caving system in which the slice drifts of the conventional system are replaced by inclined drifts and the sublevel interval is greater. Thus, instead of using scrapers to drag the broken and caved ore through the slices to the chutes the ore runs by gravity. This system is shown in figure 65, in which the slides correspond to the slices in the conventional sublevel-caving system.

The Old Dominion vein is 2 to 60 feet wide, and the dip ranges from 45° to 90°, averaging 60°. The ore bodies occur in the vein as localized shoots which have a flat rake to the west. The size and depth of the ore shoots vary widely, and each section of the mine presents a different mining problem. In some places the ground will stand well without support, but in others it is very heavy. The

[28] Shoemaker, A. H., Mining Methods at the Old Dominion Mine: Inf. Circ. 6237, Bureau of Mines, 1930, 2 pp.

Morenci slide method has been used for mining comparatively small ore bodies in the Old Dominion vein. The general method is to divide the 100-foot interval between levels into two lifts. Timbered pony drifts are run along the footwall. On the haulage level the pony drift is immediately over the footwall development drift, and on the intermediate level it is a small drift on the footwall. Trans-

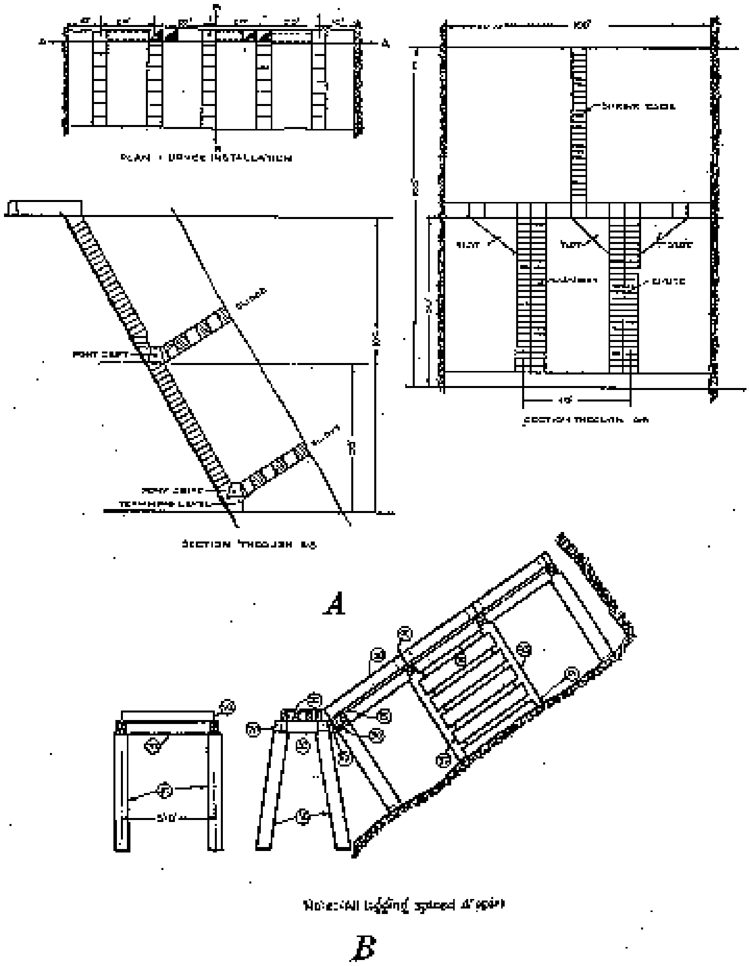


FIGURE 65.—Morenci slide system, Old Dominion mine, Arizona: A, Typical caving stope; B, timbering of pony drift and slide.

fer raises are run on 40-foot centers from the bottom pony drift to the intermediate drift for removing the ore from the upper lift which is mined first. From the pony drifts timbered slides are run on 20-foot centers to the hanging wall at an inclination of 36° from the horizontal. A shrinkage stope is then run on the footwall to the top of the lift, and after it is completed the remainder of the block is undercut and drawn.

In undercutting the first operation is to cut draw holes. These are untimbered raises beginning at the side of the incline at the draw sets and extending upward at an angle of 45° for about 9 feet. These raises meet similar raises from the adjoining inclines or slides to form an apex. The bottom of the apex between two draw raises is left higher than the top of the timber in the slides so that the pillar thus formed may bear part of the weight of the overlying broken ore. A cut of 4 feet or more is then taken out directly above and the length of the inclines. Controls for drawing ore are placed on both sides in every other set of timber.

The next step is to blast another round in the back of the draw holes, then the pillars between draw holes are drilled and blasted. From the hanging-wall side of the draw-hole raises holes are drilled vertically and fanned into the pillar toward the footwall. A pillar from apex to apex between slides is blasted down as a unit and must always be entirely cut to reduce the probability of a large section of unbroken ground settling on the slides.

The initial cut over the inclines is important. There must be enough room to accommodate the swell from the first slough of ore so that no direct weight will be thrown on the timbers. A draw hole from the next slide beyond the pillar being shot down should be completed before blasting to give a safe entrance from which to observe the effect of the blast. Often it is possible to bar down and drill the back a second time. After being undercut in this manner the back caves and the ore is drawn off. Where the stope is in an inclined vein drawing must start from the footwall side and be continued for some time to have the hanging-wall waste follow on top of the ore. More ore is drawn from the footwall draw points than from those nearer the hanging wall. As the highest-grade ore generally is on the footwall side it is important to recover the footwall ore as completely as possible.

A fairly large ore body having a maximum cross section of 350 by 200 feet and a thickness of 100 feet is being worked by this system (1930).

In 1928, 17.3 tons were mined by this system per man-shift stoping.

SUMMARY OF SUBLEVEL CAVING

1. *Applicability.*—Sublevel caving is applicable to the mining of low- and medium-grade ores in the form of large or moderately large bodies of considerable vertical height. It may be used in smaller deposits than block caving and is usually employed for mining medium-soft ores, although hard ores have also been mined successfully by this method. It can be used where block caving would be impracticable because of the tendency of the broken ore to pack and reconsolidate and where top slicing would be hazardous because of the tendency of the gob to hang up temporarily.

2. *Flexibility.*—Although sublevel caving lacks the flexibility of some other methods it is more flexible than block caving. Owing to the relatively thin backs caved and the development of the deposit on each of a series of sublevels which serve to explore and block out the ore as well as to prepare it for mining, irregular boundaries can be followed and mined closer than with block caving.

The larger inclusions of waste can often be left and caved with the gob, thus keeping the waste and ore separate. On the other hand, little opportunity is afforded for sorting out small bands of waste. Some degree of control can be exercised, although dilution with waste generally is unavoidable if a high extraction is desired.

3. *Recovery.*—Since the backs are caved under a gob consisting of a mixture of broken capping and slice timbers which cannot always be kept from breaking through and cutting off undrawn ore, some ore is bound to be lost. The recovery is not as complete as that with top slicing as a rule. It should be remembered, however, that it is hazardous to employ top slicing under a capping which is best suited to sublevel caving. In some iron mines where this method is used, the capping contains enough iron so that moderate dilution with cap rock is allowable, and under such conditions a high percentage of the total ore in the deposit can be extracted.

4. *Development.*—Considerable preliminary development is required before slicing and caving begin. Chute and manway raises must be put up to the top of the ore body, and the sublevels must be developed. However, all sublevels are not developed at once. Usually slicing and caving are progressing on one sublevel while the sublevel below is undergoing development. The number of long raises required may be reduced by using branched raises, which is the practice at some mines. Table 22 shows the amount of stope development required at three mines employing this method.

TABLE 22.—*Stope development, sublevel caving*

Mine no.	Sublevel interval, feet	Tons developed per foot of development[1]	Development per ton of ore,[1] linear feet
18	20	120	0.0083
19	20	100	.0100
20	18	58	.0172

[1] Long tons of 2,240 pounds.

5. *Cost of sublevel caving.*—The stoping cost per ton for this method is considerably higher than that for block caving. The difference in the cost of sublevel caving and top slicing is not pronounced at the few mines for which costs are given in this paper. Theoretically sublevel caving should cost considerably less than top slicing and probably would at a mine where either method could be employed interchangeably, if such a mine existed. In sublevel caving over half the ore is usually mined by caving, whereas in top slicing all is mined by drilling and blasting. In top slicing each slice is timbered up to the floor of the one above, whereas in sublevel caving less than half the height between sublevels is timbered, so that much less timber should be required per ton of ore. Less sublevel development is required in sublevel caving, as the interval between sublevels is greater. All these factors contribute to a lower cost for sublevel caving. In actual practice, however, conditions seldom are ideal, and the costs often are about the same for the two methods because they are employed under different conditions which

result in leveling the theoretical cost differential between them. Table 23 shows direct stoping costs at two mines employing sublevel caving. Table 24 gives direct stoping costs in terms of man-hours per ton at 6 mines and table 25 consumption of explosives and timber in sublevel caving at 4 mines.

TABLE 23.—*Direct stoping costs, sublevel caving*

Mine	Year	Direct cost per ton of ore mined[1]					
		Labor and supervision	Compressed air, drills, and steel	Explosives	Timber	Other supplies	Total
Eureka-Asteroid:							
Stoping	1929	$0.260	$0.067	$0.069		$0.002	$0.309
Timbering[2]	1929	.193	.011	.001	$0.008	.006	.270
Total[2]		.459	.078	.070	.008	.008	.477
Montreal:							
Stoping	1928	.259	.020	.061		.020	.360
Timbering[2]	1928	.149		.007	.060	.008	.224
Total[2]		.408	.020	.068	.060	.028	.584

[1] Long ton of 2,240 pounds.
[2] Includes cost of timber for development and stoping.

TABLE 24.—*Direct stoping cost in man-hour per ton, sublevel caving*

Mine	Man-hour per ton, stoping					Tons per man-shift
	Breaking (drilling and blasting)	Mucking (hand or slusher)	Timbering	Other stope labor	Total	
Eureka-Asteroid[1]	0.194	0.179	0.008	0.139	0.520	14.64
Montreal	.170	.103	.074	.045	.397	20.15
No. 16[1]					.636	12.58
No. 19[1]					.200	40.00
No. 20[1]					.365	21.92
Old Dominion[2]					.462	17.30

[1] Per long ton of 2,240 pounds.
[2] Per short ton of 2,000 pounds.

TABLE 25.—*Explosive and timber consumption, sublevel caving*

Mine	Explosive consumption, pound per ton of ore[1]	Timber consumption, board feet, per ton of ore[1]
Eureka-Asteroid	[2] 0.382	[3] 1.35
Montreal	[4] .504	[3] 2.76
No. 16	[4] .421	
No. 20	[4] .563	

[1] Long ton of 2,240 pounds.
[2] Stoping only.
[3] All mine timber; round timber calculated to board feet.
[4] Stoping and development combined.

TOP SLICING

Top slicing is similar to sublevel caving in that the ore body is mined by removing timbered slices, starting at the top and working progressively downward, and the overburden caves and settles as the ore is removed from beneath it. It differs from sublevel caving in that the sublevel interval is less, the ore is stoped by a series of timbered slice drifts (one alongside the other) which are mined and timbered up to the mat above without pillars being left between the slice drifts or over their backs between the slice and the mat, and only the capping is broken by caving.

Top slicing is employed in this country principally in the iron mines of the Lake Superior region. It is virtually the only underground method used on the Mesabi range and is also used on the Cuyuna, Vermilion, Marquette, and Menominee ranges. During 1929 shipments from the Lake Superior region aggregated 66,157,359 long tons, about 12,000,000 tons of which were mined by top slicing and 6,500,000 tons by sublevel caving.

Figure 66, *A*, shows the general scheme of top slicing employed on the Mesabi range before the introduction of scraper mucking in which the ore was shoveled from the slices into cars and trammed to the chutes. Figure 66, *B*, shows top slicing in a flat ore body, one slice thick. In both illustrations it will be noted that the slices are parallel and at right angles to the tramming drifts. This system may be termed "parallel slicing." Radial slicing has been adopted in some thick ore bodies where a series of chute raises can be put up from the haulage level without much work in waste rock. Figure 67 shows sublevel plans for radial slicing. Radial slicing was developed along with the general use of scraper mucking to facilitate scraping by eliminating sharp turns in the scraping lines. Inclined top slicing (fig. 68) has been used in a few mines in the West to eliminate mucking in the slices. In this system the ore runs by gravity from the face of the slice to the chute.

