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Methods and Economies in Mining

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Introduction

In any discussion of mining one is repeatedly confronted with the difficulty of dealing with so many variable conditions. It is not an exact science and in the choice of a method each varying factor has a certain weight, which, in many cases, experience alone can determine. In mathematical terms, it is a function of many variables.

A discussion of mining also loses much of its value unless costs are considered, because the expectation of profit is the only excuse for carrying on mining at all. As conditions vary they cause fluctuations in cost and there are few operations to which a definite cost can be assigned. The character of the ore may make it difficult to drill, yet because of the ease with which it breaks the total cost of drilling and blasting may be low.

In preparing this article the attempt has been, not to cover the whole field of mining, but to describe the different methods of stoping and mining which have distinct principles. In addition to this an effort has been made to show the advantage of dissecting the methods into their detailed operations and applying to these a mathematical study as an aid to the judgment in determining which is the best method to adopt, or in attempting to reduce the cost of a method already in use.

Methods of mining include stoping, caving, and various methods of working large deposits which, in addition to the method of actually breaking the ore, require elaborate and definite plans of development of the orebody. The ordinary methods of stoping are too familiar to all for any elaborate discussion, but it has seemed advisable to review the subject and give the principal advantages and disadvantages of the different methods.

Factors Affecting Choice of Method

In addition to the various external factors, such as the supply of labor and the financial status of the operating company, the principal items that influence the method of mining to be adopted are:

The size and shape of the deposit:
Character of the ore, whether high or low in grade.
Whether the values are regularly or irregularly distributed.
Physical character, whether hard or soft, tough or brittle.
Character of the country rock.
Immediate and future demands for ore.
Amount of development work done or that may be necessary.
Amount of water to be handled.
Cost of power, timber, and supplies.
Ventilation.
Whether drilling is to be done by hand or with machine drills.

Review of Stoping Methods

Depending on these factors, the following methods of stoping may be employed:

- Underhand stoping:
  - Ore is hoisted to the level above; Cornish stoping.
  - Ore is drawn from the level below.
- Overhand, or back stoping:
  - Starting stopes.
  - Drift stoping.
  - Cutting out, or lead stoping.
  - Raise stoping.
  - Longitudinal back, flat-back, or long-wall stoping.
  - Rill-cut stoping.
  - Saw-tooth back stoping.
  - Shrinkage stoping.
- Combination stoping.
- Breast stoping.
- Side stoping.
- Sublevel stoping.
- Square-set stoping.
- Filling methods.

Underhand Stoping

The method of underhand stoping in which the ore is drawn to the level above (*B*, Fig. 1), is called Cornish stoping. It finds application only when it is necessary to mine a lens of good ore below a level and it is not practicable or financially possible to do the necessary development to come up from underneath. Its disadvantages are the excessive cost of raising both ore and water.

Underhand stoping where the ore is drawn from the bottom (*A*, Fig. 1), has more merit than is usually accorded it, especially in the Western States. On the Rand it has been used almost exclusively. Its advantages and disadvantages as compared chiefly to overhand stoping are as follows:

Advantages:

- Ease in drilling down holes when drilling is done by hand.
- Holes are drilled wet and dust is eliminated.
- No platforms are required on which to drill.

Disadvantages:

- Limited to steeper pitches than overhand stoping because the ore does not work straight down the foot wall to the raise; in flat pitches this would necessitate more shoveling.

Limited to good walls if the vein is steep. Loose rocks sluff off from poor walls and endanger the workmen below.

Levels ordinarily must be driven closer together to reduce the height of the unsupported walls; this necessitates an added cost of development.

Raises must be put up at frequent intervals. This work in some cases amounts to 35 or 40 per cent. of the total development.

No ore reserves are possible.

The waste that can be sorted in the stope if the vein is steep is limited to what can be thrown on lagging supported by stulls.

Certain efficient types of stoping drills cannot be used.

The great advantage if, of course, the drilling of down holes when drilling is done by hand. Few men to-day can or will drill very many

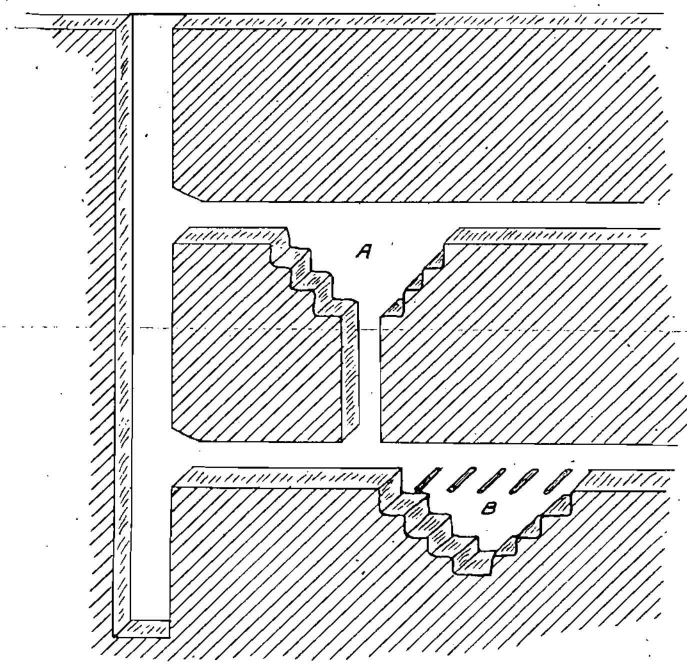


FIG. 1.—UNDERHAND STOPING.

"uppers" in a shift. South Africa had its native labor which could not be taught to drill them. It might seem on first thought that the disadvantages are so numerous that they preclude any chance of the method being adopted under ordinary labor conditions, but I have examined mines where the combination of conditions in narrow veins indicates very strongly that the method would be more economical than overhand stoping and quite as safe.

Overhand Stoping

Overhand stoping, in general, has the following advantages and disadvantages:

Advantages:

Stoping can be started directly from the level without any raises or winzes.

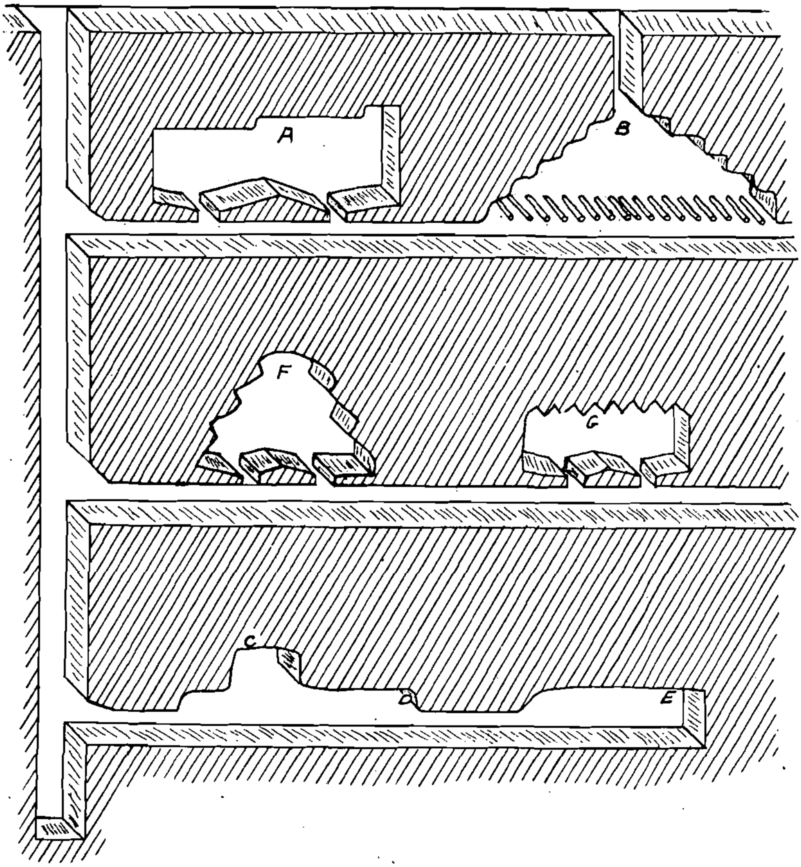


FIG. 2.—OVERHAND STOPING.

Levels can be driven far apart. These two considerations mean less cost foı development.

Advantage is taken of the force of gravity in breaking the rock.

Miners work at the back and can inspect it so that the danger of falling rocks is largely eliminated.

Either ore or waste can be stored in the stope.

A flatter seam can be worked than in underhand stoping without the necessity of shoveling, as the ore runs directly down the pitch and is given an impetus by the blast.

Water takes care of itself.

Pillars of ore or waste can be left easily.

Disadvantages:

In most cases holes must be drilled up or flat.

If the stope is not filled with ore or waste, stulls and platforms must be used, even though the walls require no support.

If the raises are far apart ventilation is poor.

Starting an Overhand Stope.—In Fig. 2 are shown various methods of working the back in overhand stoping. To start the stope, if the ore is low grade and timber is scarce, a pillar or pillars are left above the level as shown in *A*, and if the ore is high grade it is all removed above the level and stulls are placed as shown at *B*. To start a stope as shown at *B*, a cutting out or lead stope is broken immediately above the level as shown at *D*, Fig. 2. *C*, Fig. 2, is a wide raise or raise stope, which is one method of opening an overhand stope, and *E* is a drift stope, which is a term used in the Lake Superior copper mines and means a wide and high drift as a start for the overhand stoping.

Longitudinal Back, Flat-Back, or Long-Wall Stoping

After starting an overhand stope the shape in which the back is carried is often of prime importance. In *A*, Fig. 2, is illustrated the method of carrying a flat or longitudinal back (the term long-wall is used sometimes when the vein lies nearly flat). In this method the benches or breasts are frequently made of a height sufficient to allow of a square set being placed. This is done at Butte and it is often referred to as breast stoping. In general, a flat-back stope as illustrated has the following advantages and disadvantages:

Advantages:

If the stope is filled with waste or ore it is more convenient for men to work on a level surface. Tramming, shoveling, or sorting in the stope can be done to better advantage, and waste raises can be driven farther apart.

Square sets may be conveniently placed by making the benches the proper height.

Timber struts or cribs can be used between the filling and the back, in case the latter is weak.

A long stope distributes the broken ore well along the level, and tramming is thus facilitated.

Disadvantages:

The principal disadvantage of a flat-back stope is that if filling is used, and kept close to the back, it must be distributed in wheelbarrows or cars.

Rill-Cut Stoping

Rill cut, or rill stoping, as shown at B, Fig. 2, has the principal advantage that when filling is used it can be run down through a raise at the apex of the stope and will fill the stope without shoveling. Its disadvantages, on the other hand, are as follows:

When waste filling is run in from the raise it has a sloping surface which makes it difficult to keep ore and waste from mixing, and is unhandy for men to work.

Raises must be put in at frequent intervals, and the added cost of these raises may exceed the cost of spreading the filling in a flat-back stope.

Ore chutes must be carried up through the filling, and the timber used in their construction cannot be recovered. This is a disadvantage as compared with shrinkage stoping.

At F, Fig. 2, is shown a method of rill stoping in which the benches are inclined so that down holes can be used. Down or water holes give off no disagreeable dust, and can be drilled faster with piston machines.

Saw-Tooth Back Stoping

This method of carrying the back in an overhand stope is shown at G, Fig. 2. It is claimed that it makes drilling more convenient if machine drills mounted on a bar are used.

Shrinkage Stoping

Shrinkage stoping refers to any overhand method in which the stope is kept full of broken ore until it is completed. The miners stand on top of the broken ore and work at the back. As broken ore takes up more space than ore in a solid mass, about 35 to 40 per cent. of it must be drawn to leave room for working.

Advantages:

Raises may be placed far apart.

The broken ore serves as a support to the walls. This does away with the necessity for much timber.