Top slicing finds its chief application in the mining of fairly wide to wide ore bodies too weak to stand without support except over short spans and having a weak capping which will cave readily when its support is removed and fill the excavation tightly. A capping that breaks in large blocks which arch over or hang up so as to leave open spaces in the gob is dangerous in top slicing. Hence, for successful top slicing, top weight or vertical pressure is a requisite. The side walls may be weak or strong. If the ore occurs in the form of a narrow, dipping ore body, a weak hanging wall is desirable because with a strong hanging wall open spaces may be left in the gob due to incomplete caving; later blocks may let go suddenly and drop through the spaces, throwing sudden, excessive weight on the gob and endangering the miners in the slices below.

As top slicing requires considerable timber and lagging, it is essential that the supply of timber available be plentiful and cheap. Slice timbers are required to stand only a short time and are then blasted down, therefore a somewhat poorer grade of timber is acceptable than would be required for more permanent support. However, the timber should be tough and capable of standing considerable crushing and bending before breaking; hence round, green timber with the bark on is usually preferred if it can be obtained at low cost.

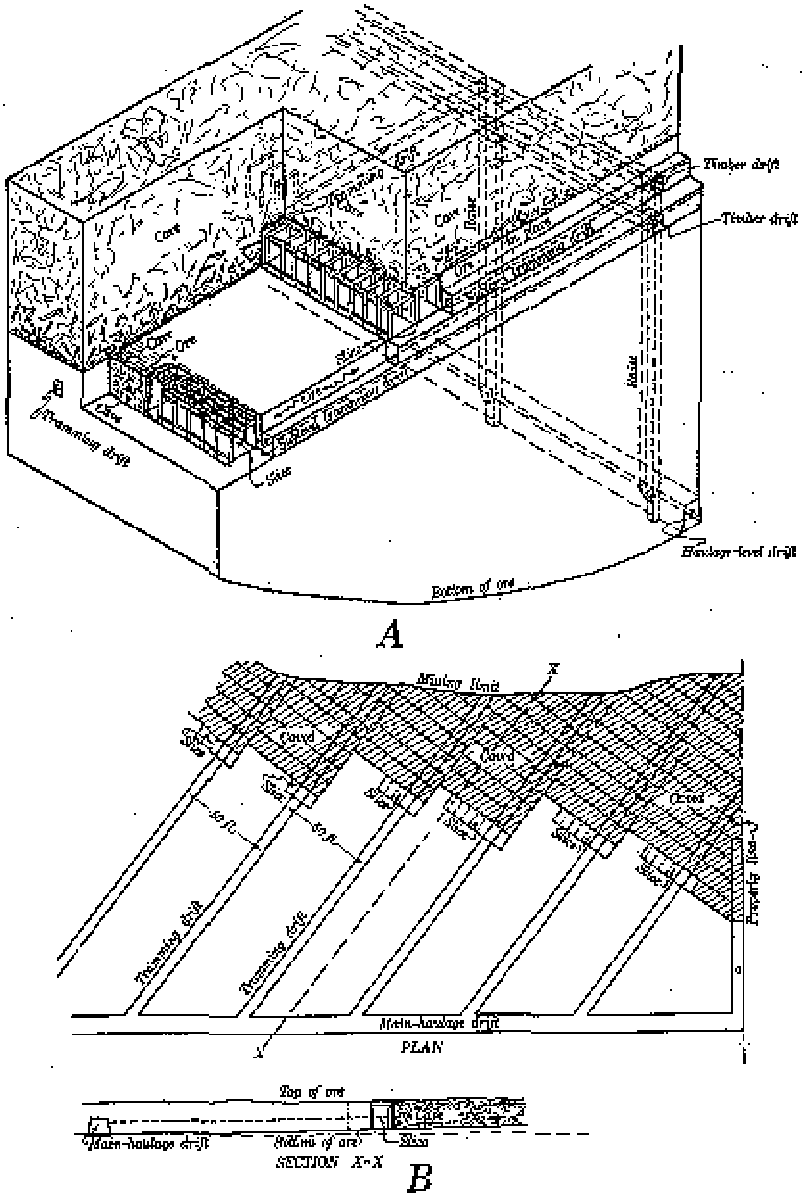


FIGURE 66.—*A*, General scheme of horizontal top slicing; *B*, top slicing in flat ore body, one slice thick.

Obviously top slicing can be employed only where caving of the overlying rocks and disturbance of the surface are permissible. As noted later the method is employed successfully under a strong capping which does not cave; here sand filling is dumped in beneath the capping to fill the space above the gob as mining proceeds downward step by step.

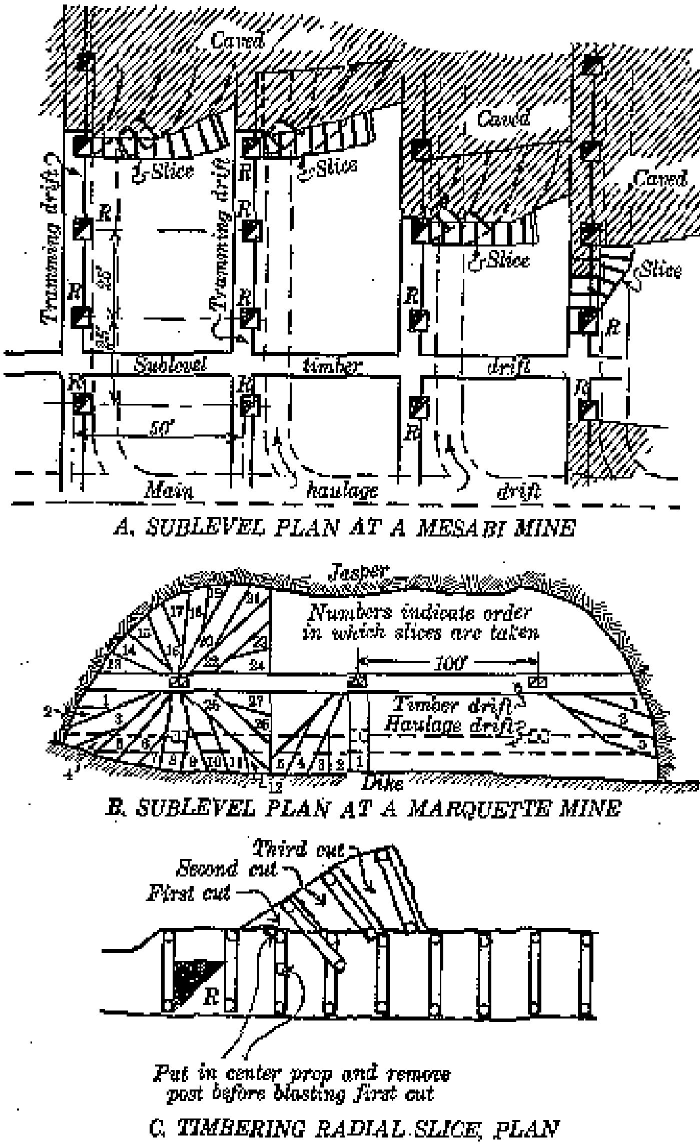


FIGURE 67.—Radial slicing; sublevel plans.

Top slicing can often be employed advantageously for extracting pillars between filled or caved stopes and for robbing operations in caved or badly broken ground, where it would be unsafe to stope from the bottom upward by overhand methods.

EXAMPLES OF TOP-SLICING PRACTICE

MESABI RANGE

After the use of more wasteful methods of stoping on this range during its early history and a realization of the value of Mesabi ores top slicing, which had been used successfully for recovering

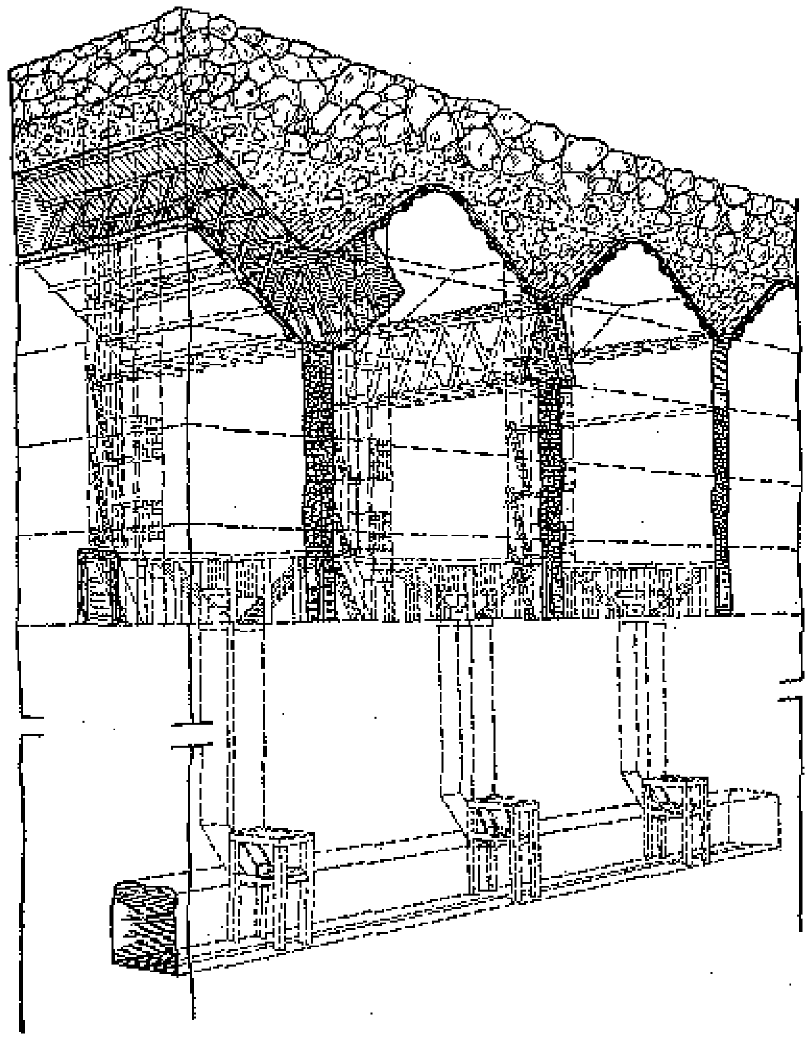


FIGURE 68.—Perspective sketch of Coronado incline top-slicing mining method.

pillars, was adopted as the standard method of underground mining. Mesabi ore bodies are adapted admirably to stoping by this method. Although the greater part of the output of this range comes from open-pit operations there are also many underground mines. The ore bodies usually lie at comparatively shallow depths and are in the form of relatively thin, broad, elongated, basin-shaped deposits. The boundaries are ordinarily quite sharp and well-defined, although gradations occur between ore and low-grade material within the ore bodies.

The wall rock is taconite, a hard, ferruginous chert. The region is covered by glacial drift from a few feet to 150 feet or more thick, and the immediate capping of the ore bodies may be glacial drift, paint rock, slate, lean ore, or taconite. The glacial drift or surface material ranges from fine sand, clay, or gravel to large boulders imbedded in the finer material. The taconite capping is hard, but unlike the underlying and wall taconites it is ordinarily partly leached and contains slips and joints which allow it to break and cave readily when undercut. Sometimes the ore must be mined out from beneath for a width of two or three slices before the taconite capping will break, but once a cave is begun the following slices cave readily. The ore itself consists of soft hematite and limonite. Top slicing has been employed successfully under a wet quicksand cover, although great care must be exercised. At one such operation with which the senior author was familiar it was customary to leave a few feet of ore against the sand for protection in mining the first slice. Hay was packed over the lagging, and before the slice was blasted down the floor was covered with a thick mat of hay. Owing to unevenness of the sand-ore contact a sand break occasionally occurred, filling drifts and raises for several hundred feet. Such occurrences were rare, however, and top slicing was carried on for some years without serious accident.

Three principal types of ore bodies are mined by top slicing on the Mesabi range:

1. Thin, flat-dipping ore bodies only one slice (up to 20 feet) thick, including thin margins or "shore lines" of thicker ore bodies. (See fig. 66, *B*, p. 261.)
2. Ore bodies two to four slices thick. (See fig. 11, p. 17.)
3. Thick ore bodies many slices thick, such as occur near Eveleth.

Development.—After the hoisting shaft and haulage-level development near the bottom of the ore body are completed preparation for stoping consists of putting up raises to the top of the ore and connecting them on the upper sublevel by a "timber drift." Slicing either may start from this drift at right angles for parallel slicing or at various angles for radial slicing, or a system of tramming drifts may be driven first. (See fig. 66, *B*, p. 261.) The first method is usually applicable to the thicker deposits where chute raises may be spaced so as to come within scraping distance of each slice face without much development and raising in rock. The second method is employed in the thin, flat deposits where the sublevel tramming drifts are driven in ore to avoid a large amount of dead work on the haulage level. Frequently one or several timber shafts are put down from the surface at convenient places for lowering timber and lagging directly to the sublevel on which it is to be used.

Parallel slicing.—Parallel top slicing usually is begun on any given sublevel at the property line or ore margin and retreats therefrom, longwall fashion. The slices ordinarily are driven at right angles to the tramming or timber drift; for this reason the system sometimes is called "right-angle slicing." Sublevel intervals in regular slicing usually range from 10 to 14 feet. Ore only 16 to 20 feet thick, however, often is taken out in a high slice from floor to capping in one operation. Each slice is started by putting up "opening" sets at right angles to the drift sets. The drifts usually are small, and as the opening sets go up to the capping or mat the back of the drift opposite the slice has to be taken down first to make room for them. From these opening sets the slice is advanced in one direction to the cave or to the limit of the slice. In some mines the practice is to slice only in one direction from the tramming drift and carry the slice through to the cave or to the next tramming drift. (See fig. 66, *B*.) In others the slices are driven in both directions, but only halfway through the pillar between tramming drifts, to meet the slices from adjacent drifts.

When hand loading was employed it was customary to drive two slices in width before blasting down, thus making one track-laying job serve for loading the ore from two slices. An 8-foot cap and 8-foot (wide) slice were generally used. Sometimes three slices could be opened up safely without their taking undue weight. Since the advent of power-scraper loading it has become more or less standard practice to drive slices 10 feet wide and to blast down each slice as it is completed. If a second slice is driven alongside an open slice much of the ore is blasted into the latter and is difficult to reach with the scraper, especially if the first slice has become heavy and center props have to be set under the caps.

The slices are usually advanced 5 feet each round, and slice timbers are set on 5-foot centers as the slice advances. The slice timbering consists of simple drift sets of round timber, with "joggles" cut on the ends of the caps. These sets are placed and blocked to the sides of the slice (timber on the caved side and ore on the opposite side) and are lagged over and blocked on top. Collar braces of light, round poles are placed between caps at the ends and between posts of adjacent sets. The lagging usually is of round poles or split, round timber termed "split-lagging."

After blasting and before the face of the slice is mucked out, two or more round poles are driven over the cap of the last standing set and under the cap of the next to last set, forepole fashion. These poles are crosslagged with short boards or pieces of lagging to protect the men working at the face and to catch up the overlying mat or capping.

Two miners usually drill, blast, scrape out, and timber one set of ground (5 feet) per shift.

Slice timber.—Round, green timber with the bark left on is employed for slice sets. Tamarack makes about the best timber now available. Considerable pine was used in early years, but it is now very scarce and cannot be obtained in any quantity. Some hemlock is used and makes good timber. Birch, which is plentiful, does not make desirable slice timber, although some is used.

The timber is purchased on specifications as to size of the "tops" or small ends of the timber. Specifications vary somewhat at the different mines, depending on the width and height of slices, pressure developed, and size and price of available timber. Sizes specified by several typical mines are as follows:

Slice-timber specifications used by several mines

Mine	Length of cap (width of slice), feet	Diameter of cap timber, inches	Height of standard posts, feet	Diameter of post timber, inches
1	9 or 10	10–12	Up to 18	7–9.
2	10	7–9	11–12	7–9.
3	8 to 12	8–10	10	8.

In the shallower mines where pressures are not heavy the posts are set with little if any batter. In deeper mines where the pressure is heavy and especially where considerable side pressure develops the posts are well-battered.

Radial slicing.—Radial slicing has come into use with scraper loading and is employed mostly in, and is best suited to, mines where the ore is several sublevels thick. For this reason it is not so common on the Mesabi range as in the other iron-ore districts. Radial-slicing practice varies somewhat at different mines but is the same in principle.

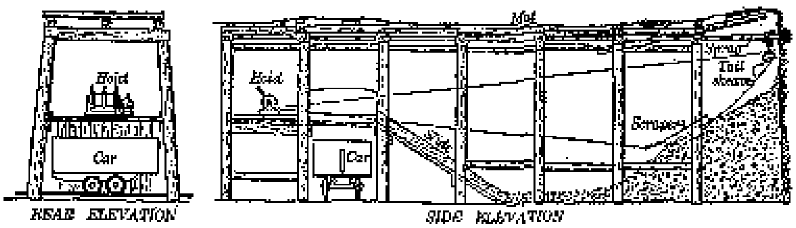


FIGURE 69.—Scraper loading from slice to cars.

Whereas in parallel slicing the ore is commonly loaded into cars (fig. 69) and then trammed to chutes, in radial slicing all ore is scraped directly into chutes without intermediate handling. The radial slice was, of course, designed to permit scraping the ore from a considerable area directly into chutes without dragging it around sharp turns. Right-angle turns and sharp bends can be negotiated with the scraper, generally by using deflecting sheaves for the ropes and rollers or boards nailed to the bottom of the corner posts to ease the scraper around them and prevent it from catching on the timber. Small scrapers (36-inch box-type) used at one of the mines visited are pulled around right-angle turns without rollers or boards and seem to work very successfully. Larger scrapers probably would not work so well. By close spacing of raises on the sublevels parallel slicing may also be used for direct scraping to chutes. Where this method is employed the tops of the chutes are generally 15 to 18 or even 35 feet apart on the sublevel; the closer spacing is obtained by driving branches from the main raises.

Figure 70 shows the general plan of workings at a Mesabi range mine where radial slicing is employed and the sublevel interval is 12 feet. Slicing is begun at the raises nearest the cave, property line, or mining limit. The first slice is generally driven along the cave if the cave comes to the side of the chute, and succeeding slices are fanned out therefrom. If the cave does not come to the raise a slice is started toward the cave and normal to the side of the raise. After the first slice is driven the second slice is begun by propping the cap of the first set beyond the opening set, removing the post on the side next to the radial slice, and blasting out a triangular cut. (See fig.

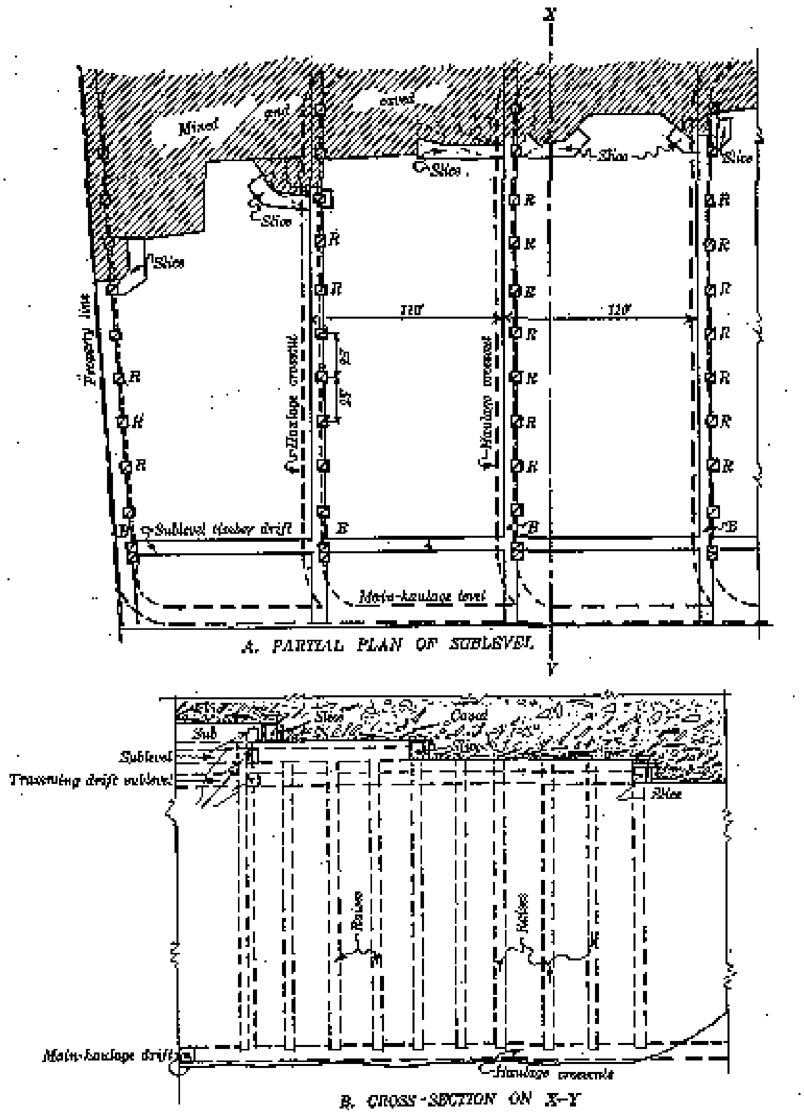


Figure 70.—Radial top slicing to chutes at a Mesabi range mine; thick ore body.

67, C.) A radial slice set is then placed as shown. The standard slice set at this mine has a 10-foot post and an 8-foot cap, but the cap of the first set usually is shorter, and when the end of the slice is reached 12- to 13-foot caps may be required. At other mines standard caps are used for all sets. In some mines the first radial slice is taken by removing a triangular cut (fig. 67, B) at the inner end of the first straight slice.

One objection to radial slicing is the difficulty of standardizing the cuts and timbering. While it may not be much more difficult to place timbers in radial than in parallel slicing, it is not feasible to standardize operations, timber, and slicing cycle, as in parallel slicing.

Handling and loading ore.—Today virtually all the ore from both development and slicing is handled by power scrapers which either load into cars or scrape directly into chutes. With cars a short slide is constructed, and the loaded scraper is pulled up the slide and discharges the ore into a car underneath. (See fig. 69.) For scraping into chutes no special preparation is required, except to provide a safe station for setting up the scraper hoist.

The earliest scraper equipment consisted of the ordinary two-handled surface scraper or "slusher", drawn by a small, single-drum air tugger. The scraper was pulled back to the face by hand and guided into the muck pile by the man at the face, the tugger pulling it into the dirt and out to the car or raise. This was an improvement on hand shoveling but was inefficient for distances greater than 25 or 30 feet. The small slusher is still employed in some mines for clean-up work.

Present scraping equipment consists of 36- to 48-inch (or larger) box- or hoe-type scrapers operated by 6½- to 25-horsepower double-drum electric hoists. The trend has been toward larger equipment, and 42- to 48-inch scrapers and 10- to 15-horsepower hoists are standard for new equipment. At many mines the operators are now buying nothing smaller than 15-horsepower hoists. For top-slice mining the economical limit in size probably has been reached, as enough ore per shift is not handled in each place to warrant larger equipment. Furthermore, larger equipment would be hard to move through raises, handle, and install, and larger scrapers would not pass between timbers as easily.

In top slicing the miners work in pairs on contract. Normally, two shifts are worked, there being four miners on a contract. Each contract is usually assigned the slicing and drifting tributary to one chute and may have 1, 2, or occasionally 3 faces in which to work. The contractors do their own timbering, drilling, blasting, scraping, and tramming at a set price per car of ore. Each contract is furnished with all equipment, including scraper, hoist, rope, sheaves, drills, and steel.

A tail sheave for pulling the empty scraper to the breast is secured at the face, usually to a boom, which permits cleaning the face out well after each blast. Often the boom consists merely of a wooden sprag or pole with one end against the last cap and the other headed in a hitch cut in the face near the top of the slice. Another type of boom is the telescoping boom of heavy pipe. A third is of extra-heavy pipe, suspended under the last cap with a chain and having two chains on the rear end to secure the boom in any desired swing position.

The trend toward larger scraping equipment has resulted in longer scraping distances with less development work required. At most mines visited longer slices either were contemplated for new levels, or it was stated that if development were to be done over again plans would be made for longer slices. With the present scraping equipment the length of slice is governed to a considerable extent by the length that will stand up well, which depends in turn upon the character of the ground and the gob. It also may be governed by the number of working places needed to furnish the required output rather than by the capacity of the equipment.