The miners work on top of the broken ore; timber platforms are eliminated, and the work is made much easier. It is also convenient to work a larger number of men in the stope.

A large ore reserve is maintained.

Large rocks can be broken with sledges in the stope and blocking the chutes is avoided.

No ore passages are required from the level up to the back of the stope. Manways are necessary up through the broken ore, but the timber used in their construction can usually be recovered when the ore is drawn.

Ventilation is better than in an empty stope.

Disadvantages:

Stoping must be kept ahead of the demand for ore. This requires additional capital.

There is practically no opportunity to sort ore in the stope.

Ore filling is not permanent and the stope may cave when it is withdrawn.

Scaling or unstable walls may cause waste to mix with the ore or prohibit the use of the method altogether.

Although shrinkage stoping is rather generally used, it would be used more if it were not for the fact that many mines are not in a condition to keep an ore reserve, but must draw the ore for the mill as fast as it is broken. The added efficiency to be obtained from the miners when working on a firm floor of ore, and not on loose lagging laid on stulls, is a very important advantage of the method.

Combination Stoping

In a discussion of stoping methods, steeply pitching veins are usually referred to, because the majority of veins in nature are steeper than 45°, and also because to refer each method of stoping to veins of all dips causes confusion. Before considering combination stoping, however, a brief *résumé* of the methods of handling ore in veins of different dips is necessary. In veins dipping from 35° or 40° up to 90° the ore runs down by gravity. From 20° to 35° it must be assisted by shoveling, or by using chutes with smooth bottoms or placed at an angle steeper than that of the vein. From 10° to 25° the ore may be taken down to the level by shoveling, shaking chutes, mono-rail trams, gig-back railways, or conveyors. From veins that are horizontal up to those having dips of 10° to 15° tracks are usually laid on the foot wall and cars are pulled up to the face by men or animals. An animal can pull an empty car up a 6° slope and a loaded car up a 3° slope. So in dips greater than 6° the track must be laid at an angle with the line of the dip.

In combination stoping, which is illustrated in Fig. 3, and which is a combination of underhand and overhand stoping, it is possible to keep the working face more nearly in a line parallel to the raises and perpendicular to the level. This is a distinct advantage if a stationary or shaking chute is being used to carry the ore to the level, because a large part of the face of the stope is accessible to the chute. This becomes a double advantage if a large output is required. In combination stoping development work is also reduced because levels may be driven far apart, and fewer raises are required than in simple underhand stoping.

Side Stoping

This is an extreme of combination stoping in which the working face of the stope is vertical. The face is parallel to the raises just as the face in flat-back stoping is parallel to the levels.

Breast Stoping

Breast stoping refers to the working of a flat orebody, or a flat section of an orebody, just as coal is mined from a flat coal seam. That is, a slice is worked in a horizontal direction. The assumption is that there is no open stope either above or below the slice, or else the method

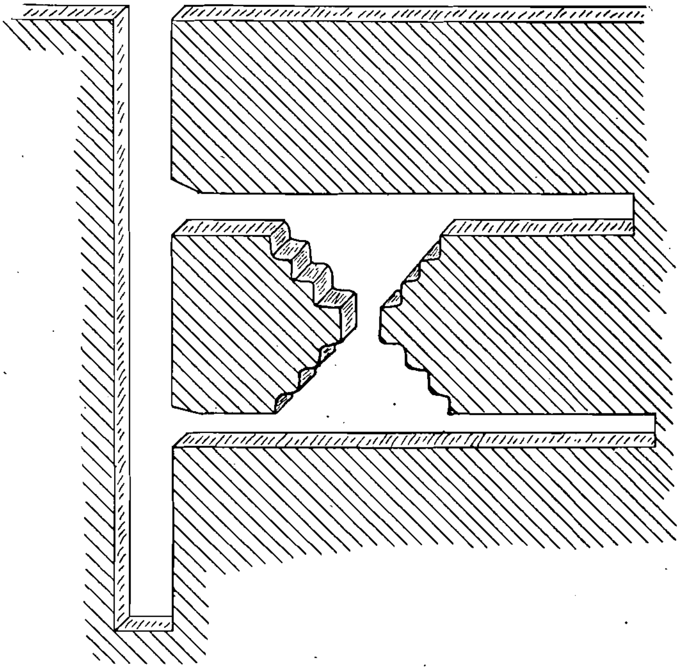


FIG. 3.—COMBINATION STOPING.

becomes either underhand or overhand stoping. In some cases the benches in underhand or overhand stoping are called breasts, and the method breast stoping, but I believe that is not the usually accepted definition. Breast stoping, in the strict sense, is the only name applicable to the mining of a horizontal slice, such as the sill floor of a large overhand stope, or the slices in the top-slicing method.

Sublevel Stoping

This method, described by F. W. Sperr in the *Engineering and Mining Journal* of June 5, 1912, and by P. B. McDonald in the *Mining and*

Scientific Press of July 5, 1913, is really a combination of several different methods of stoping. Fig. 4 is an illustration of the method, and shows half of the vein cut away where the stoping is being done. The miners go from the haulage level up the raise into the sublevels. On the first sublevel they will blast into the shape of funnels the raises that come up from the haulage level, and after working back the slice, *s*, by breast stoping will drill holes at *a* and shoot part of this bench down into the raises. On the second sublevel the miners work back the slice, *s*, and then with down holes at *b* will blast the remainder of the first bench down into the raises, and with uppers at *c* shoot off part of the second bench. Both sets of holes will be fired at the same time. In this way

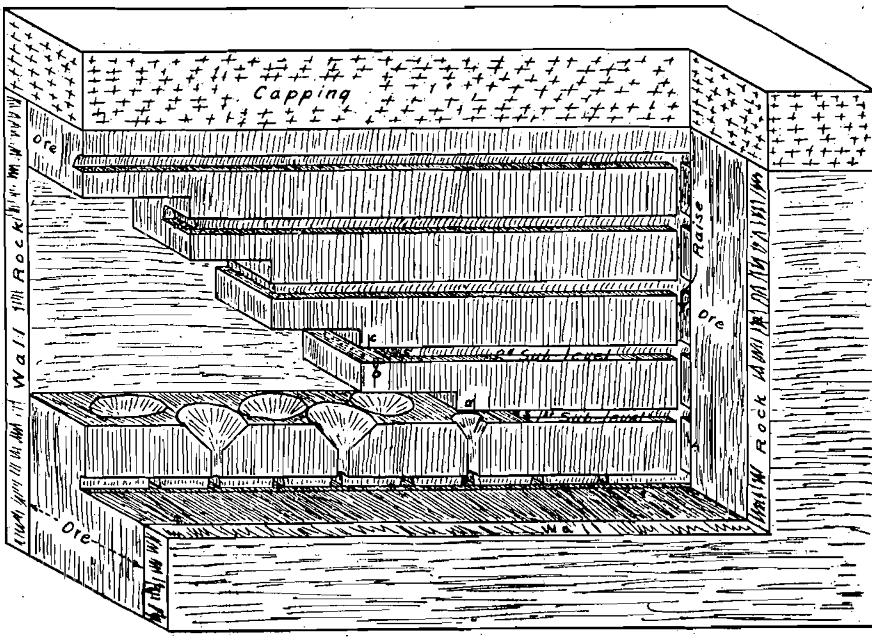


FIG. 4.—SUBLEVEL STOPING.

each sublevel is drawn back; the broken ore is drawn off through the funnel-shaped raises, which leaves a large open stope. The sublevels are driven about 25 ft. apart vertically, and 8 to 10 ft. are shot off from each bench from the sublevel below. Obviously where there is a capping above the ore, as shown in the cut, part of it will fall when the upper bench is blasted and part or all of this ore may be lost. "When the ore is from 50 to 100 ft. wide, a narrow stope of about one-third of the width of the ore is drawn back through the middle, leaving a pillar of ore standing on each side." These pillars or slabs left against the walls are then drawn back in the same manner. The method is applicable to veins from 12 to 100 ft. wide, or wider, if pillars are left between stopes, and evidently comes into competition with shrinkage stoping, square-set stoping, and top slicing. It requires expensive development but permits a large tonnage to be broken in a relatively small stope. The working back of the slices on the sublevel is expensive mining, but after that is done the rest of the ore blasted from the benches requires a comparatively small amount of powder.

Square-Set Stoping

Square-set stoping may refer to any method of stoping in which square-set timbers are used. Unless otherwise specified, however, the term is limited to overhand flat-back stopes in which square sets are used, either with or without waste filling. Square-set mining is advantageous when the vein is wide and the walls will not stand without timbering and shrinkage stoping cannot be used; when, on account of surface conditions, caving cannot be allowed, or where caving might cause the loss of other orebodies; when the ore changes rapidly in grade and requires frequent sampling; when the orebody is irregular in outline; and when old stopes may have to be approached or passed through at some later time.

Filling Methods

Mine workings are filled with waste as an aid to timbers in supporting weak walls or back, or to avoid fully or in part the use of timbers. Workings that must be prevented from caving for a length of time in the future are best protected by waste filling. The filling may be waste rock from development, rock blasted from the walls or surface for the express purpose of filling, or, if available, sands from concentration mills make excellent filling and are cheaply placed in the stopes.

Caving Methods of Mining

There are three distinctive methods of working large deposits that involve, in some way, the factor of caving. Slightly different names are in use, but those that are simple as well as descriptive are top slicing, sublevel caving, and block caving. In a discussion of these methods, three things must be borne in mind: that the methods are usually applied to massive deposits; that the deposits are usually divided into blocks or panels, and the description of mining one panel is practically a complete description of the method; and that, in most cases, the orebodies do not come to the surface but are covered with a capping. This capping may be glacial drift, as in some instances in the iron regions around Lake Superior, or it may be rock from which the ore values have been leached, as is the case at some of the large copper deposits.

Top Slicing

Top slicing, illustrated in its ideal form in Fig. 5, consists in the working of an orebody in horizontal slices, beginning at the top. Levels, for haulage, are established at proper intervals and, from these, raises are put up to the top of the orebody about every 50 ft. Starting at the tops of these raises drifts are run out and then, retreating toward the raise, the slice is worked back by breast stoping. The overburden or

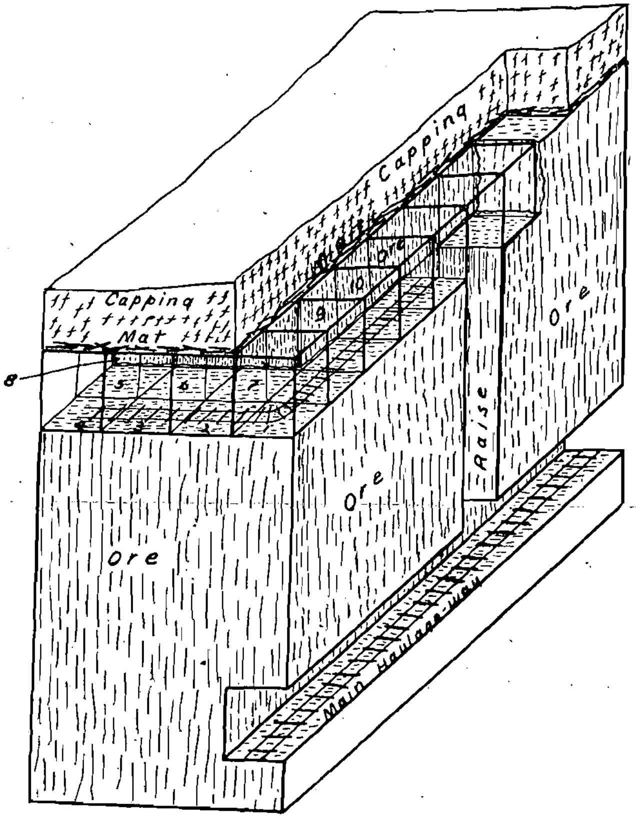


FIG. 5.—TOP SLICING.

capping lying above the ore is supported on square sets or posts until a slice, or part of a slice, has been worked. The floor of the slice is then covered with slabs or plank and the supporting timbers are shot out and the capping is allowed to cave on to the timber floor. This floor and the broken posts form what is called a mat, which keeps ore and capping separated. The miners then start in the raise and work out another slice of the ore just under the previous one and catch up the timber mat with posts or square sets. When this slice is completed another floor is laid and the supports again blasted. In this manner the orebody is worked in successive slices from the top downward. The broken ore from each slice is run to the raise in wheelbarrows or cars and dropped to the haulage level. The capping caves and follows down on top of the mat.