At one mine the main and sublevel drifts are 50 feet apart, and the raises are 25 feet apart along these drifts. On the new levels it is planned to drive the tramming drifts on 110-foot centers which will give 55 feet of scraping length on each side. At another mine where the tramming drifts were 50 feet apart and the slices driven on one side only this short spacing was necessary to provide enough working places. The operators of a single-sublevel mine where the drifts were 75 feet apart stated that if they were to develop the mine over again they would make the interval 100 feet or more.

Blasting down.—When a slice has been completed the side next to solid ore is usually boarded up or covered with wire fencing, and the bottom is covered with mining boards, lagging, or lagging and wire fencing. On the Mesabi range there is often a good deal of sand in the gob; where this is the case it may be necessary to cover the bottom and sometimes the side of the slices with boards to prevent contamination of the ore. Some mines use wire fencing on the side and boards on the bottom of the slices; others use wire on both the side and bottom. When fencing is used on the bottom poles are laid along the slice first, and the wire is secured on top with fence staples. The poles are generally 12 to 24 inches apart and long enough to span 2 or 3 sets.

In earlier years it was standard practice to cover both the bottom and the side with 1-inch pine "mining boards", which are inferior to the poorest commercial grades. Today mining boards are mostly of hardwood, 3/8 to 1/2 inch thick.

The wire fencing is a low-priced, galvanized-wire, diamond-mesh "station-ground" fencing put up in 10-rod rolls 42 inches wide and costing $12.50 to $13.50 per 1,000 square feet (1929). The cost of material per square foot covered is about the same for boards as for wire fencing, but the cost of handling and placing the fencing is much less. At one mine the cost of boards was about 4 cents per ton of ore mined, whereas the cost of fencing was only 0.7 cent per ton of ore, the difference being largely in the cost of handling and installation. In some deeper mines on the other ranges, where a thick timber mat has accumulated, it is unnecessary to cover the side of the slice since, when it is blasted down, virtually all the gob is wood.

After the slice has been covered the entrance is boarded over, the posts next to the cave side are drilled with a wood auger, and portions of sticks of powder with fuze and caps are inserted in the holes and the timbers blasted. Usually work can be started in the next slice alongside the blasted slice the next day.

Slicing on the sublevel below may begin after a considerable area above has been mined and caved, and the cave has become quiet and settled. Thus slicing may be in progress on several sublevels at the same time provided on any given sublevel it is kept well ahead of that on the sublevel below so that it is always under a well-settled mat.

Drilling and blasting.—The Mesabi ores are soft hematite and limonite and are drilled with auger steel, using light hammer drills equipped with special rotation mechanism. Hand augers are still used for drilling holes in treacherous ground where vibration of the hammer drills might loosen and jar down a slab of ore. The soft ores drill easily and break fine. It is usually unnecessary to drill regular cut holes in the slices, and all holes except the lifters are drilled at an angle above the horizontal.

The ground is broken with 35- or 40-percent low-freezing, gelatin dynamite, and about 0.4 pound of explosive is used per ton of ore on the average. For rock development 50- or 60-percent-strength explosive is commonly employed.

In top slicing care must be taken to avoid blasting out the timbers while the slices are being advanced. In hard, tough ground the difficulty obviously is greater than in soft, easily broken ore. Before the face is blasted collar braces are placed between the timbers of the last two standing sets and braces between the bottoms of the posts. Usually this prevents the timber from being knocked down, but it may be broken if the side and back holes are loaded too heavily or if the first holes do not break well.

CUYUNA RANGE

On the Cuyuna range in Minnesota the ore bodies occur as lenses in intensely folded slates, schists, and cherty formations. Most of the ore is fairly soft, but hard ores also occur. Here, as on the Mesabi range, some of the ore bodies are wide, lie under shallow cover, and are minable by open-pit methods. The district as a whole is covered by a thick overburden of glacial drift which is water-bearing.

Narrow lenses of ore dipping at rather steep angles are common, and these are mined by underground methods, both top slicing and sublevel caving being employed. At one mine visited the ore ranges in thickness from a few feet to a maximum of 50 or 75 feet; the immediate hanging wall is a soft paint rock which is heavy and sloughs readily, especially when wet, as it often is. The dip ranges from 55° to 75°, averaging about 65°. The ore does not occur uniformly along the strike but consists of several lenses or bulges more or less connected by narrow seams of ore.

Main levels are driven about 150 feet apart vertically and consist of a short crosscut from a vertical shaft and a single longitudinal drift along the footwall. (See fig. 71.) Two-compartment cribbed raises are put up along the footwall at about 50-foot intervals, although this varies, depending upon the location and lengths of the ore lenses.

Timber drifts connect these raises along the footwall and are about 80 feet apart on the dip. Connecting drifts are not driven on the sublevel between these timber-tramming levels, but tuggers are employed to hoist timber to them from the timber drifts. Timber and men are handled through a timber shaft some distance from the ore-hoisting shaft.

The radial top-slicing system is employed at this mine, the sublevel interval being 11 feet. Radial slicing is begun from the raise on any given sublevel, and the first slice is usually driven along the footwall. Succeeding slices are swung around from the footwall on one side of the raise to the footwall on the other side, although this practice sometimes is varied to meet local conditions. As the heavy hanging wall is approached the slices are swung to meet it at an angle of 90° (the last caps being parallel to the strike). Contamination of the ore by hanging-wall paint rock is carefully avoided.

A full-width slice is 12 feet, but the slice narrows where it branches off from a preceding slice or from the raise. Slices are as long as 100 feet but usually are less, the length depending upon the spacing of the raises and width of the ore.

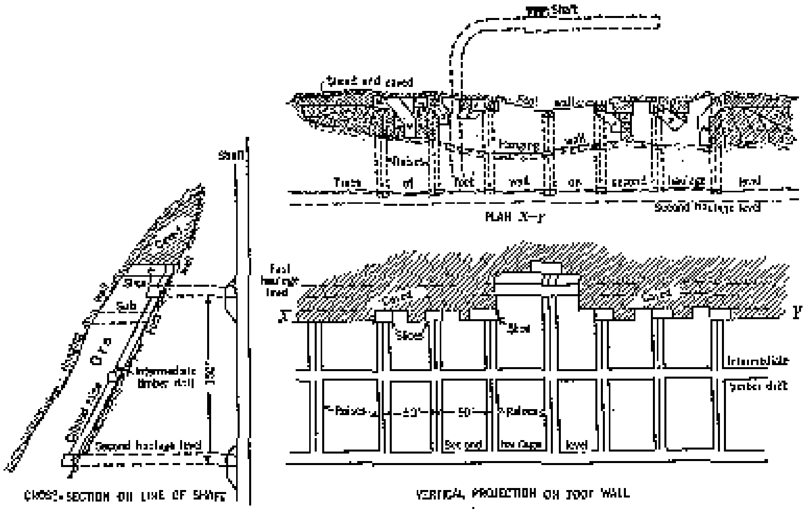


FIGURE 71.—Radial top slicing at a Cuyuna range mine.

After a mat has been started no boards or wire fencing are used on the sides of the slices, but the bottom of each slice is carefully covered with poles across which ½-inch hardwood mining boards are laid. No wire fencing is used at this property.

Specifications for timber are as follows:

	Top size, inches		Top size, inches
Posts	9–12	Lagging	5–7
Caps	12–15	Poles	3–5

Scraping is done with a box-type scraper 34 inches wide having rounded sides and a 200-pound lead weight on the back.

VERMILION RANGE

At Ely, Minn., the ore occurs in a deep trough between greenstone walls and under a capping of jasper. (See fig. 72, A, p. 272.) The immediate wall rock is usually a thin layer of paint rock. The ore bodies are irregular in outline and range up to 300 or 400 feet in width. The ore varies in character from soft hematite to hard, specular hematite and is high in iron content.

In the earlier years a chute-caving method of mining was employed, and later top slicing was introduced to obtain higher recovery and less dilution. More recently some of the mines have used sublevel caving for mining part or all of the ore.

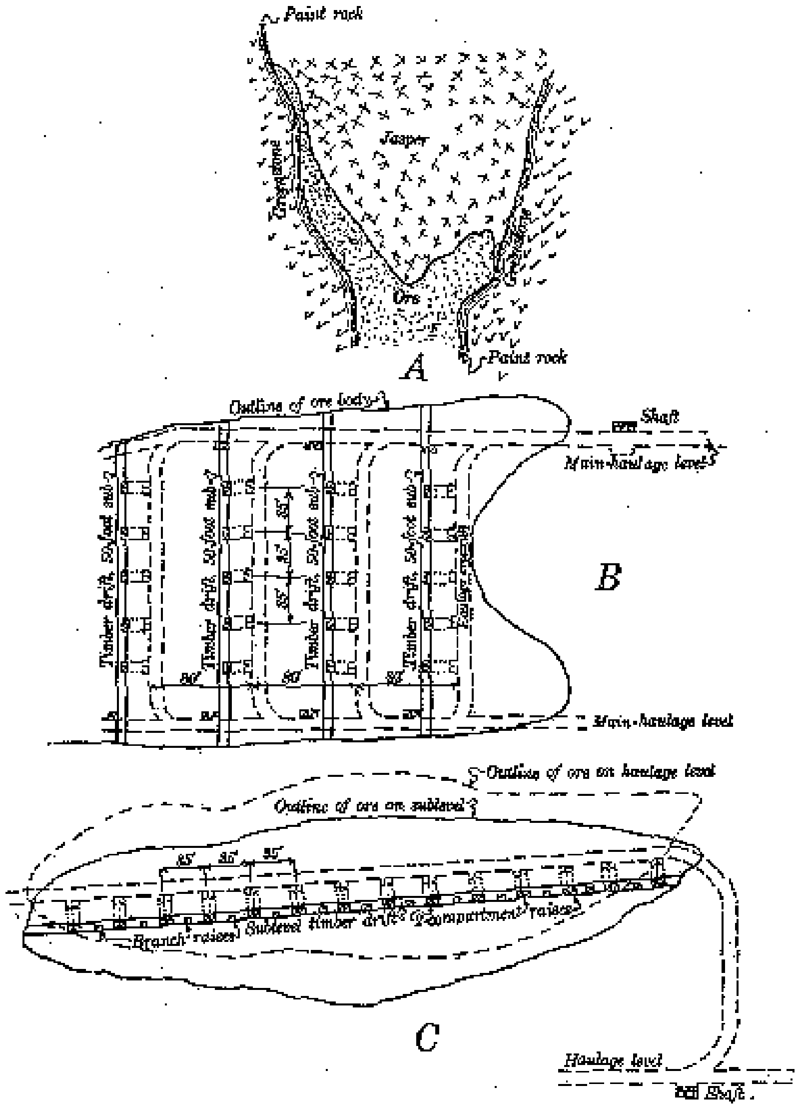


FIGURE 72.—*A*, Typical cross section, Vermilion range ore body; *B*, plan showing scheme of main and 50-foot subdevelopment in wide ore for radial slicing; *C*, Vermilion range development plan in narrow ore for parallel top slicing.

Slicing and sublevel caving are now being carried on at considerable depth below the surface and often under a mat accumulated from several hundred feet of top slicing on the upper levels. The mines are developed by shafts from which main haulage levels are driven at 100-foot vertical intervals. In the widest ore bodies the main levels are sometimes laid out with a loop-haulage system having crossdrifts 50 to 80 feet apart.

At one mine a new level is being developed with crossdrifts 80 feet apart center to center. (See fig. 72, *B*, p. 272.) At this mine raises are put up from the crossdrifts on 35-foot centers. Radial top slicing is employed with a sublevel interval or slice height of 12½ feet. A timber drift is driven on the sublevel 50 feet above the haulage level (halfway between the main haulage levels), but no timber drifts are driven on the other sublevels, all timber being hoisted up to them with tugger hoists.

The mat consists of timber accumulated from 400 feet to top slicing on the upper levels, and although very heavy it serves to prevent contamination of the ore with rock. The timber specifications call for 8- to 10-inch round timber for both posts and caps. Caps are 8 feet long. The posts, which are 10 feet long, are battered to give a slice width of 10 feet at the bottom.