In this method the ore is not caved at all, but the ground above the ore does cave, and the surrounding country will cave more or less according to the amount of ore removed. If there are any other orebodies in the region affected by the cave, they will be lost or their recovery made more difficult. This is one of the reasons which may prohibit the use of caving methods. Also, unless the deposit is small compared to the distance to the surface, surface subsidence will take place and the surface must be free from buildings or roads. Top slicing is adaptable to heavy ores that can be mined easily, and require heavy timbering and filling if worked by overhand stoping. Deposits of large extent, over which the overburden will cave readily, are customary conditions. In some cases, top slicing may require as much or even more timber than an overhand stope with square sets, but for very heavy ore, which if worked overhand necessitates strong sets, reinforced and braced, it takes less timber, and in any case poorer timber may be used.

The following are the advantages and disadvantages of top slicing, chiefly as compared with overhand square-set work or with other methods of caving.

Advantages:

- Cheaper timber, and for heavy ores less timber, is required than for square-set stoping.
- No waste filling is necessary.
- Ore can be sorted as mined.
- Very little waste from the capping becomes mixed with the ore.
- Less skilled labor is required.
- Any rich fine ore produced in breaking will be saved on the slice below.
- Complete extraction of the ore is possible.

Disadvantages:

- Caving may cause injury to surface structures or render unworkable other orebodies in the region.
- Mining the ore by breast stoping requires more drilling and blasting.
- Broken ore on the slice must be shoveled into barrows or cars and taken to the raises.
- Development must be kept ahead of the demand for ore.
- It is difficult to leave bodies of low-grade ore which may be found in the deposit.
- Ventilation is difficult, and in some cases, the decaying timbers in the mat give off heat and obnoxious gases.

It is possible to work only on the top of the ore, and although several slices may be worked at the same time, it is not possible to establish work at different levels. This may make it difficult to obtain a required tonnage from a deposit of small horizontal section. Capping should cave readily, and not arch, or the method may become very dangerous.

Sublevel Caving

This method, otherwise referred to as sublevel drifts and back caving, or sublevel slicing, resembles top slicing in that the orebody is worked from the top down, and the ore is taken out in horizontal slices. A block

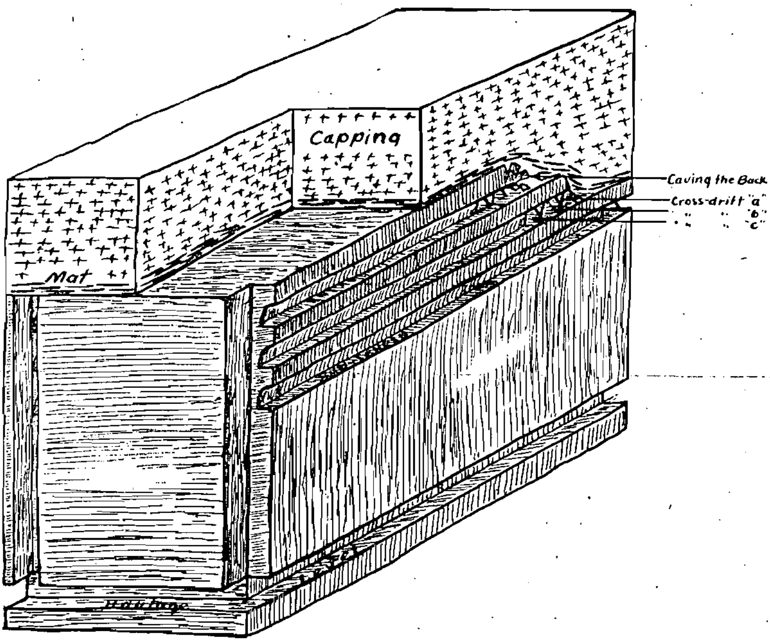


Fig. 6.—Sublevel Caving.

of ore is left above the slice, however, and when a small portion of the slice has been mined out, this back is allowed to cave. Fig. 6 shows a block or panel of ore worked by sublevel caving. Raises are put up from the haulage level, and from these sublevel drifts are driven 14 to 20 ft. apart vertically. When these drifts reach the boundary of the property, or of the panel to be worked, cross drifts as *a*, *b*, and *c* are driven across the block and timbered. The back of ore is thus undercut but is supported by the drift timbers. The timbers of cross drift *a* are next blasted and the weight of the capping caves the back of ore down on to the floor of the slice, where the miners shovel it into cars and push it to the raise. A timber mat may be used as in top slicing. Sublevel caving is adaptable to massive deposits or wide veins in which the ore is not difficult to break, yet is firm enough to hold the capping while supported by the drift timbers. An overburden that will readily cave is necessary.

Advantages:

The cost of mining is low, because a large percentage of the ore is broken with little or no blasting, and the amount of timber used is small.

A large output is possible.

A large percentage of the ore can be saved.

The ore can be kept freer of waste than in block caving.

Disadvantages:

It is limited in application to certain ores.

A large amount of development is required which must be kept ahead of the demand for ore.

A large part of the ore must be shoveled.

Caving endangers surface and other ore deposits.

There is some danger to the miners.

Ventilation is difficult.

Block Caving

Block caving is an extreme case of sublevel caving in which instead of a back of ore 5 to 10 ft. thick, one 50 ft. thick is undermined and allowed to cave. The method is illustrated in Fig. 7. After the bottom of the block, or panel, is cut up into pillars by drifts and cross drifts, the pillars are robbed, and then the remaining stumps are blasted out with one large blast. This allows the entire block above to cave. In settling it disintegrates so that it can be shoveled with very little additional blasting. For a block to cave it is usually necessary for it to be freed on one or more sides. This is done, as shown in the cut, by narrow stopes, called isolating stopes. After the ore has settled for from two to six months timbered drifts are driven through it. The broken ore is allowed to run into these drifts, starting farthest from the shaft, is shoveled into cars and trammed out. As soon as capping shows at any point shoveling is stopped.

The advantages of this method are:

The cost of mining is low because very little drilling and blasting, or timbering is required.

The amount of development is small.

A large output is possible.

Disadvantages:

The method is limited in application to low-grade ore of such character that it will cave and disintegrate.

A large amount of ore becomes mixed with capping and lost.
There is no opportunity to select or sort the ore.

Back Caving into Chutes

There is another principle that should be mentioned under mining methods. It is called back caving into chutes or chute caving and in some ways resembles block caving. (The description of a mine employing this principle is given in *Mining Without Timber*, by Brinsmade, p. 181.) Large overhand stopes are worked, not by drilling and blasting the entire back, but by blasting out narrow isolating stopes around the edges of the

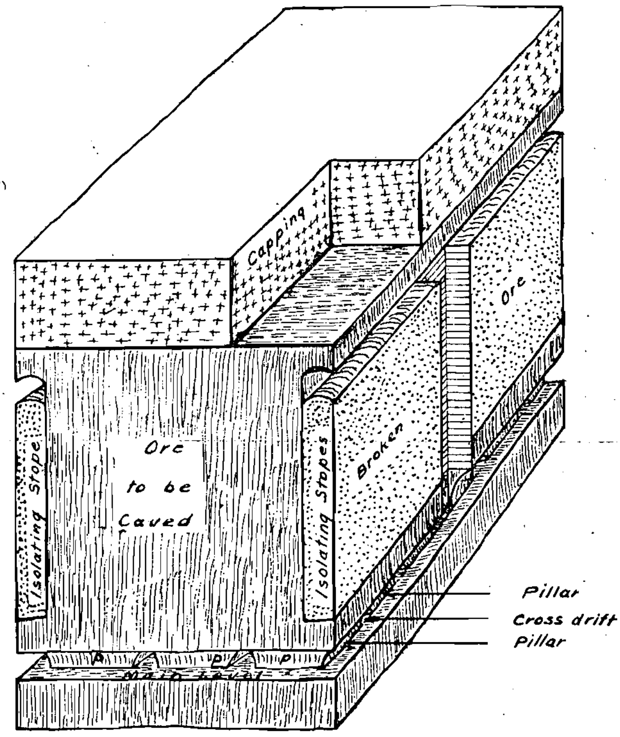


Fig. 7.—Block Caving.

main stopes and allowing the rest of the ore in the center to fall of its own weight. That is, a large percentage of the ore is mined by caving. The broken ore is drawn off through chute raises in the bottom of the stope.

As far as I am aware, the preceding pages cover every method of mining that has a distinctive feature and can be readily classified. There are innumerable different systems of mining but they all involve only these principles, or modifications and combinations of them.

Costs of Mining Methods

To compare properly two different methods of mining, it is not wise to attempt to calculate the total cost of each, but rather to compare the

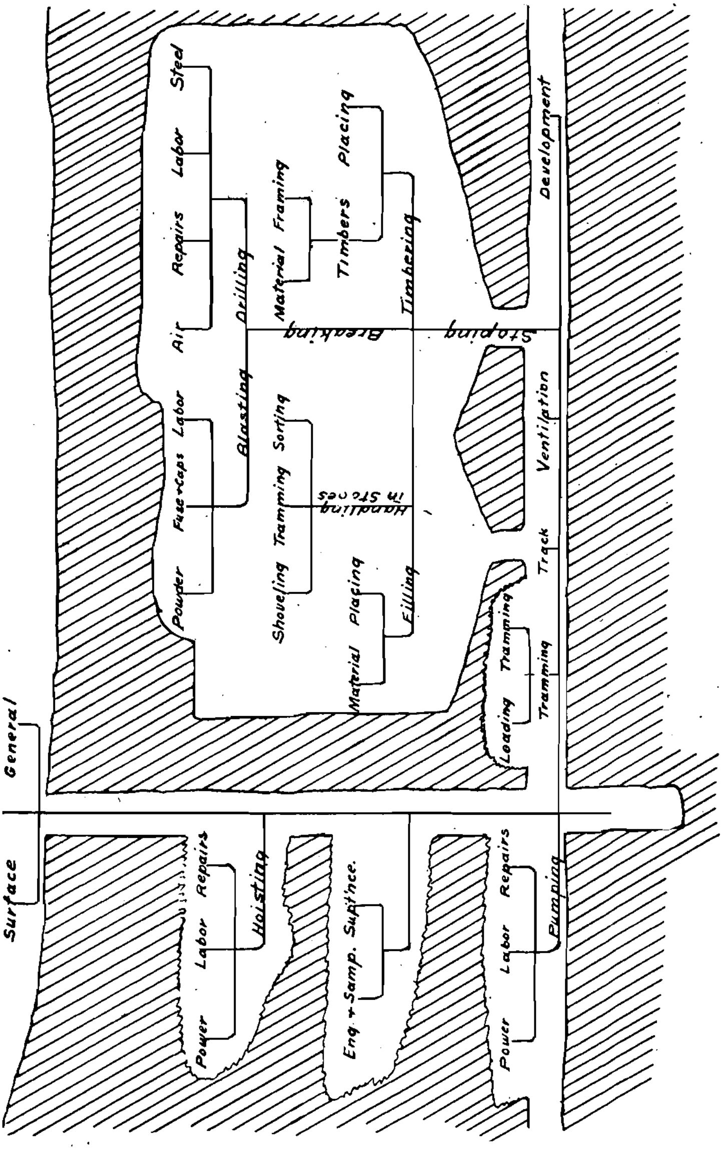


Fig. 8.—Diagram showing Arrangement of Mining Operations.

costs of those operations that are differently executed in the two methods under consideration. When any method is in operation the proper way to reduce the total cost is to dissect it into its different operations and see if the cost of one or all of these cannot be reduced. Keen competition has forced large manufacturing and other industrial enterprises to give careful attention to cost accounting and efficiency engineering. This calls for the investigation of the smallest details; conditions are adjusted so that a workman will not lose time in going after a tool or in walking from one machine to another. An increase in the efficiency of a number of small operations means a corresponding decrease in the total cost of production. The reasons why efficiency engineering has been so generally adopted are two: First, it attacks its problem in a scientific manner and, second, it brings results in dollars and cents. Mine managers are not excusable from applying its principles just because the ore may be high enough in grade to pay dividends even under lax conditions.