Before a slice is blasted down round poles are placed on the bottom, and wire fencing is laid on and secured to the poles with fence staples. As virtually all the mat is timber with little admixture of capping the side of the slice does not have to be covered at all. Often when a slice is driven along the cave the mat looks like a solid pile of cordwood, it is so clean and free from waste.

The broken ore is scraped from the slice directly into the chutes by means of 48-inch, hoe-type scrapers weighing about 750 pounds and operated by double-drum, 15-horsepower, electric tugger hoists. The scrapers are constructed with teeth on one side and a manganese-steel edge on the other. There are eight manganese-steel teeth. In spite of the thick accumulation of mat at depths of about 1,200 to 1,400 feet good ventilation is maintained, and very warm places are rare.

Another mine, where the ore trough dips about 70°, is developed by a vertical shaft in the footwall from which haulage levels are driven at 100-foot intervals. The deposit averages 100 to 125 feet in width and is as much as 300 feet wide in one section. The ore ranges from soft to very hard, specular hematite. The main level is developed as shown in fig. 72, *C*. Raises are put up on 35-foot centers at an inclination of 70°. Parallel top slicing is used in some parts of the mine and sublevel caving in others. The sublevel interval is 12⅓ feet for top slicing and 20 feet for sublevel caving. The line of raises is connected with a timber drift on each sublevel. Before the slices are driven to the foot and hanging wall on each sublevel, inclined branch raises are driven from a point 20 feet below to split the distance between the main raises on the slicing level. This permits scraping directly to the chutes from each slice without resorting to radial slicing or scraping around sharp turns. Slices are 10 feet wide, and 10-foot caps and posts are used. There is a thick accumulation of timber in the mat. Sets are 5 or 6 to 7 feet apart, depending upon the ground, and the posts of the sets are battered. Slices average 60 feet in length and are longer in wide ore. Hoe-type scrapers 48 inches wide, operated by 15-hp. electric, double-drum hoists are used to scrape the ore into the chute raises.

Each 10-foot slice is blasted down separately in top slicing, and because of the good timber mat it is unnecessary to cover the sides of the slices before blasting down. A floor of 2-inch plank is laid on the bottom of the slice, as it is advanced, to form a smooth surface for the scraper, and this is left in when blasting down.

Heavy drifters and hollow steel with crossbits are employed for drilling in hard ore, whereas light hammer drills and auger steel are used in soft ores.

MARQUETTE RANGE

On the Marquette range, Michigan, the soft ores are usually mined by top slicing but sometimes by sublevel caving. The ore bodies are enriched portions of iron formation in minor folds or troughs superimposed on a wide, westward-pitching synclinorium or at the intersection of diorite dikes. The iron formation consists of ferruginous cherts and slates intruded by diorite dikes and sills. The footwall formation usually is the Siamo slate, which contains some cherty, ferruginous phases. The footwall ordinarily is quite regular, but irregular lenses and chimneys of ore often extend upward into the overlying iron formation, and the upper boundaries of the ore bodies are not sharp. The overlying beds of chert and jasper are hard but friable and, when caved, break into a medium-size rubble. The rocks of the area are overlain by thick glacial drift in most places. Present mining operations are at depths ranging from less than 1,000 feet to about 2,500 feet, so that the pressures encountered in top slicing often are considerable.

Individual ore bodies range in size from narrow lenses to bodies 800 feet wide and 150 feet or more thick.

Mine No. 2, Marquette range.[29]—The ore is a soft, hydrated hematite, of which some is very high grade, but some contains considerable siliceous material. The high-grade ore generally requires timber to support all but the smaller openings and is mined by top slicing; it can be drilled with auger drills. Dikes which occur in the ore bodies are badly decomposed and do not stand well when exposed to the air; they must be sorted carefully from the ore as it is mined.

Haulage levels are 150, 200, or 250 feet apart, and in 1929 the bottom level was at a depth of 1,650 feet. Cribbed raises 50 to 100 feet apart are put up from the haulage level at inclinations of 65° to 75°. They are put up to the top of the ore body and 10 to 15 feet below the top are connected by a sublevel drift. Slice drifts 10 feet high are mined radially from the top of each raise. (See fig. 67, *B*.) A slice is first taken along one side of the drift, followed by succeeding slices as shown. These are timbered with 8-foot round timbers and are lagged carefully as they are driven. Three and sometimes four slices are taken before the timbers show that they are taking weight. When this occurs the floor in each slice is covered with 3½-inch round poles 9½ feet long laid alongside each other and overlapping 2 feet on the ends, and the timbers are blasted down. Slicing is continued until all the ore on the sublevel tributary to a given raise has been sliced and caved, then the miners cut a drift from the raise 10 feet lower down and start

[29] Eaton, Lucien, Mining Soft Hematite at Mine No. 2 of the Marquette Range, Michigan: Inf. Circ. 6170, Bureau of Mines, 1929, 15 pp.

slicing up to the mat on the first sublevel in the same manner as before. The ore is drilled with auger drills of the jackhammer type, using twisted auger steel of diamond section 1½ inches by ⅞ inch on the center lines. Three rows of holes are required, three in the top row and four in each of the others; 60-percent gelatin dynamite is used in blasting. The ore is dragged to the chutes with 42-inch, hoe-type scrapers weighing 325 pounds. Each working place is supplied with two to four long extension cords with 50-watt lamps attached.

In 1928, 240,000 tons were mined by top slicing at this property. Direct stoping costs were as follows:

Direct stoping costs, Mine No. 3, Marquette range, 1928

	Costs per long ton of ore hoisted		Costs per long ton of ore hoisted
Labor	$0.370	Explosives	$0.075
Supervision	.025	Timber	.070
Compressed air, drills, and steel	.045	Other supplies	.075
Power	.047	Total	.707

Mine No. 5, Marquette range.[30]—The ore body lies in a large syncline dipping about 15° to the west; it is quite regular, being about 100 feet thick in the middle of the syncline and thinning out on both sides, and ranges from 400 to 800 feet in width, averaging 600 feet. The footwall is uniform in strike and dip, but irregular lenses of ore often extend 50 to 75 feet up into the overlying jasper. The ore is high-grade soft hematite, is easily drilled, and breaks into a fine aggregate which is easily handled by scrapers. Some hard seams of blue ore occur near the hanging wall which are easily drilled but break in large chunks. The overlying jasper breaks into coarse rubble when it is caved.

Parallel, main-level crosscuts are driven across the ore from foot to hanging wall, and from these two-compartment cribbed raises are put up on 35-foot centers and at an inclination of 65° to 70°. This arrangement provides for sublevel panels 35 by 75 feet on each side of each raise. These panels are mined by one straight slice drift and two side slices. (See fig. 78, p. 276.) Only one set-up of the scraper hoist is required for taking the three slices on each side of the raise. Sometimes it is necessary to retimber around the top of a raise after one side has been sliced out and before a slice is started on the opposite side, but as a rule this is not required. The three-slice panels are of such a size that all the ore can be mined without undue pressure being developed before the ore near the raises is extracted. On each sublevel the lines of raises are connected by drifts for handling timber, for ventilation, and for safety. These drifts are timbered with sets on 5-foot centers, using 8-foot posts and caps. A sublevel interval of 10 to 11 feet was used formerly, but lately (1930) this has been increased to 12 or 13 feet.

After the breast of a slice has been blasted the back is carefully trimmed to the mat above, and at least three fore poles 5 or 6 inches in diameter are put in and covered with lagging to catch up the

[30] Graff, W. W., Mining Practices, Methods, and Costs at Mine No. 5 of the Marquette Range, Michigan: Inf. Circ. 6380, Bureau of Mines, 1930, 10 pp.

mat and protect the miners. The ore is then scraped to the raise, after which the next set of timber is installed. Poles 9½ feet long and 8 to 6 inches in diameter are laid along the drift and spiked to three crosspieces or sills. Under the hanging wall, or near dikes

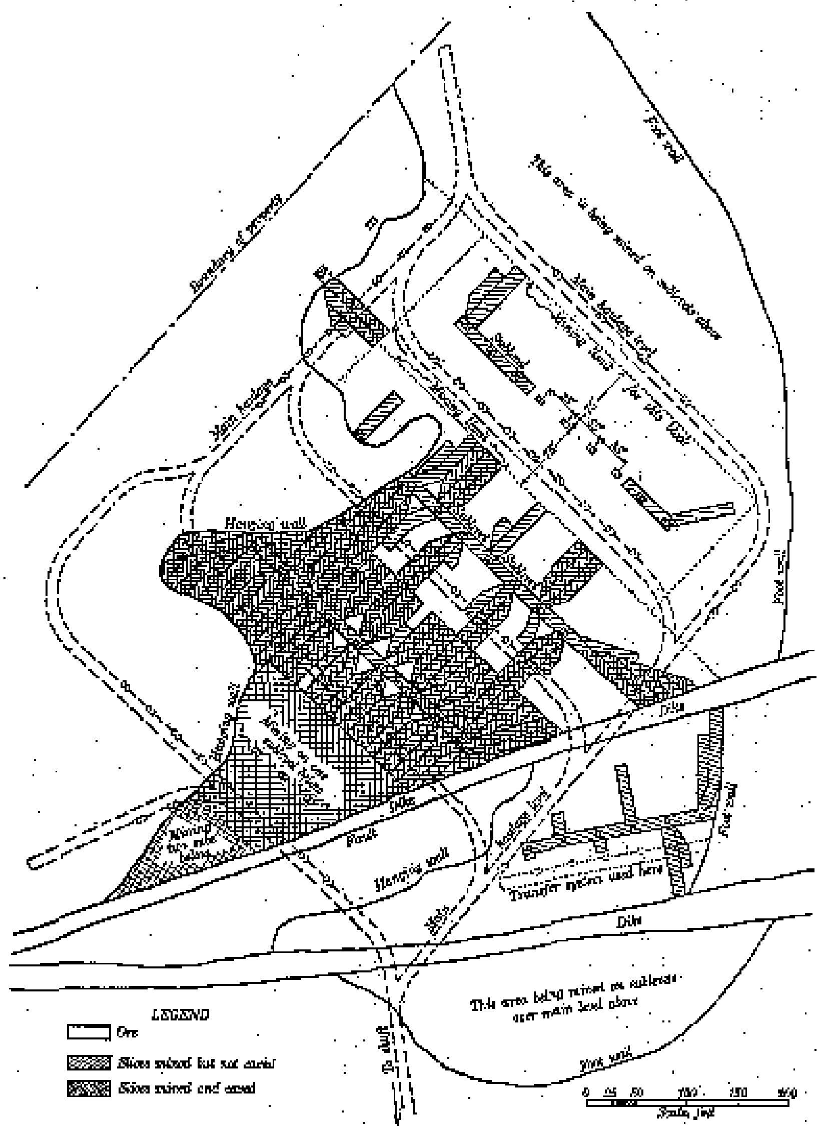


FIGURE 73.—Plan of typical sublevel, Marquette range, showing main-level development and arrangement of slices in mining blocks.

or horses of jasper, these poles are placed close together and covered with cross lagging to protect the sublevel below from runs of rock. Where there is a good mat overhead the poles are 6 to 8 inches apart. The poles form a good floor for the scrapers to ride on and an excellent cover to work under on the sublevel below. Wire netting is spread over the poles in new territory, particularly under new hanging wall, to make a more secure mat.

Electric double-drum scraper hoists are used for handling 36-inch box-type scrapers in the slices.

Direct stoping costs during 1929 were as follows:

Direct stoping costs, Mine No. 5, Marquette range, 1929

	Cost per long ton of ore hoisted		Cost per long ton of ore hoisted
Labor	$0.421	Explosives	$0.055
Supervision	.018	Timber	.079
Compressed air, drills, and steel	.095	Other supplies	.063
Power	.008	Total	.739

Labor performance in stoping was as follows:

	Man-hour per long ton	Long tons per man-shift
Breaking (drilling and blasting)	0.093	86.02
Timbering	.276	28.98
Mucking (scraping)	.164	48.78
Supervision	.029	292.40
General	.095	84.20
Total	.657	12.18

Consumption of explosives in stopes was 0.372 pound of 50- and 60-percent low-freezing gelatin dynamite per long ton of ore mined. Consumption of timber was 0.67 linear foot per ton of ore.

Athens mine.—At the Athens mine a top-slicing system is used which is the reverse of the usual one in that mining is begun at a block in the lowest section of the mine and advances in successively higher blocks up the pitch of a flat-pitching ore body. The following is an abstract of a paper in which S. R. Elliott[31] describes the system:

> The Athens system is based on the principle that in a mine under deep rock cover, actual caving to the surface often takes years to accomplish. If slicing is started in the upper section of the mine and the surface is covered by a heavy mantle of sand and gravel saturated with water, this water and sand will start to find its way into the mine as soon as the rock capping breaks, which will be much earlier than if slicing were started in the deepest section of the mine. Then, as mining progresses downward, the burden of pumping will steadily increase, as will the weight to be controlled. To overcome this objection the Athens system was devised. (See fig. 74.) The system is practicable only in a comparatively flat deposit.

According to Elliott the principal argument against the system is the interest on the original investment. He expected this disadvantage to be offset as follows:

(1) Prohibitive pressures will be avoided, thus making a large saving in timber cost.

(2) The ore body will be drained, and no surface water will pass through the working places because any caves to the surface will be below the active workings. By draining the ore and providing dry working places efficiency will be increased and the cost lowered.

[31] Elliott, S. R., Athens System of Mining: Trans. Am. Inst. Min. and Met. Eng., vol. 66, 1921, pp. 220–223.

The saving in freight by shipping dry ore will be considerable over a period of years.

(3) Pumping will be less than in the usual system, because it will take years to break the surface. After caving to the surface occurs and the size of the hole increases, draining a larger area, the pumping head will be decreasing as successive levels are abandoned.

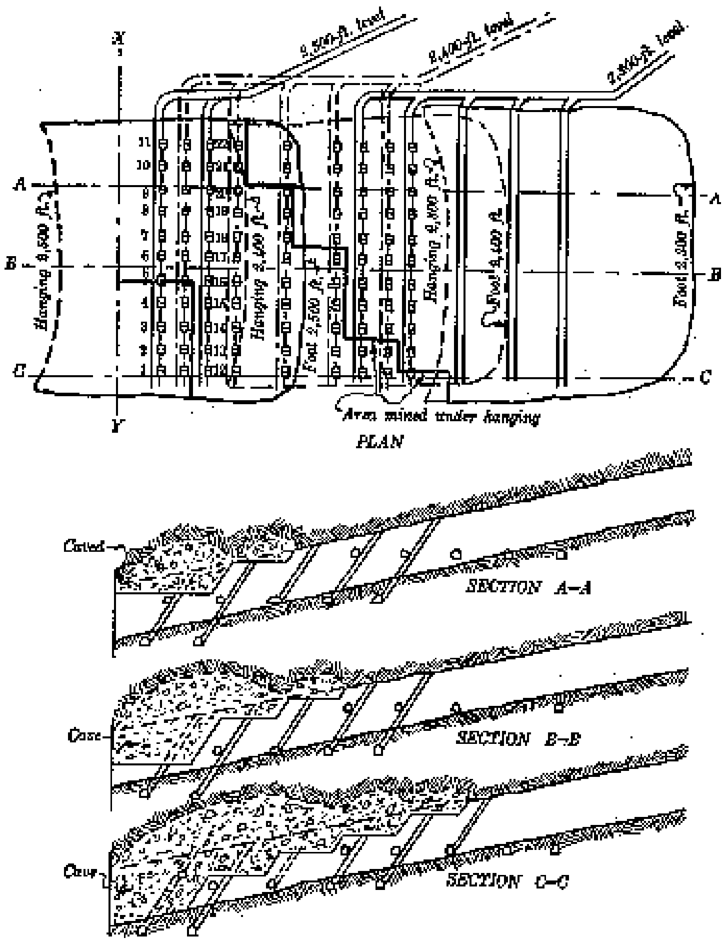


FIGURE 74.—Top-slicing system at Athens mine, Marquette range (after Elliott, S. R., The Athens System of Mining: Trans. Am. Inst. Min. and Met. Eng., vol. 66, 1921, pp. 220–228).

(4) The system is thought to be safer than the ordinary system because a large proportion of the miners always are working under a solid capping.

CANANEA, MEXICO

At the Capote and Veta Grandes mines square-setting and top slicing have accounted for most of the tonnage to date (1929).[52]

[52] Catron, William, Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico: Inf. Circ. 6247, 1930, 41 pp.

Top slicing also is employed in the Colorada mine for mining pillars between finished cut-and-fill stopes and parts of the ore body which are too soft and irregular for cut-and-fill stoping. Slices are 11 feet thick, and each floor has drift connections with safety exits outside the ore body. Main raises are 40 feet apart along the center of the block of ore, and branch raises are driven from these to give the required spacing on the slice floors. Blocks above the 800-foot level are 30 feet wide, and those below this level are 50 feet wide.

In one pillar between cut-and-fill stopes a modification of the usual method was on trial in 1932. Under this scheme the broken ore is dropped through a series of short raises from the slicing floor

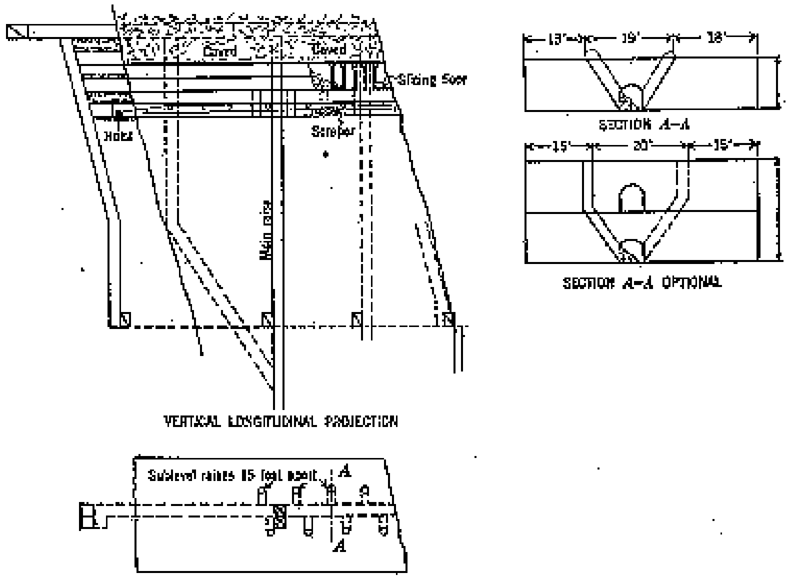


FIGURE 75.—Transfer sublevel top-slicing system, La Colorada mine, Mexico.

to a sublevel one or two floors below. From here the ore is dragged by a scraper to a main raise to the haulage level. (See fig. 75.) This necessitates less development than under the old system and gives better chute spacing on the slicing floor. The figure shows the reduction in the amount of raising required. The dotted lines on the longitudinal projection are raises required under the old system but not with the transfer system.

Direct stoping costs in top slicing for the first 6 months of 1931 were as follows:

Direct stoping costs, top slicing, Colorada mine, 1931

	Cost per ton		Cost per ton
Labor (timbering, breaking, and mucking)	$0.4237	Timber	$0.2634
Compressed air, drills, and steel	.0959	Miscellaneous supplies	.0072
Explosives	.1314	Gobbing	.1098
		Total	1.0354

MIAMI, ARIZ.

A block or panel system of top slicing formerly was employed at Miami for mining the higher-grade ores.[31] The area in which top slicing was used was about 800 feet square. Most of the ore was soft but was considerably harder than the capping. The latter was siliceous and seldom contained any clay or other binding material, so that it broke into fine particles which ran like sand.

Haulage levels were 150 feet apart, with two intermediate sublevels at 50-foot intervals to facilitate building chutes and for ven-

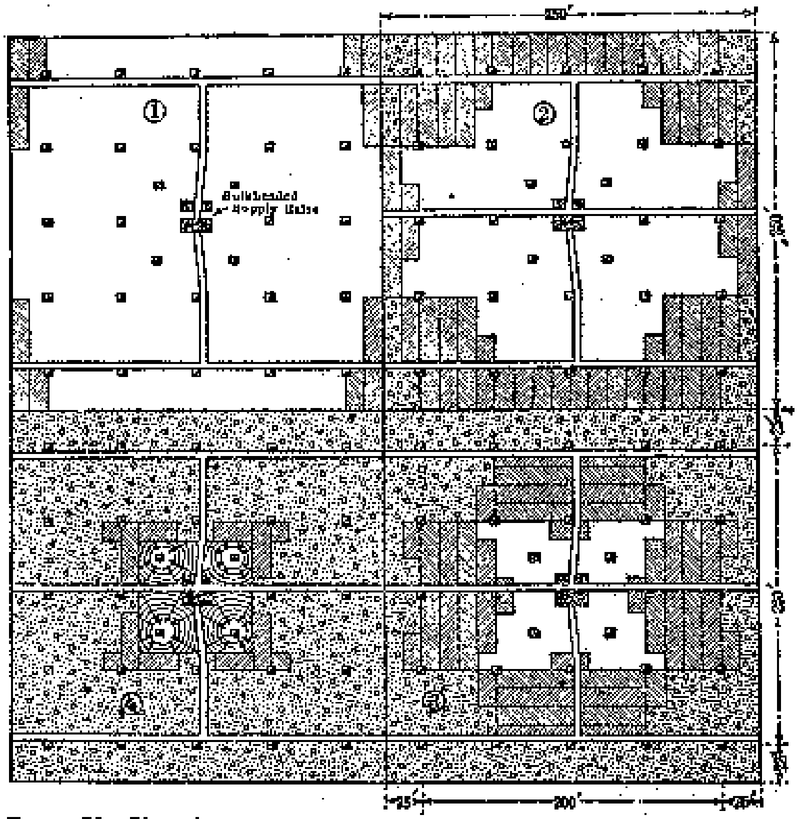


FIGURE 76.—Plan of successive steps in block method of top slicing, Miami, Ariz.

tilation during slicing. Haulage drifts were driven on 50-foot centers, and raises were put up 50 feet apart along these drifts. For slicing the area was blocked out into mining units 200 to 250 feet square after it was found that in working a long face of retreating slices the entrance drifts took weight and required a large amount of retimbering.

At the center of each block (fig. 76) a two-compartment supply raise was put up. Four bulkheads were placed as shown in section

[31] Dean, E. G., Block Method of Top Slicing at Miami: Min. and Sci. Press, Oct. 21, 1916, pp. 601–602.

1, figure 76, and drifts were run to the boundaries of the block. The sequence of slicing is shown in the figure. Slices were timbered with sets consisting of two 8-foot posts and a 12-foot cap, or with double sets of three 8-foot posts and two 7-foot caps. The slices were 10 feet high. As soon as the first slices had advanced a few feet the second and third slices were begun, also the first slices toward the centers of the block limits. Where a good mat had not been established a floor of 2-inch plank was spiked to 2- by 10-inch sills. Where there was a good mat no floor was laid in the slices before blasting down. When the slice timbers showed signs of taking weight bulkheads of old timbers were built at the entrance to the slices and the posts were drilled and shot down.

As the last of the ore in a block was being taken, crosscuts were driven to inclined raises put up to about the center of the four central pillars (fig. 76, section 3) and slicing was continued, working from the outside of the remaining ore toward the central supply raise. The ore was shoveled by hand into wheelbarrows and dumped into the raises.

This system had an advantage in that a large number of working places could be concentrated in a comparatively small area and the blocks could be worked out quickly, thus reducing retimbering and maintenance. Dean states that, compared to the first system tried using long faces, the production per shoveler-shift was raised from 9 to 20 tons and that per man-shift from 5 to 10 tons.

MORENCI DISTRICT—INCLINED TOP SLICING

In 1910 top slicing and sublevel caving were used at the Metcalf, Morenci, and Coronado mines in addition to open, cut-and-fill, and square-set stopes.[34] Flat or horizontal slicing was used at this time with a sublevel interval of 9 to 11 feet. Before top slicing was begun, 2 or 3 floors of square-set stopes were taken below the capping to level off the top of the ore body, and a good timber mat was established under which top-slicing progressed.

In 1917 an inclined top-slicing method was in use in the Coronado mine.[35] (See fig. 68, p. 263.)

The following description is taken from Bureau of Mines Information Circular 6107, page 14.