The different operations of mining can be separated in different ways. Fig. 8 is a diagram which shows a logical arrangement and the smallest operations shown in it could be subdivided into yet smaller details *ad infinitum*. In determining upon a new method or in attempting to reduce the cost of one already established, these operations must be carefully studied. If modifying a method will increase the efficiency of one operation without affecting the rest, the problem is a simple one, but usually more than one operation is affected, some adversely, and some to advantage, in which case to determine which is the cheaper method it is necessary to apply to each operation its cost under the different conditions.

Costs of Drilling and Blasting. Comparison of Stoping Methods

—As an appendix to this article I have collected a number of costs of mining as published in the reports of mining companies.

Obviously but few of these reports show the costs of the operations given in Fig. 8, and in order to bring out their detailed costs it is necessary to make certain assumptions. To bring out the costs of the different operations of drilling and blasting, consider the data on North Star, Alaska-Treadwell, and the Rand as given in the appendix, with general data on Cripple Creek and the Michigan copper mines. All are straight overhand stoping methods with the exception of the Rand, and, although not so stated, I assume that these data refer to underhand stoping. The first consideration is drilling. Up to the present time for underhand work, mounted machines were the only ones available, but it is probable that in the future they will be largely replaced by light unmounted machines of the Jackhamer type. For overhand work, the hammer stoping drill, unmounted, has demonstrated its superiority in reduced labor cost, increased ease of handling, and greater drilling speed in rock that is not too hard. While actually at work machines will drill from 0.25 to 15 in. a minute, usually 1 to 3 in. At the North Star mine, evidently, 25 to 30 ft. are drilled for each machine shift. Light stoping drills are used and break $7\frac{1}{2}$ tons in a 4-ft. stope. In Cripple Creek the average of several mines is 60 ft. for each machine shift. Light stoping drills are used and the machine runner is using the machine about 6 out of every 8 hr. In a 5-ft. stope 14 $4\frac{1}{2}$-ft. holes will break about 15 tons. In the Lake Superior copper region, in the amygdaloid lodes, machines drill from 30 to 47 ft. a shift in different mines.[1] In the conglomerate, on account of more massive copper and higher quartz content, about 24 ft. for each machine shift is the average (9-hr. shift) and seldom is more than 28 ft. drilled. Holes in both cases are usually 8 ft. deep and a burden of 2 to 3 ft. is placed on them. In the conglomerate, although harder to drill, 54 tons are broken for each machine shift, whereas in the amygdaloid only about 38 tons are broken. The stopes are 10 to 30 ft. wide. At the Alaska-Treadwell each machine in the stopes drills 28 to 34 ft. and breaks $32\frac{1}{2}$ to 45 tons. Heavy piston drills are used and the stopes are about 60 ft. wide. On the Rand large machines break 7.2 tons in a 55-in. stope, 10 tons in a 6-ft. stope, $19\frac{1}{2}$ tons in a 7-ft. stope, and small machines break 3.7 tons in a 4-ft. stope. To arrive at the actual cost of drilling it is also necessary to consider the cost of compressed air, machine labor, repairs, and drill steel and sharpening. The cost of compressed air varies from \$0.40 to \$2.50 for each machine shift. It depends on the cost of power, size and efficiency of the compressor, size and efficiency of machine drill, and the conditions of the drill. The accompanying table taken from *Ore*

			Cost per 1,000 Cu. Ft. Free Air Compressed			Cost per Drill Shift.		
Style of Compressor	Maximum Capacity Cu. Ft. Free Air per Minute	Total Cost per H. P. Hour	Sea Level	5,000 Ft. Altitude	10,000 Ft. Altitude	Sea Level	5,000 Ft. Altitude	10,000 Ft. Altitude
Simple steam (non-condensing..........	200	2.2c.	5.9c.	5.3c.	4.8c.	\$2.07	\$2.22	\$2.40
Compound steam non-condensing).....	300	1.5	4.0	3.6	3.3	1.40	1.50	1.65
Simple steam (condensing)............	2,500	1.0	2.7	2.4	2.2	0.95	1.01	1.10
Compound steam (condensing)........	3,000	0.8	2.2	1.9	1.8	0.76	0.81	0.88

Table based on 8-hr. shift with coal at \$5 a ton.

[1] Claude L. Rice: *Engineering and Mining Journal*, vol. xciv, No. 5, p. 217 (Aug. 3, 1912).

Mining Methods, by W. R. Crane, gives the cost of operating a 3-in. drill in granite.

At the North Star mines machine men cost $3 a shift or 43c. a ton; machine power, 40c. a shift or 6c. a ton; repairs and lubrication, 42c. a shift or 6c. a ton; drill steel and sharpening, 75c. a shift or 10c. a ton; total for drilling, $4.57 a shift or 65c. a ton. To this must be added 25c. a ton for powder, 4c. for fuse and 1c. for caps, which makes a grand total of 95c. a ton. Allowing for rock left in the stopes this would be reduced about 15 per cent.

At Cripple Creek machine men cost $3.50 a shift or 24c. a ton; air, $2 a shift or 13c. a ton; repairs and lubrication, 25c. a shift or 2c. a ton; drill steel and sharpening, 50c. a shift or 3½c. a ton. Total for drilling, $6.25 a shift or 42½c. a ton. To this add 24c. a ton for powder, 3½c. for fuse, and 1c. for caps, which gives a grand total of 71c.

At the Alaska-Treadwell machine men and helpers average $6.85 a shift or 21c. a ton; air, 75c. a shift or 2c. a ton; repairs, 50c. a shift or 1½c. a ton; drill steel and sharpening, 45c. a shift or 1½c. a ton. Total for drilling, $8.55 or 26c. a ton. Adding 15c. for powder, 1c. for fuse, and ½c. for caps gives 42½c., to which must be added an additional 6c. for extra labor which includes powder men. This gives a grand total of 48½c.

The data given in the appendix on operations on the Rand are too incomplete to analyze as the others are analyzed. From other data which I have, I assume that the costs are about as follows: Machine labor, $4.50 a shift; air, 75c.; repairs, 50c.; drill steel and sharpening, 50c. This is a total of $6.25. An average of 12 tons were broken each shift, which gives a cost of 52c. a ton for drilling. To this must be added 25c. a ton for explosives, which gives a grand total of 77c.

On the Rand, despite the greater stoping width, the cost of explosives is nearly as much as at North Star or Cripple Creek and the drilling cost per shift is much greater. The influence of the width of the stope on the tonnage broken is well illustrated in the Alaska-Treadwell and the added cost of machine labor is minimized. The long holes used and the character of the ore, no doubt, preclude the possibility of using one-man machines.

From the above, it is seen that an item worthy of consideration is that of drill steel and sharpening. It varies greatly but in any case is a factor worthy of attention. Not only the cost of new steel and the wages of the blacksmith, but the delivery of sharp drills to the stope and the removal of dull steel, should be considered. By comparing cost items occurring in the reports, which are given in the appendix, other features influencing drilling and blasting might be shown, but the greatest value results in making comparison when conditions are similar and are known with exactness.

Lighting

It is stated in one of the mining books that the cost of lighting for each ton of ore mined is about the same in all mines. This is not the case. The cost varies according to the light used and the number of men underground. The entire subject of mine lighting has been well covered by Frederick H. Morley in the *Mining and Scientific Press* of April 11, 1914. The following table taken from this article gives the cost of acetylene lighting in 10 of the large metal mines.

Name of Company	Number of Men Employed Underground	Number Using Lamps	Carbide Consumption per Lamp, Oz. Shift	Cost Carbide per Lb., Cents	Cost per Lamp Shift, Cents	Cost of Candles per Shift, Cents
Homestake Mining Co......	1,025	1,025	8.0	3.50	1.75	7.00
Ray Con. Copper Co.......	1,400	1,200	9.0	4.50[a]	2.50[a]	5.00[a]
Quincy Mining Co.........	1,389	575	6.7	3.50[a]	1.46[a]	....
Osceola Con. M. Co	625	625	6.0	3.50	1.38	....
United Verde Copper Co....	600	575	6.5	5.50	2.23	5.40
Bunker Hill & Sullivan Co..	460	208	7.0	5.25	2.30	6.18
Calumet & Arizona M. Co...	1,000	60	7.0	5.50	2.40	6.64
Ohio Copper M. Co.........	96	37	8.0	5.80	2.90	....
Nevada Con. Copper Co....	200	20	4.0	4.67	1.12	3.27
Mammoth Copper M. Co...	....	12	10.0	5.86	3.66	5.15
Average...............	....	....	7.22	4.76	2.17	5.52

[a]Estimated.

Timbering and Handling Ore in Stopes. Square-Set vs. Top-Slicing Methods.

There is often a question as to whether square setting or top slicing is more economical. For instance, in massive deposits of heavy sulphides the ore is easily drilled and breaks well and if overhand stoping with square sets is used the weight of the ore necessitates very heavy timbering. In order to compare the two methods and also to show cost data on timbering and handling ore in stopes, assume the methods applied to a block of ground 50 ft. square and assume that the development work is the same in each case. The sets to be 6 by 6 by 8 ft. In the top-slicing method a drift one set wide is started at the top of the raise and run to the boundary of the block and then across the end of the block. This will give 10 sets of drift. It will take about eight holes, each 4½ ft.

deep, to break a round, or 12 holes to break a set. All work in top slicing is breast work, and the drilling must be done with a drill mounted on a column. A mounted drill will drill slower and require more time for moving than a light, unmounted, stoping drill, which can be used in square-set mining. The cost of each set for the first 10 sets of drift in top slicing will be about as follows:

Drilling, labor, 2 shifts @ $3.50	$7.00
Drill repairs @ 0.30	0.60
Drill steel and sharpening @ 0.20	0.40
Air @ 1.00	2.00
Powder, 25 lb. @ 12½c.	3.00
Fuse, 60 ft. @ ⅔c.	0.40
Caps, 12 @ 1c.	0.12
Cost for each set	$13.52

Each set contains 288 cu. ft. or approximately 28 tons; this makes the cost of drilling and blasting 48c. a ton. The distance to tram the ore to the raise will average 50 ft. and the cost for mucking and tramming may be estimated as follows: A man will shovel this ore at the rate of about 2½ tons an hour, or 1 ton in 24 min. A wheelbarrow will hold about 1/7 ton and to wheel seven barrow loads 50 ft. will require from 10 to 15 min., say 11 min; this makes the time consumed in shoveling and wheeling 1 ton 35 min. or 22c. a ton if shovelers, wages are 37½c. an hour. This is equal to 14 tons handled for each 8-hr. shift and a cost for each set of $6. Exclusive of the raise there will be a total of 63 sets in a slice, of which 10 sets are mined for 70c. a ton (48c. for drilling and blasting plus 22c. for mucking). The remaining 53 sets have two free faces to break to. The cost of breaking will be 30 per cent. less or 35c. a ton. The average distance from the raise will be less, which will reduce the cost of shoveling, say to 20c., which gives a cost of 55c. for the remaining 53 sets or an average cost of 57c. a ton to mine the whole slice. On one slice there will be required 72 posts, 64 caps, and 72 girts. A post 8 by 8 in. by 8 ft. will contain 42 ft. B.M. and costs 84c. if lumber is figured at $20 per M. Caps and girts 8 by 8 in. by 6 ft. will cost 64c. each, figured on the same basis. Framing the posts will cost about 10c. each by hand or 5c. each by machines, say 6c. Framing the caps and girts will cost about the same. This gives:

72 posts @ 90¢	$64.80
136 caps and girts @ 70¢	95.20
	$160.00

which is $2.55 a set. The cost of taking the timber into the stope and placing it will amount to from $1 to $2 a set, say $1.50. Other timber for lagging and blocking will cost about 85c. a set, which gives a total for timbering of \$4.90 a set or 17c. a ton. This added to the cost of breaking and mucking gives 74c. a ton.

To mine this same block of ground with square sets, it would be necessary to divide it into two stopes four sets wide and eight sets long. The ore has been assumed to be heavy and it would be impossible to carry a wide stope. As soon as a stope is mined it should be filled. For drilling, a light stoping drill can be used which, under the assumed conditions, will drill easily 50 per cent. more footage than the machine on the slice. One-third less explosives per ton will be required, so the reduction in cost of breaking will be at least 30 per cent. or to 30c. a ton. A flooring of plank will have to be laid on the top set and about 75 per cent. of the broken ore will not fall into the chutes but must be shoveled. A shoveler will handle 3 tons an hour at a cost of 12½c. a ton for 75 per cent. of the ore. This is equivalent to 9c. for the total tonnage. We have, therefore, reduced the cost of breaking and shoveling from 57c. in top slicing to 39c. in overhand stoping with square sets, a saving of 18c. a ton.

Consider the timbering. It is not necessary to go again into detail but first assume that the ground can be held up by square sets of 10 by 10 in. timber. A post 10 by 10 in. by 10 ft. long contains 66 ft. B.M. and at \$20 per M. will cost \$1.32. Since it is heavy timber and framed on both ends, the framing will cost about 12c., making its total cost \$1.44. Caps and girts will cost \$1 each plus 8c. for framing, or \$1.08 total. This makes a total of \$3.60 for timbers in the square set instead of \$2.30 in the set used in top slicing, an increase of \$1.30. There will also be an added cost of placing of about 50c., which makes a total additional cost for timbering in the square-set method of \$1.80 per set or 6½c. per ton.

These figures show very plainly that the timbering used in square set must be very heavy to make the increased cost over top slicing greater than is the saving in breaking. Here the saving in the latter is 18c. but the increased cost of timber is only 6½c., leaving a balance of 11½c. in favor of square-set mining. This is in accord with experience, but when timbers will not support the ground and many braces are required, or filling must be resorted to, top slicing becomes cheaper. Filling will cost at least 20c. a ton and probably 50c. One mine that I have had in mind in preparing these figures has a cost of timbering and filling of more than \$1 a ton. At the Esperanza mine, Mexico, in 1907, 35 ft. B.M. of timber were used for each ton of ore, which at \$20 per M. is 70c., and that for timber alone. It must be remembered, however, that in ore that does not break easily the additional cost of breaking in the top-slice method becomes much greater.

I do not want to give the impression that these figures are to indicate the total cost of mining by the different methods. There are many other items such as superintendence and ventilation that enter into the cost of mining. There are also incidental expenses in breaking and timbering, such as repairing, timbering chutes, and the like, the cost of which will enter into the total cost of any method, and the amount can be judged only by experience. *The figures given bring out only the relative costs of different operations in different methods under assumed conditions.* In my opinion, a few figures are necessary to aid in determining the merits of different methods or in reducing the costs of methods in use. With the figures, experience and judgment must be added. These are probably more necessary in mining than in any other business, because there are so many variable conditions.

Tramming

The costs of tramming as given in published reports nearly always include not only tramming, but loading also, either from chutes or from the floor of a drift. The following interesting data are from the *Engineering and Mining Journal* of Mar. 8, 1913.

"At the Elkton Consolidated Mining & Milling Co., Cripple Creek, Colo., the 1911 cost of tramming was 14.6c. per car of approximately 0.7-ton capacity. The South Utah Mines & Smelters, Newhouse, Utah, reports its tramming cost for the year ended June 30, 1912, at 15.76c. per ton of ore, which evidently includes the cost of handling waste removed. In Goldfield, Nev., tramming has averaged about 18c. per ton of ore produced from stopes and has ranged from about 14 to 25c. These figures are for actual tons trammed and do not include any shoveling in stopes. At the North Star mine, Grass Valley, Calif., observations show that a man pushes an 18-cu. ft. car about 150 ft. per minute and shovels about 3 tons per hour from a plat into car against 2 tons when shoveling off a rock bottom. According to this a shoveler's efficiency is increased about 50 per cent. by using a plat. The Wolverine Co., Houghton, Mich., reports tramming costs at 17.4c. per ton of ore, and the Wettlaufer-Lorain, Cobalt, Ont., 21c. per ton of ore."

Data on Tramming

Mine	State	Size Car Used	No. of Men	Length of Tram Ft.	Amount Trammed per Man-Hr.
			Shoveling from Rock Bottom		
Erie Consolidated	Calif.	1¼ ton	2	1,000	1.17 ton
Erie Consolidated	Calif.	1¼ ton	1	1,000	1.6 ton
Pittsburg-Silver Peak	Nev.	1.1 ton	1	700	1.52 ton
Cananea Consolidated	Mex.	16.8 cu. ft.	1	300	35.8 cu. ft.
			Shoveling from Plat		
Erie Consolidated	Calif.	1.0 ton	1	100	1.75 ton
Pittsburg-Silver Peak	Nev.	1.1 ton	1	1,000	1.575 ton
Cananea Consolidated	Mex.	16.8 cu. ft.	1	300	42.0 cu. ft.
Ohio Copper	Utah	20 cu. ft.	1	100	41.0 cu. ft.
			Loading from Chute		
Erie Consolidated	Calif.	1¼ ton	1	1,500	3.12 tons
Pittsburg-Silver Peak	Nev.	1.1 ton	1	700	6.19 tons
Cananea Consolidated	Mex.	16.8 cu. ft.	1	300	84.2 cu. ft.
Ohio Copper	Utah	20.0 cu. ft.	1	150	206 cu. ft.
Mother Lode	B. C.	2.15 tons	1	450	8.4 tons[a]

[a] Tramming with horses and locomotives.

If a man shovels from a plat at the rate of $2\frac{1}{2}$ tons an hour and wages are $37\frac{1}{2}$c. an hour the cost of shoveling is 15c. a ton. If a 1,000-lb. car is being used he will fill it in 12 min. If the tramming distance is 1,000 ft. and the trammer walks 200 ft. a minute the trip will require 10 min., allowing 2 min. to dump, the tramming will take the same time and cost the same as the loading, that is 15c. a ton or a total of 30c. a ton. Three things are very evident: Tramming is an important item of cost in nearly every mine, the size of the car makes a decided difference, and the resistance to traction of the car itself and the grade and condition of the track are important factors. I remember seeing one stretch of track 1,000 ft. in length over which it took two men to push a car of 1,000 lb. capacity. Figuring as above this increased the cost of tramming 15c. a ton, and in fact it was even more because the two men could push the car but slowly.

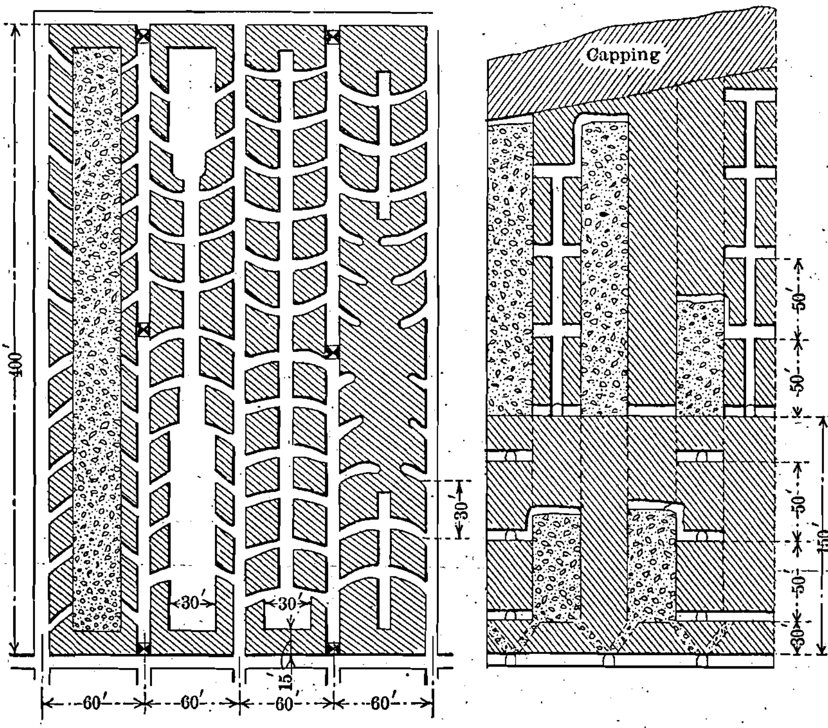
Henry Nagel, Superintendent of the Vindicator mine at Cripple Creek, has made some interesting observations in regard to tramming. He noted in one case that tramming on 800 ft. of level track cost 3c. a ton more than tramming on 800 ft. of track which had a grade of $\frac{1}{2}$ per cent. in favor of the loads. In another case a man who had been shoveling and tramming in a drift with bad air, did 50 per cent. more work when a good circulation of air was secured. In his opinion the greatest efficiency is secured from a trammer when the track grade is such that he can ride the car when going in loaded. As to the latter assertion a few figures may show how it would not be true under all conditions. A car weighing 500 lb. and holding 1,200 lb. of ore will weigh 1,850 lb. when a 150-lb. man is riding. If it has good bearings and has a resistance to traction of only 20 lb. to the ton it will run on a 1 per cent. grade. If it has very poor bearings and has a friction of 60 lb. to the ton it will require a 3 per cent. grade for coasting. On this grade the empty car returning weighs 500 lb. and would require one-fourth of 120 lb. or 30 lb. force to push it. This is too much for the average man. The car with the good bearings would take only a push of 10 lb. to bring it back up the grade, which shows the importance of good bearings, as well as of good track.

Mining of the Massive Porphyry Copper Deposits

The past decade has witnessed the development and successful operation of a half dozen or more immense deposits of low-grade copper ore that occur in the Western States. In every case sufficient capital was available to carry out the development of the mines in the manner decided upon, and it is interesting to note the different methods that were employed.

Utah Copper and Boston Consolidated Mines

The first two "porphyry coppers" were the Utah Copper, and Boston Consolidated, at Bingham, Utah. The Utah Copper started out with a modification of the chute-caving method described in Crane's *Ore Mining Methods*, p. 141. The method was not satisfactory and was



PLAN END VIEW OF STOPES

FIG. 9.—STOPING SYSTEM, BOSTON MINE.

discarded for stopes and pillars, although very little mining is done by this method because the ore will be handled by steam shovels. The Boston Consolidated mine, now called the Boston mine of the Utah Copper Co., was originally laid out into stopes and pillars. Fig. 9 illustrates the method.[2] Above the main haulageway there is a 30-ft. pillar of ore to protect it. Above this the deposit is divided into a series of vertical stopes 30 ft. wide and 150 ft. high alternating with vertical pillars of the same dimensions. Above these comes another and similar

[2] Figure taken from article by C. T. Rice in *Mines and Methods*, September, 1910, and copied in *Mining Without Timber*, p. 172.

series of stopes and pillars, but the pillars in the upper series come immediately above the stopes in the series below. The plan was to work all the stopes on both levels as shrinkage stopes. When the stopes were all completed drawing would commence from the bottom of the lower stopes. The upper pillars being over stopes below would settle and crumble so that they would pass down with the ore from the upper stopes when the lower pillars were weakened by blasting. It will be noted that in this method the capping must cave and follow down on top of the broken ore. Just what alterations in the plan of working would have been necessary if the method had remained in use it is difficult to state. The stopes are still being worked but only the excess ore is drawn off; the remaining ore is left to be mined by steam shovels after the capping is removed. The width of stopes has been reduced from 30 to 18 ft., and the pillars increased to 42 ft.