> In 1917 an inclined top-slicing method was in use in the Coronado mine of the Arizona Copper Co. This method had the advantage of eliminating all shoveling of ore in the stopes. The broken ore was blasted down on the sloping floor of the stope from which it ran into a narrow shrinkage stope at the bottom of the trough. The ore from the shrinkage stope was drawn directly into cars through chutes on the motor-haulage level. As a result of the substitution of inclined top slicing for flat top slicing, the tons of ore mined per man-shift in top-slice stopes increased from 4.90 in 1916 to 11.20 in 1917. The total cost of labor and material decreased from $1.06 to $0.75 during the same period.
>
> A variation of the same method of inclined top slicing was adopted in the Humboldt mine at Morenci in 1918. At this mine the development work consisted of running sublevels at 55-foot vertical intervals and extending raises from the sublevels to the stopes. Each sublevel had a central drift through

[34] Mosier, McHenry, and Sherman, Gerald, Mining Practice of Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz.: Inf. Circ. 6107, Bureau of Mines, 1929. 33 pp.

[35] Scotland, P. B., The Coronado Top-Slicing Method: Eng. and Min. Jour., vol. 103, Apr. 7, 1917, pp. 561-562.

the ore body, with crosscuts at right angles to it at 50- or 60-foot intervals. From the latter, untimbered raises not over 4 feet in diameter were put up every 15 feet. These lines of raises formed the center lines of the individual slices. In stoping, an inclined heading was driven 44 feet above the sublevel, starting at one end of the line of raises. This heading was timbered with sets 11 feet high and 2½ feet apart. As soon as the heading had gone far enough to intersect four raises, stoping was begun. The ground was taken out in panels 10 feet wide and 50 or 60 feet long, corresponding to the distance between drifts on the sublevel. The floor of the panel was inclined 33° on either side of the center heading, and the floor of the heading was funneled down to the raises, so that ore blasted from the faces of the panels would run into one of the raises without any handling. When the heading had advanced to open another raise, which completed the panel, the timber was blasted and another started. Posts in the panels were 7-inch round timber, 10 feet long, on 4-foot centers up the incline and 5-foot centers parallel to the heading. These posts were not set normal to the floor of the panel but inclined toward the center heading at an angle of 16½° from the vertical. Flooring consisted of 2- by 12-inch planks laid upon sills of round timber. The sills were so placed as to be readily picked up by the posts of the next lower slice. The ore was drawn from chutes on the sublevel and hand-trammed to motor-haulage pockets. The substitution of inclined top slicing for flat top slicing resulted in increasing the tons mined per man-shift from 5.4 in 1917 to 8.8 in 1919. The timber consumption was 7.9 board feet per ton with flat top slicing and was 9.0 board feet with inclined slicing. The total cost showed a reduction of 15 percent in favor of the inclined method.

It would seem that, with the development of underground scraping and scraper hoists to their present efficiency, any advantage that inclined top slicing might formerly have had over horizontal slicing would exist no longer.

The labor of handling and installing timber, drilling equipment, and gear on the inclines and the difficulty of working on a sloping floor compared to a flat floor are disadvantages of the inclined top-slicing system. Tons per man in flat slicing using scrapers average about 15 (2,240 pounds) in the Lake Superior district as against 11.2 (2,000 pounds) for inclined top slicing at Morenci. Moreover, inclined top slicing at the Humboldt mine required more timber than flat slicing.

BISBEE, ARIZ.

About 1914 it was decided that the top-slicing method employed on the Mesabi range might be applied to a large body of oxide ore in which heavy and swelling ground made the cost of square-set stoping excessive.[58] The deposit was divided into sections 45 feet wide across the ore body, which ranged up to 80 feet in width; the economical limit was 100 feet high.

The sections were developed by and mined through a pair of double-compartment timbered raises. (See fig. 77, *f* and *g*.) From the chute compartment parallel lead rows of square-sets were carried the width of the ore or section on the three floors beneath the hanging wall. On each succeedingly lower floor the square-sets are carried for a shorter distance to leave a footwall of unbroken ground sloping toward the chutes in the main haulage drift. A pillar 40 feet wide is left between the square-set rows which is mined by horizontal slices from the top downward, one-half of the pillar being taken from each side. The back is caved as stoping progresses.

It was found that the maximum slice that could be carried safely was 10½ feet thick, and this thickness was not attempted until the

[58] Willey, Howard, Top Slicing with Filling of Slices, as used at the Charcas Unit of Am. Inst. Min. and Met. Eng., vol. 15, 1916, pp. 127–131.

fourth slice was taken and the caved mat was under perfect control. Stoping cost was reduced 10 percent compared to ordinary top slicing; labor cost was reduced about 15 percent; and consumption of powder was less. The ore could be mined rapidly and safely, and an output of 10 tons could be obtained per man-shift, or 125 tons per section per day.

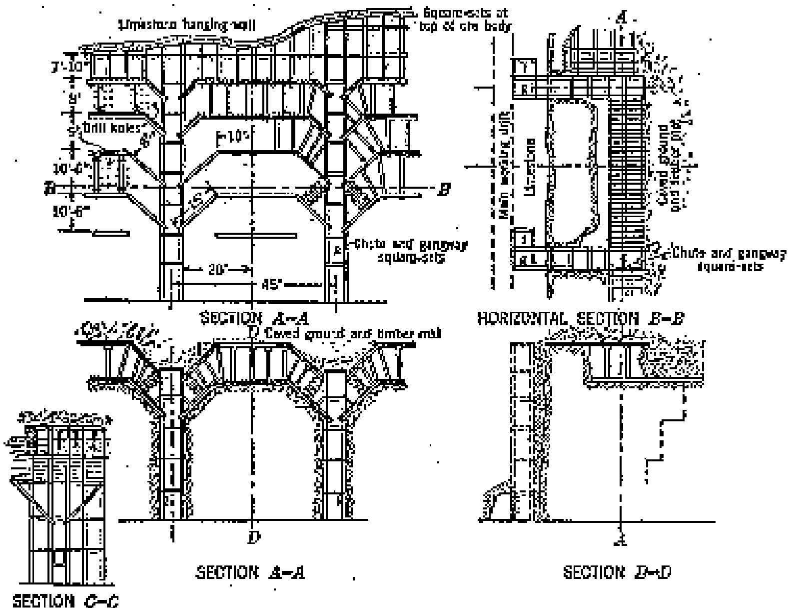


FIGURE 77.—Mitchell top-slice system, Bisbee, Ariz.

CHARCAS, SAN LUIS POTOSI, MEXICO

Top slicing with filling of slices has been used at the Charcas unit of the Cia Minera Asarco S. A.[37] to mine partly worked-out sections of the mine and the very heavy ground in most of the stopes opened in virgin ore. The ore shoot is about 1,400 feet long in a fissure, with an average dip of 70°. The footwall ranges from strong limestone to broken, metamorphosed limestone that requires heavy timbering. In some sections it is an altered, crumbly porphyry. The stoping width ranges from a few feet to more than 90 and averages about 30 feet.

The method employed is stated to have the following advantages over square-setting: (1) Greater safety for the workmen, (2) better total recovery and cleaner mining, and (3) lower cost per ton mined.

The stopes are developed by two-compartment raises 100 feet apart, which are driven from a footwall tramming drift and leave a 6-foot pillar between the raises and the vein.

The top floor or slice is mined by square-setting without any sills under the posts. As square-setting proceeds, sills 10 feet long or longer are laid between the posts from foot to hanging wall and are placed on 5-foot centers. A floor of 2-inch plank is laid on these

[37] Willey, Howard, Top Slicing with Filling of Slices, as used at the Charcas Unit of the Cia. Minera Asarco, S. A.: Tech. Pub. 364, class A, Met. Min., no. 41, Am. Inst. Min. and Met. Eng., 1930, 17 pp.

sills. As flooring is laid the square-sets are back-filled as closely as possible with waste.

After the square-set floor has been completely mined and filled a crosscut is started to the vein from the ore pass 12 feet below the floor, from which a narrow drift is driven along the footwall of the vein 50 feet in each direction to connect with the raises on either side and to develop the next slicing level. This drift is usually 12 feet high with two posts at 5-foot intervals catching up the sills of the floor above.

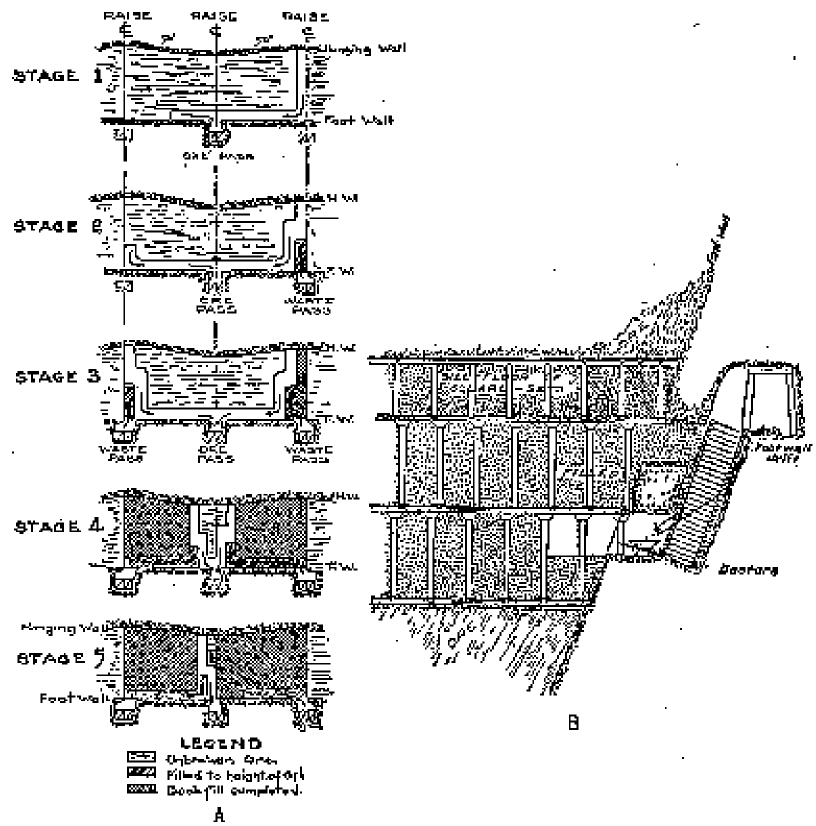


FIGURE 78.—*A*, Plan of stope floor, Charcas, San Luis Potosi, Mexico, showing progress of mining; *B*, section through waste pass. (After Willey, Howard, Tech. Pub. 354, Am. Inst. Min. and Met. Eng., 1930.)

Slices 5 feet wide are driven from this drift toward the hanging wall, using pairs of posts on 5-foot centers to catch up the sills of the floor above. A plan of the method is shown in figure 78, *A*. It differs from the usual top-slicing system in that (1) the slices are back-filled as they advance, (2) posts and headblocks are used instead of drift sets to catch up the floor above, and (3) since the slices are filled the timbers are not blasted out. Waste for filling is obtained by connecting the footwall drift with the waste pass as shown, the floor of the connecting crosscut being 6 feet above the bottom of the slice. (See fig. 78, *B*.) As the slices advance, sills are laid and floored as on the top floor and the waste filling is placed on this floor. Successively lower floors are worked in the same manner.

Cost savings per ton with this method compared with square-setting are as follows:

	Mining in virgin ore	*Mining in old workings*
Labor	$0.106	$0.102
Explosives	.008	.011
Lumber and timber	.411	.270
Total saving by use of filled top-slice stoping at Charcas	.525	.383

Production per man-shift is approximately the same as for square-set work, as follows:

	Virgin ore, metric tons	*Old workings, metric tons*
Square-set	1.55	1.24
Filled top slice	1.44	1.34

The recovery of ore by the filled top-slice method is estimated at virtually 100 percent as against 95 percent by square-setting with back filling.

FRANKLIN, N. J.

For some years most of the output from the Franklin mine has been produced by top-slicing pillars between filled transverse stopes. These pillars average about 32 feet in width, their length being the same as the width of the ore body, which ranges up to a maximum of 150 feet. The level interval is 100 feet, so that these pillars comprise blocks of ore 100 feet high and 32 feet wide, extending from foot to hanging wall.

The pillars are developed for top slicing from individual footwall raises connected by longitudinal footwall drifts. The raises are opposite the center lines of the pillars, and, from these, drifts are driven through the centers of the pillars on each sublevel, beginning at the top. Raises are put up from haulage-level crosscuts through the pillar at 20-foot intervals to serve as ore passes from the slices. The sublevel interval is 10 feet.