Ray Consolidated Mine

The Ray Consolidated Copper Co. at Ray, Ariz., is controlled by the same interests as the Utah Copper Co. and when their system of mining was laid out they no doubt profited by the earlier experience at Bingham. The orebody is flat and averages a little more than 100 ft. thick. The main haulageways are at the bottom of the ore and above them the ore is divided into vertical stopes 15 ft. wide leaving 10-ft. pillars between as shown in Fig. 10[3]. The stopes are worked as shrinkage stopes to the top of the orebody, then the bottoms of the pillars are blasted and drawing is commenced from under both stopes and pillars. The pillars, being undercut, settle and break up as the ore is drawn. By taking care to draw equally from all the chutes under a block of ground the capping will cave and follow down uniformly on top of the ore and there will be little mixing of the two. The Ray system, from all reports, has been eminently successful. Stopes 15 ft. in width are wide enough so that a large tonnage will break compared to the footage drilled, and all the advantage is had of the shrinkage method of stoping. After the ore in the stopes is broken, the pillars, which contain more than one-third of the total ore, are available for extraction with only a slight expense for drilling and blasting. The cost of mining for the last quarter for which reports have been published was 71c. a ton. This includes a proportion of all general and fixed charges, but does not include an allowance of $12\frac{1}{2}$c. a ton for the retirement of mine-development suspense account.

Miami Mine

The orebody of the Miami Copper Co. is much deeper than that at Ray. In horizontal cross-section it is roughly circular, about 1,000 ft.

[3] Abstract of article by L. A. Blackner, *Mining and Scientific Press*, Jan. 3, 1914.

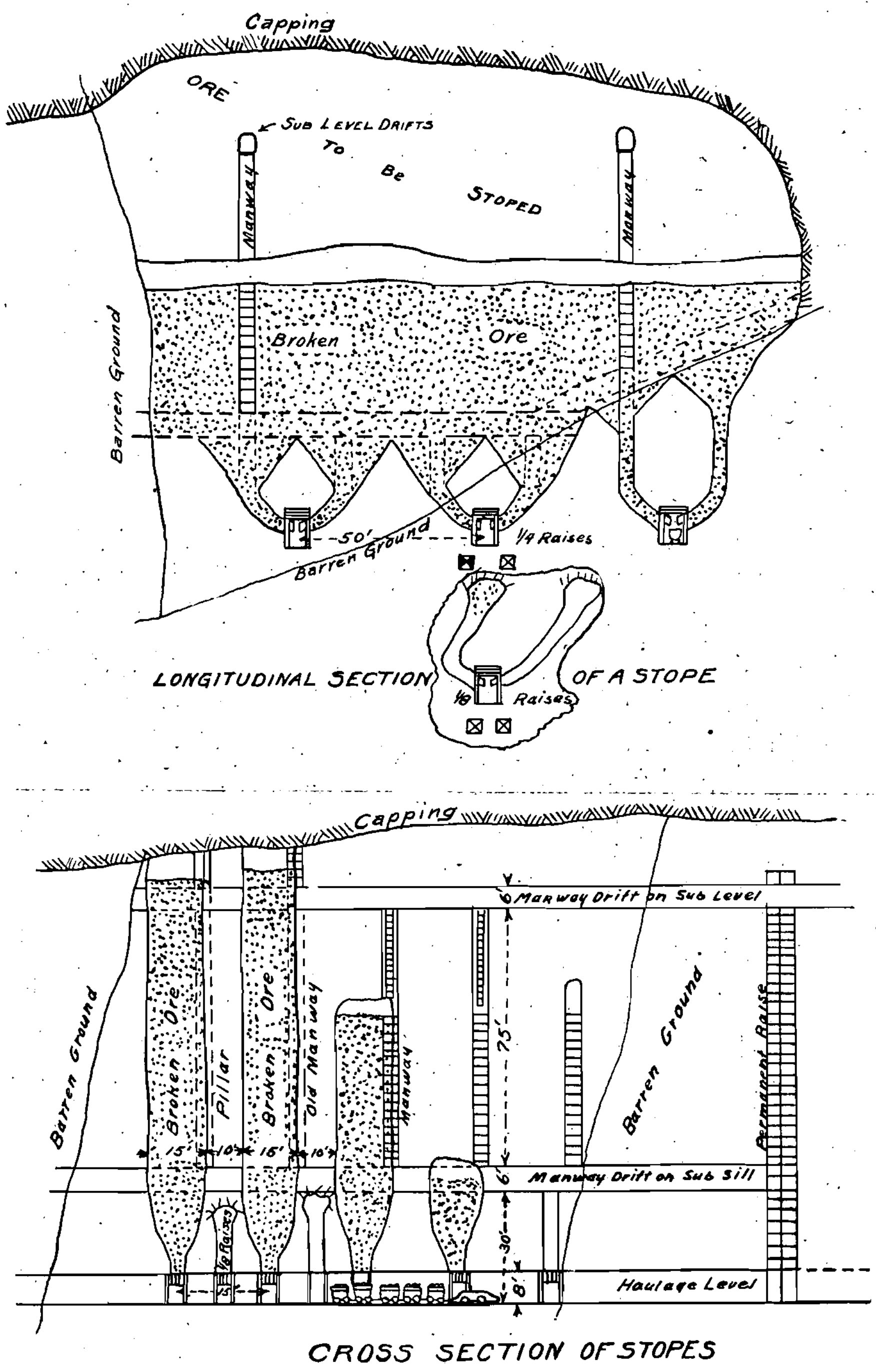


FIG. 10.—SECTIONS OF STOPES, RAY CONSOLIDATED MINE.

across. A tongue of waste rock intrudes into the ore from one side. The irregularities in the orebody under the capping are first worked by square-set stopes and timber is placed to form a mat between ore and waste. Through the main body of ore sublevels are spaced 25 ft. apart vertically as shown in Fig. 11[4] and on these sublevels drifts and cross-drifts are driven each way 50 ft. apart. The main body of the ore is to be mined with shrinkage stopes and pillars similar to the Ray method except that

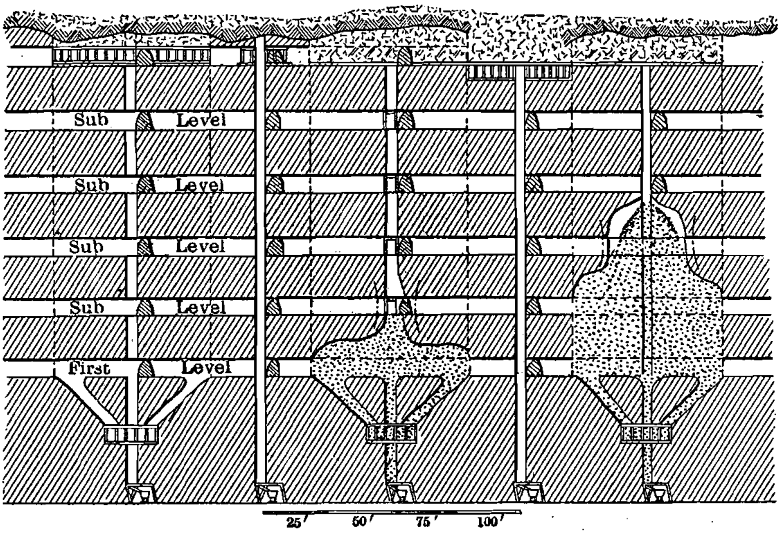


FIG. 11.—CROSS-SECTION OF STOPE, MIAMI MINE.

at Miami the stopes were planned to be 60 ft. wide and the pillars 40 ft. In working the stopes the miners do not work on top of the broken ore but approach the stopes through the drifts on the sublevels and break the ore into the stopes by the method of sublevel stoping previously described. The ore breaks readily into small pieces and obviously a 60-ft. shrinkage stope would be very dangerous if worked in the usual way, but by the sublevel method the danger is eliminated. Even under these conditions the last report states that it has been found advisable to make the stopes narrower. After the shrinkage stopes are carried up to the mat, which is next to the capping, the pillars are worked from the top down by top slicing. As this is done the ore from the stopes is drawn and a uniform settlement of the capping results.

The cost of mining at Miami has been about \$1.20 a ton. Last year a premature crushing of square sets under the capping started a cave

[4] R. L. Herrick in *Mines and Minerals*, July, 1910; *Mining Without Timber*, p. 167.

which extended through into the stopes below. The repair of this damage caused an increase in the cost for 1913. The Miami orebody is of higher grade than that at Ray and the Miami method will save a greater percentage of clean ore. The cost per ton is greater and there is a question as to how much the cost can be reduced in the future. To date, a large amount of ore has been taken from the square-set stopes, an expensive operation, but, on the other hand, in the future there will be an increased amount of ore to be mined from the pillars, which is also expensive. It would be interesting to know how much of the difference in cost between the methods at Ray and Miami is due to the difference in cost of labor. At Miami the miners receive $3.50 a day and up. At Ray mostly Mexicans are employed and they are paid, I believe, about $2.50 a day and up.

Inspiration Mine

The Inspiration mine, adjoining the Miami, has not yet begun to produce on a large scale, but it has been developed, in part at least, with the idea of using block caving. The method was described by Claude T. Rice in *Mines and Methods,* June, 1909, and is similar to the block caving illustrated earlier in this paper. The individual blocks were to be 75 by 200 by 200 ft. high and were to be isolated by stopes on the sides and breast stopes between ore and capping. The pillars under the blocks instead of being as shown in Fig. 7 were to be very narrow, 75 ft. long and 75 ft. high. Between the pillars narrow shrinkage stopes filled with ore would prevent the pillars from early crushing. This method, I believe, has never been given a trial and now is to be replaced by the method of block caving which has been so eminently successful at the Ohio Copper mine at Bingham. This method is worthy of a fuller description. The following is abstracted from a description of the method by Clarence G. Bamberger in the *Engineering and Mining Journal* of Apr. 6, 1912.

Ohio Copper Mine

"In *résumé* the conditions are: An orebody opened 400 ft. in width, 450 ft. in length, 1,300 ft. in depth, dipping from the horizontal at an angle of 50°. The entire mass being a broken shattered quartzite containing copper, chiefly in the form of chalcopyrite disseminated throughout, inclosed by foot and hanging wall of the same formation with boundaries not clearly defined.

"The preparation of this ground for a caving system of mining has been carried out as shown in Fig. 12. An incline shaft was sunk in the foot wall on the dip of the vein connecting with the different levels for the transportation of men and supplies. Two main ore chutes No. 1 and No. 2 were driven in the foot wall on an angle slightly greater than the dip of the vein. . . . . These chutes connect by diagonal raises through the foot wall, with the different levels where extraction is taking place and deliver into bins of large capacity at the Mascotte tunnel or main haulage adit which in turn leads to the concentrator situated at the portal about 3 miles distant.

"Lateral main levels were driven across the orebody at the 100-, 300-, 400-, 500-,

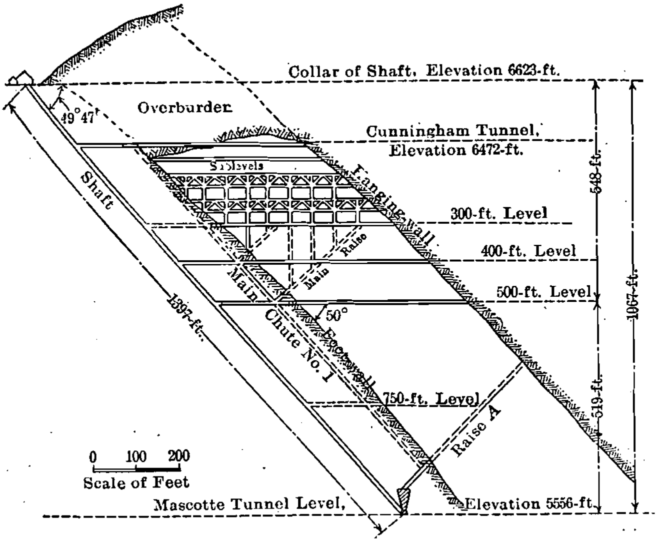


FIG. 12.—CROSS-SECTION, OHIO COPPER MINE.

750-, and 1,000-ft. or haulage level. By a network of crosscuts and drifts these different levels have been cut up into blocks 200 ft. square which in mining will again be subdivided. Between the several levels the ground has been blocked out into sublevels

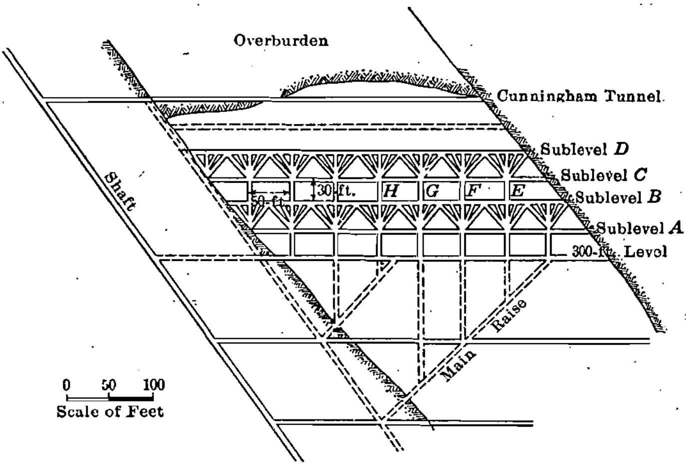


FIG. 13.—CROSS-SECTION, OHIO COPPER MINE.

30 ft. apart and again subdivided by crosscuts, drifts and raises into blocks approximately 30 x 50 x 25 ft.

"The actual procedure of extraction can be clearly understood by reference to Fig. 13 which shows in detail the sublevels above mentioned. Thus we have for example, the four sublevels, *A*, *B*, *C*, and *D*. Raises are driven from the crosscuts and drifts on the 300-ft. level through sublevels *A* and *B*. From these vertical raises at the various points *E*, *F*, *G*, and *H* raises are run into sublevel *C*, radiating like fingers from the palm of the outstretched hand. At the breast of these several raises sublevel *D* is blasted down, the ore falling by these leads into the cross inclined chutes connecting with the main ore chute, which delivers into bins at the loading station on the haulage-tunnel level.

"The same procedure is carried out on sublevel *B* which in turn is blasted down; at the same time sublevel *C*, which has already been cut up by this finger-like network of raises, comes with it as the solid ground below is blasted down. Thus each alternate sublevel is cut up by the numerous raises and the corresponding sublevel above and below is blasted down.

"The actual cost per ton for the different phases of the operation is shown in the accompanying table which is an average covering a period of 31 days in October, 1911, during which period 56,311 tons were mined.

Typical Daily Labor Report. Ohio Copper Co.

Description of Labor	Number	Rate	Amount
Superintendent	1	$11.66	$11.66
Foremen	2	5.33	10.66
Shift bosses	3	4.00	12.00
Timekeeper	1	3.00	3.00
Sampler	1	3.00	3.00
Mine engineer	1	5.00	5.00
Hoist engineer	2	3.25	6.50
Stationary engineer	2	3.50	7.00
Chainman	1	3.50	3.50
Tool sharpener	2	3.50	7.00
Machine men	21	3.25	68.25
Machine men	8	3.00	24.00
Muckers	23	2.50	57.50
Muckers	10	2.75	27.50
Loaders	2	3.25	6.50
Loaders	3	3.00	9.00
Chute tappers	3	3.00	9.00
Nippers	2	2.50	5.00
Blacksmith	1	4.25	4.25
Blacksmith helper	1	3.25	3.25
Carpenter	1	4.00	4.00
Carpenter's helper	1	3.25	3.25
Trackmen	1	3.00	3.00
Trackman's helper	1	2.50	2.50
Pipeman	1	3.25	3.25
Pipeman's helper	1	2.75	2.75
Timbermen	7	3.25	22.75
Timbermen's helpers	7	2.75	19.25
Total	110		$344.32

Total labor charge for Oct., 1911	$11,582.78
Total stores consumed for period	3,743.82
Total power consumed for period	476.00
Total operating expense for period	$15,802.60
Total development and equipment for period	3,429.00
Total actual operating expense for period	$12,373.60
Average number of feet driven per day	31
Average number of feet raised per day	42
Average tons mined per day	1,817
Total number of feet driven for period	961
Total number of feet raised for period	1,302
Total tonnage mined for period	56,311

All calculations based on net weight.

"From the above data several interesting conclusions can be drawn. Considering the whole working force of 110 men as producing the ore, approximately 17 tons of ore are delivered to bins per day per man. Considering, however, only the men who are actually breaking the ore, approximately 63 tons of ore are delivered to bins per day per man. From the "total operating expense for period" and "total tonnage mined for period," the cost per ton delivered to bins is shown to be 28.06c., which figure includes development and equipment charges. From "total actual operating expense" and "total tonnage mined for period," the actual cost per ton for mining delivered to bins is shown to be 21.97c."

This method of mining is similar to the ideal case of block caving already described, only instead of having to shovel the ore after it is caved a large number of branching raises are brought up from underneath and the ore runs into these and on down to bins at the bottom of the mine without any handling. The raises contain chutes and the ore is drawn evenly from a large area so that the capping will follow down without mixing too much with the ore. While considerable ore is lost the cheapness of the method is remarkable. I have in my possession the figures for April, 1913, almost two years after the above figures were taken, and the cost of mining including development and equipment is reduced from 28.1c. to 22.2c.

Development

The extensive development necessary for the successful operation of some methods of mining naturally brings to mind the question of what this work costs. In the appendix, I have placed a collection of figures on the cost of development as published in the reports of mining companies Easily driven drifts and crosscuts will cost $3.50 to $4.50 a foot. Average-sized drifts in hard rock with a small amount of timbering cost $6 to $7 and under unfavorable conditions the cost will be $10 or more a foot. In a 5 by 7 ft. drift costing $6.50 a foot the labor for drilling and blasting will be about \$2.15, labor for shoveling and tramming about \$1.20, air about 30c. a foot, explosives about \$1.25, and miscellaneous 60c. a foot. Raises are as a rule less expensive than drifts and, in very favorable ground, can be driven for \$1.50 to \$2.50 a foot. The average 4 by 5 ft. raise in hard rock will cost \$4 to \$5 and raises difficult to drive or requiring close timbering will cost \$8 a foot and up. Winzes are not driven often but when they are they cost from \$10 a foot up.

Space permits of a consideration of only a few methods from the point of view of development. At Kalgoorlie (J. Cheffirs, *Transactions Australian Institute of Mining Engineers*, vol. xiii) in a block of ore 500 ft. long and 200 ft. high (the distance between levels), rill stoping required four raises and shrinkage stoping required one; 800 ft. of raises cost about \$5 a foot or \$4,000. If we assume the vein to average 8 ft. wide a block of ore would contain about 60,000 tons, which means a cost for development of almost 7c. a ton. With shrinkage stoping this cost would be less than 2c. a ton. Of these methods touched upon in the preceding pages the Miami mine has the greatest footage of development work. The scheme of development allows a good many different methods of mining to be used without any changes; it also permits a large tonnage to be produced from a limited area. A block of ore 50 by 50 by 50 ft. contains approximately 50 ft. of raise and 200 ft. of drifts and crosscuts. Estimating raises at \$3 a foot and drifts at \$5 a foot gives a total of \$1,150 for the block. It contains about 10,000 tons of ore, which would make 11½c. a ton as the cost for development; this excludes the cost of shafts, stations, and similar requirements.

At Ray 12½c. a ton is allowed for the retirement of mine-development suspense account but just what this covers is not stated. It evidently covers more than development for stoping.

The method used at the Ohio Copper mine requires considerable development of which a large percentage is raising. Due to the fact that many of these raises were inclined, so that a miner could work at the face without using timber, and that all the ore, after being blasted, runs by gravity to the bins, the cost of the raises was only at the rate of 50 or 60c for each ton of ore mined in driving them.

Appendix

Mininy Costs

Montana-Tonopah Mine. (*Mining and Scientific Press*, Mar. 22, 1913)

Silicified and mineralized veins in altered andesite, 3 to 5 ft. wide, steeply pitching. Sometimes thrown over by faults through the stopes. Overhand stoping.

Year ended Aug. 31, 1912; 53,874 tons mined.

Labor:	
Ore breaking	$0.579
Mine machines	0.031
Hoisting and dumping	0.197
Boilers	0.034
Shoveling and sorting	0.675
Tramming	0.224
Timbering	0.209
Tool sharpening	0.026
Surveying	0.028
Foreman and bosses	0.078
Sampling	0.016
Storekeeper	0.011
Assaying	0.016
Watchman	0.011
Superintendence	0.046
Maintenance and repairs	0.001
Total labor	$2.182

Supplies:	
Water	$0.005
Ore breaking	0.292
Compressed air	0.111
Hoisting and dumping	0.059
Hoisting (elec. power)	0.129
Timbering	0.197
Total supplies	$0.793
Total mine cost	$2.975

Tonopah-Belmont. (*Engineering and Mining Journal*, May 11, 1912)

Year ended Feb. 29, 1912; 182,000 tons dry ore and waste; 152,550 tons dry crude; 115,560 tons dry, sorted.

Fracture zones in andesite and rhyolite-dacite, filled with quartz. Overhand stoping.

	Per Ton Dry, Sorted, Cents
Development:	
Miners	47.4
Muckers and trammers	22.2
Timbermen and helpers	8.4
	78.0
Stoping:	
Miners	44.5
Shovelers	33.9
Trammers	19.2
Timbermen and helpers	97.8
Filling	4.2
Piston-drill repairs and maintenance	5.0
Stoping-drill repairs and maintenance	2.9
Steel and sharpening	7.1
Explosives	28.6
Hoisting to surface	30.9
Auxiliary hoisting	9.4
Ore sorting and loading	27.3

Sampling and assaying	4.7	
Surveying	5.6	
Supt. and shift bosses	13.7	
Mine office	12.9	
Surface and plant	16.2	
Lighting	4.6	
Heating	4.1	
Drayage	6.1	
Maintenance and repairs to buildings	2.5	
Maintenance and repairs to machines and machine tools	2.3	
Maintenance and repairs to pipe lines and tanks	1.6	
Maintenance and repairs to railroad spurs	0.8	
Maintenance and repairs to pole lines	0.2	
Pumping	6.2	
Ventilation	1.7	
	394.0	
		$3.94
		0.78
		$4.72
Administration, taxes, safety, and depreciation		0.719
Grand total		$5.439

West End, Tonopah. (*Mining and Scientific Press,* Aug. 16, 1913)

Superintendent and foreman	$0.135
Breaking	0.802
Timbering	0.090
Tramming	0.372
Hoist, etc	0.200
Ore loading	0.233
Ore sorting	0.366
Assaying, sampling, surveying	0.091
Surface, ore dump, drayage	0.195
Development	0.862
General expense	0.554
Miscellaneous	0.262
Total	$4.162

Bunker Hill and Sullivan. (*Engineering and Mining Journal*, June 14, 1913)

Large replacement deposits of sulphide ores in quartzite. Overhand stoping with stulls or square sets.