When the central drift on the top sublevel reaches the hanging wall slices are driven right and left along the hanging wall toward the filled stopes on each side and timbered with hardwood timber, principally oak. Light blasts are used for breaking the ore, and space loading is employed to prevent blasting out the timbers. When the fill is approached the last of the ore in the slice is removed a little at a time, beginning at the top and working breasting boards in to hold back the sand filling. Slices are 6 to 8 feet wide, and when the hanging-wall slice has been completed to the fill a second slice is driven alongside it in a similar manner. Sometimes three or more slices can be driven before excessive weight develops. Before the timbers are blasted down the floor is covered with old timbers and lagging.

After the top sublevel has been sliced back far enough slicing may be begun on the next sublevel, 10 feet below, beginning at the hanging wall as before. Slicing is continued from the top downward until the entire pillar has been mined.

As the hanging wall is hard and firm it hangs up over the top slices, leaving an opening between the capping and the top of the gob. Because this hanging wall might collapse suddenly when the unsupported span becomes too great, waste rock and mill sands are dumped in from the level above to keep the space filled and to provide a cushion over the top-slice workings.

This application of top slicing is quite different from the usual one, where the capping caves and fills the space formerly occupied by the ore.

SUMMARY OF TOP SLICING

1. *Applicability.*—Top slicing is adapted particularly to mining fairly wide ore bodies in which the ore is too weak to stand except over short spans and where the capping will cave readily when the support is removed and fill the excavation tightly. A capping that breaks into large blocks which arch over and hang up so as to leave open cavities in the gob would constitute a serious hazard in top slicing. Thus heavy top weight is essential for the best results by this method. Top slicing is used for mining bodies of ore only one slice (5 to 18 feet) thick, when it is virtually a longwall retreating system of stoping. It also is adapted to mining pillars between caved or filled stopes and to recovering badly broken or caved ore which it is unsafe to stope from the bottom upward. Obviously it can be used only where there is no objection to caving the capping and to probable resultant breaks to the surface. It also is employed sometimes under a strong hanging wall or capping, as at Franklin, N. J. Since top slicing requires considerable timber and lagging a plentiful supply of cheap timber should be available.

2. *Flexibility.*—Top slicing is a nonselective method, as usually only a small amount of sorting may be done in the slices, although sometimes considerable waste can be thrown aside in adjacent mined-out slices which have not yet caved, particularly where the slices are not blasted down but are allowed to crush and close gradually. Small blocks of lean ore or waste within the ore body must be taken out or blasted into the cave to insure their breaking up and caving; otherwise such blocks will be a menace to the miners when they come to slice below. If a good mat is maintained between the caved overburden and the backs of the slices there will be little if any dilution of the ore and clean ore can be obtained. If it becomes necessary to curtail or stop production for a time the mine can be left in good condition for resumption of operations by completing the active slices so that no small pillars of ore are left between them, then boarding them up and blasting them down.

3. *Recovery.*—Top slicing permits recovery of virtually all the ore if conducted properly and a good mat is maintained. Small stubs occasionally are lost between the ends of slices or between the end of a slice and the cave. Two or three feet of ore sometimes must be left around the margins of the deposit where a flat footwall comes up to the overlying mat, or some wall rock must be taken up to make working room.

4. *Development.*—Although considerable preliminary development in the form of chute and manway raises and sublevel drifts is required it serves to prospect and block out the ore thoroughly, and in soft ores such as those in which top slicing is used most this development is relatively inexpensive and produces considerable ore. At one property, for example, the cost per ton of development ore is only about 20 percent more than that of slice ore. Table 26 gives

figures on the tons of ore developed per foot of stope development at a group of mines employing top slicing.

5. *Cost.*—Table 27 presents costs of top slicing at a number of mines. In this table, as in cost tables for other stoping methods, general mine development, tramming, haulage, hoisting, pumping, and general mine expense are not included. Although considerable timber and lagging are required the grade of timber need not be as good as that for permanent supports, and in localities where cheap timber is plentiful the cost per ton for timber is moderate.

TABLE 26.—*Relation between feet of stope development and tonnage developed, top slicing*

Mine	Variation of top slicing	Sublevel interval	Ore developed per foot of stope development, tons	Stope development per ton of ore developed, foot [1]
Mesabi:				
No. 1	Parallel slices, scraping into cars	(2)	90	0.0111
No. 2	do	(2)	99	.0101
No. 3	do	(2)	48	.0208
No. 4	do	14	36	.0278
No. 5	do	13	49	.0204
No. 6	Radial slices, scraping to chutes	12	41	.0242
Vermilion, No. 7	do	12½	64	.0156
Cayuna, No. 8	do	13	90	.0111
Marquette:				
No. 9	Parallel slices, scraping to chutes	11	70	.0143
No. 10	Radial slices, scraping to chutes	11	70	.0143
No. 11	Radial or parallel slices, scraping to chutes	10	164	.0061
No. 12	Parallel slices, scraping to chutes	11	114	.0088
No. 13	do	10	164	.0061
Menominee, No. 14	do	12	38	.0263

[1] Long ton (2,240 pounds). [2] Mostly single sublevel.

TABLE 27.—*Direct stoping costs, top slicing*

Mine and year	Variation of top slicing	Costs per ton of ore mined: Labor and supervision	Compressed air, drills, and steel	Power	Explosives	Timber	Other supplies	Total
Mesabi, No. 1, 1929 [1]	Parallel slicing, scraping to cars.	$0.4643	$0.0239	$0.0230	$0.0606	$0.1411	$0.0140	$0.7269
Marquette, No. 9, 1929 [1]	Parallel slices, scraping to chutes.	.499	.005	.009	.055	.079	.003	.739
Marquette, No. 10, 1928 [1]	Radial slices, scraping to chutes.	.393	.045	.047	.075	.070	.075	.707
Menominee, No. 15 [1]	do							.411
Coronado, Morenci, Ariz., 1917 [2]	Inclined top slicing.							.750
Miami, 1916 [2]	Blocks, hand shoveling.	.350	.030		.040	.130		.550
Calumet and Arizona, 1916.[2]	Mitchell top slicing.	.540	.100		.040	.210	.040	.930
El Oro, 1925 [2]	Stringers and posts, hand shoveling.	.460	(3)	.010	.070	.250		.780
Chacras, 1929 [4]	Filled top slicing	[5] .170			.405	.411		
Do.[6]	do	.103			.011	.270		

[1] Per long ton of 2,240 pounds.
[2] Per short ton of 2,000 pounds.
[3] Included in other costs.
[4] In virgin ore; per metric ton (2,204.6 pounds).
[5] Labor.
[6] In old workings; per metric ton.

TABLE 28.—*Man-hours per ton stoping, top slicing*

Mine	Man-hours per ton stoping				Tons per man-shift stoping
	Breaking (drilling and blasting)	Shoveling and scraping	Timbering	Total	
Mesabi:[1]					
No. 1	0.080	0.350	0.270	0.725	11.03
No. 2				.615	15.62
No. 3				.585	13.68
No. 4				.540	14.80
No. 5				.617	12.96
No. 6				.645	12.40
Vermilion, No. 7[1]				.401	19.94
Cuyuna, No. 8[1]				.446	17.94
Marquette:[1]					
No. 9	.093	.104	.270	.533	16.01
No. 11				.919	8.70
No. 12				.550	14.30
No. 13				.696	11.50
Coronado, Morenci, inclined slicing, 1917[2]				.714	11.20
Calumet and Arizona[2]				.800	10.00
Miami[2]				.800	10.00
El Dorado[3]				2.300	3.33
Charcas:[1]					
Virgin ore				5.555	1.44
Old workings				5.970	1.34

[1] Per long ton of 2,240 pounds.
[2] Per short ton of 2,000 pounds.
[3] Per metric ton of 2,204.6 pounds.

TABLE 29.—*Consumption of explosives, stoping, top slicing*

Mine	Explosive used		Explosive consumption per ton of ore stoping, pound
	Kind	Strength, per cent	
Mesabi:			
No. 1	Red Cross and gelatin	40	[1] 0.395
No. 2	Special gelatin	35, 40, and 60	[1] .601
No. 3	Red Cross	40	[1] .604
No. 4	Gelatin	35	[1] .403
No. 5	Red Cross and special gelatin	30	[1] .530
No. 6	Gelatin	35 0.473; 60 .161	[2] .634
Vermilion, No. 7	do	35 .396; 60 .003	[1] .399
Cuyuna, No. 8	Red Cross	35–40–60	[1] .547
Marquette:			
No. 9	Gelatin	50–60	[1] .372
No. 10	Ammonia gelatin	60	[1] .244
No. 11	L. F. standard ammonia gelatin	50–60	[1] .482
No. 12	L. F. gelatin	50–60	[1] .310
No. 13	do	50–60	[1] .353
Menominee, No. 14			[1] .467
Calumet and Arizona			[2] .175

[1] Per long ton of 2,240 pounds.
[2] Per short ton of 2,000 pounds.

TABLE 30.—*Timber consumption, top slicing*

Mine	Sublevel interval, feet	Length of timbers, feet		Slice covering		Timber per ton of ore [1]		
		Caps	Posts	Side	Bottom	Timbers [2]	Poles, linear feet	Lagging
Mesabi:								
No. 1	(3)	8–10	8–12	Hardwood boards		2.83 board feet		
No. 3	(3)	10	8–10	Wire fencing		1.10 linear feet	1.24	0.0012 cord, 6-foot lengths.
No. 4	14	8	12	do	Hardwood boards	1.06 board feet	1.266	.0018 cord.
No. 5	13	10	11	do	do	1.34 board feet		.0016 cord.
No. 6	12	9–12	10	do	Wire fence on poles	.06 linear foot	1.26	.0082 cord.
Vermilion, No. 7	12½	8	10	None				
Cuyuna, No. 8	11	8–12	8–10	None	Boards	.740 linear foot		1.00 linear foot.
Marquette:								
No. 9	11	9	9	Wire or boards	Wire on poles	1.121 linear feet		2.686 linear feet.
No. 11	10	9	8	do	Plank or wire on poles	1.702 board feet		3.241 linear feet.
No. 12	11	9	9	do	do	1.117 board feet	1.640	4.474 linear feet.
No. 13	11	9	9	do	do	1.378 board feet		3.082 linear feet.
Menominee, No. 14	12	10	10	Wire fencing	Wire on 2-inch plank	.625 linear foot		1.493 linear feet.
El Bordo [4]	12	[5] 13	11		2½- by 8-inch plank	7.1 board feet sawed plus 4.5 board feet round.		
Coronado, 1919	12		10		2- by 12-inch plank	0.60 board feet		

[1] Per long ton of 2,240 pounds, except as noted, footnote 4.
[2] Where board feet are given round timber has been calculated to board feet except as noted.
[3] Mostly single sublevel.
[4] Per metric ton of 2,204.6 pounds.
[5] Double 2½- by 10-inch plank sills.

HEALTH AND SAFETY

The necessity of considering the health and safety of the miners in the selection and employment of a stoping method has been mentioned. Health and safety are big subjects and come properly within the scope of investigations of the Health and Safety Branch of the Bureau of Mines rather than within a discussion such as this one. However, table 31 is presented to indicate in a general way the relative hazard of each of the principal stoping methods.

TABLE 31.—*Man-hours worked and accident data, classified according to stoping methods, at mines in the United States with a production worth more than $100,000 in 1929*[1]

Stoping method	Man-hours worked	Percent of total man-hours	Accidents per thousand 300-day workers	
			Number killed	Number injured
Square-set	40,345,283	24.8	3.91	303
Cut-and-fill	11,000,872	6.8	4.09	228
Shrinkage	14,673,970	9.0	5.37	302
Open stoping	39,905,832	24.6	3.28	256
Top sliding	14,828,200	9.1	3.16	71
Sublevel caving	6,922,446	4.3	2.39	57
Block caving	11,879,024	7.2	4.81	219
Open-pit mining	23,143,160	14.2	2.18	112
Total	162,047,376	100.0	[2] 3.60	[2] 228

[1] Wright, C. W., Mining Methods and Costs at Metal Mines of the United States: Inf. Circ. 6503, Bureau of Mines, 1931, 39 pp.
[2] Average.

The figures in the last column are more significant than those in the preceding column.

INDEX