	Labor	Supplies
Superintendents, blacksmiths, and supply men	$0.177	.....
Timbering	0.086	$0.215
Miners	0.367	.....
Carmen	0.052	.....
Shovelers	0.366	.....
Power	0.027	0.045
Repairs	0.026	.....
Explosives	.....	0.075
Illuminants	.....	0.020
Lubricants	.....	0.003
Iron and steel	.....	0.012
Miscellaneous supplies	.....	0.035
Wood	.....	0.034
Stable	.....	0.001
Total	$1.101	$0.440
Grand total $1.541		

Stewart Mining Co., Cœur d'Alêne. (*Mining and Scientific Press*, Apr. 26, 1913)

Cost of mining and development	$2.15

Ferreria Mine, Rand, South Africa. (*Mines and Minerals*, March, 1911)

Overhand rill stoping, shrinkage.

Tons for each machine shift	10.48
Pounds of explosives per ton, 57, cost	16c.
Total cost of stoping, 77c., plus 25c. for timbering	$1.02
Stoping on contract, per ton	55c.

Brakpan Mine, Johannesburg. (*Mining and Scientific Press*, May 17, 1913)

Presumably underhand stoping.

Mining:

Stoping	$1.11
Timbering and packing	0.24
Shoveling and tramming mine and dump	0.66
Transport, underground	0.11
Transport, surface	0.01
Hoisting	0.20
Pumping	0.16
Other charges	0.17
Development	0.36
Total	$3.02

Rand Mining Costs. (E. M. Weston: *Engineering and Mining Journal,* Feb. 8, 1913)

Presumably all underhand stoping.

New Kleinfontein Mine.—With hand drilling each native breaks 1.2 tons a shift at a cost of \$1.03. Of this amount the white labor cost \$0.16, the native labor nearly three times as much, while food, etc., for the latter costs 22c. to 24c. per ton. Owing to easy nature of ground, explosives cost only a little over 14c. a ton. The average footage drilled per native each shift is 40 in. and the stoping width 4½ ft.

At the same mine large machines broke 10 tons a shift in a 70-in. stope at a cost of 97c. a ton broken; of this 97c., white drillers cost 23c., natives 11½ c., while their food, etc., cost 30c. Explosives cost 21c. a ton.

One of the Deep-Level Mines.—Hammer boys drilled 33 in. per shift and broke ½ ton in a 44-in. vein at a cost of \$2.73 a ton. Explosives cost 30.3c. a ton. Large machines broke 7.2 tons a shift in a 55-in. vein at a cost of \$1.87 a ton. Explosives cost 36.3c. a ton. Small machines broke 3.7 tons in a 4-ft. vein at a cost of \$2.43 a ton. Explosives cost 40.5c. a ton.

One of the Large Outcrop Mines of the Central Rand.—Each native drilled 48 in. and broke 1.5 tons a shift in a 63-in. vein at a cost of \$1.22 a ton. Of this explosives cost 18.2c. Large machines broke 19.5 tons a shift in an 83-in. vein at a cost of 77c. Of this, explosives cost 20.2c.

The above costs are for stoping only. The total mining costs ran from \$2.25 to \$5 a ton.

Portland Gold Mining Co., Cripple Creek, Colo.

Overhand stoping with stulls and square sets.

	Per Ton Broken	
Tramming	\$0.17	
Hoisting	0.10	
Sorting	0.14	equal to 30c. per ton of ore through ore house.
Assaying, engineering, superintendence, and general	0.30	
Stoping	1.63	
Total	\$2.43	

About 44 per cent. of total ore broken is trammed; 29 per cent. of ore trammed is sorted, equal to 12.7 per cent. of total.

The costs for the year 1906 at the Portland mine as given in *Ore Mining Methods*, by W. R. Crane, were as follows:

	Cost per Ton
Labor	\$1.142
Machines	0.270
Tramming	0.029
Explosives	0.380
Hoisting	0.230
Supplies	0.036
Superintendence, assaying, etc.	0.450
Total	\$2.537

The labor cost was subdivided as follows:

Machine men	$0.4761
Trammers	0.3214
Pipe and track men	0.0357
Timbermen	0.1666
Timber helpers	0.1428
Total	$1.142

Temiskaming Mining Co., Cobalt, Canada. (*Engineering and Mining Journal*, May 25, 1912)

Narrow veins in schist. Overhand stoping.

Mining and timbering $1.85, of which labor was 66.5 per cent. and power 13.5 per cent. Dynamite cost 33.7c. a ton, fuse 2.6c., candles 3c., drill repairs 21c., steel 14.7c.

Hollinger Mine, Porcupine, Canada. (*Mining and Scientific Press*, May 3, 1913)

Country rock, fine-grained compact schist. Veins 2 to 6 ft. wide, containing quartz in stringers and bunches. Overhand stoping.

General and superintendence	$0.179
Diamond drilling	0.027
Stoping and driving	1.969
Timbering stopes	0.219
Tramming	0.551
Drainage and pipes	0.092
Hoisting	0.170
Dumping	0.063
Drill steel	0.298
Assaying, sampling, and surveying	0.064
Change house and lights	0.013
Handling explosives	0.025
Handling waste	0.016
	$3.686

In the Kalgoorlie district, Western Australia the ore occurs in strong veins in quartz-dolerite. The width of the veins varies but averages about 12 ft. The stoping methods employed are overhand rill stoping with filling, flat-back stoping, and shrinkage stoping. The following costs are of mines in this district.

Great Boulder Perseverance. (*Engineering and Mining Journal*, May 25, 1912)

Average stoping width 12.94 ft.

Wages and contracts	$0.92
Explosives	0.156
Drill parts and air lines	0.0434
Candles	0.0182
Air for drilling	0.1056
Not specified	0.4682
Total cost	$1.71

Ivanhoe Mine. (*Mining and Scientific Press,* May 24, 1913)

Ore breaking	$1.38
Filling stopes	0.27
Tramming and hoisting	0.58
Development	0.52

Kalgurli Mine. (*Mining and Scientific Press,* Mar. 8, 1913)

Labor:	
Superintendence	$0.04
Breaking ore	0.66
Timbering stopes and chutes	0.04
Loading and tramming	0.42
Filling stopes	0.16
Tool sharpening, etc	0.05
Sundries	0.02
Total labor	$1.39
Stores:	
Tools, steel, drill renewals	$0.03
Candles	0.02
Explosives	0.18
Timber	0.04
Assays	0.01
Sundries	0.01
Total stores	0.29
Hoisting, machine drills	0.24
Grand total	$1.92

Oriental Consolidated, Unsan, Korea. (*Mining and Scientific Press,* Apr. 30, 1910)

Mining timbers	$0.254
Firewood	0.133
Lumber	0.105
Dynamite	0.097
Candles	0.094
Fuse	0.094
Detonators	0.011
Charcoal	0.013
Lubricants	0.010
Drill steel	0.011
Bar, sheet, track iron	0.010
Picks, shovels, hammers	$0.009
Miscellaneous (rope, pipe, cars, etc.)	0.046
Total supplies	$0.823
Assays	0.006
White labor	0.117
Korean labor	0.612
Outside expense, shops, stables, etc	0.030
Total	$1.590

Consolidated Mercur Gold Mining Co., Mercur, Utah. (*Mining and Scientific Press*, Sept. 25, 1909)

Fifteen to 70 ft. vein of soft ore in hard cherty limestone. Dip 10° to 30°. Sublevel caving, with sublevels 14 ft. apart.

Total cost of mining $1.53.

Alaska Treadwell

From the costs of 1912:

Machine drillers	$0.143	Iron and steel	0.003
Laborers, powdermen, etc	0.128	Lumber and timber	0.003
Foremen	0.009	Compressed air	0.018
Blacksmiths	0.006	Power	0.007
Machinists, carpenters, timbermen	0.005	Mechanical repairs	0.002
Powder	0.146	Blacksmith shop	0.005
Fuse and caps	0.016	Miscellaneous	0.041
Candles	0.006	Total	$0.544
Machine drill supplies	0.006		

Cost of stoping only.

Anaconda Co., Butte, Mont.

Cost of mining by overhand stoping with square sets and filling was $3.77 in 1911.

Braden Copper Mine, South America. (*Mining and Scientific Press*, Dec. 4, 1909)

Ore breaking including superintendence and general charges was 41c. Machine drilling with $2\frac{3}{4}$-in. drill was at the rate of 31 ft. per man-day at a cost of 2.025c. a foot. Hand drilling was done at the rate of 13.8 ft. per man-day at a cost of 8.2c.

Cost of Development

Bunker Hill and Sullivan, Cœur d'Alêne. (*Engineering and Mining Journal*, June 14, 1913)

11,050 ft. of vertical and horizontal development:

	Per Foot	Per Cent.
Foremen, blacksmiths, etc	$0.312	4.4
Miners	2.500	35.0
Shovelers	1.650	23.1
Explosives	0.990	13.8
Timber and lagging	0.400	5.6
Power, labor and supplies	0.524	7.33
Not specified	0.774	10.8
Total	$7.15	100.0

Portland Gold Mining Co., Cripple Creek, Col.

Cost of drifts or levels, \$6.12; crosscuts, \$6.23; winzes and raises, \$8.60. In the *Cost of Mining* by Finlay, p. 380, the cost of 896 ft. of 5 by 7 ft. drifts is given as follows:

	Per Foot
Tramming	\$1.00
Pipe and trackmen	0.14
Machine men	1.88
Total labor	\$3.02
Other costs:	
Use of machines, air, etc	\$0.97
Repairs, cars, etc	0.08
Explosives	1.43
Hoisting	0.46
General expense, surveying, assaying, bosses	0.58
Grand total	\$6.20

Montana-Tonapah Mining Co. (*Mining and Scientific Press*, Oct. 9, 1909)

Drifting, \$6.56 a foot; crosscuts, \$5.44; raises, \$4.65; winzes, \$11.92. Cost of development subdivided as follows:

	Per Cent.
Labor:	
Breaking	27.10
Timbering	4.38
Hoisting and dumping	6.32
Foremen and shift bosses	1.75
Blacksmith, sharpening	1.75
Shoveling and tramming	22.90
Surveying	1.35
Watchman	0.52
Storekeeper and timekeeper	0.35
Diamond drill hole	4.38
Supplies:	
Breaking	21.20
Timbering	1.23
Hoisting and dumping	2.86
Hoisting, electric power	3.56

At a Western Gold Mine 302 ft. of development work took 238 miner's shifts, 122 shifts of muckers and trammers, and 105 machine-drill shifts. Explosives cost \$1.41 a foot and air for drills 25c. a foot or 75c. for each machine-drill shift.

At the Standard Mine, Bodie, Cal., drifts cost \$3.07 a ft. with an average of 1.3 ft. a shift, and raises cost the same. At the *Commercial mine at Bingham, Utah*, in 1912, drifts cost from \$5.83 to \$8.13 a ft., and raises cost \$5.56. At the *Nevada Hills mine, Fairview, Nev.*, drifts and crosscuts cost \$9.75 a foot, raises \$12.50, and winzes \$30. At the *Great Boulder Perseverance, Kalgoorlie*, drifts cost \$12.90 a foot, and crosscuts \$13.80. At the *Braden Copper Co., South America*, development cost \$3.54 a foot.

At Cananea in hard quartz a $4\frac{1}{2}$ by $6\frac{1}{2}$ ft. drift took 7.8 lb. of powder for each foot, and a 6 by 11 ft. raise took 8.3 lb. At the *Pittsburg-Silver Peak* a $4\frac{1}{2}$ by 5 ft. raise took 9 lb. powder for each foot, and at the *Erie Consolidated* in slate and quartz a 5 by 7 ft. raise took 6.65 lb. In drifts driven by hand work in medium-hard ground the cost was \$4.50 to \$5.50 a foot of which the labor was 77 to 85 per cent. and explosives 15 to 23 per cent. The rate of advance was 1 to $1\frac{1}{2}$ ft. a shift.