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MINING METHODS AT THE CAMPBELL MINE OF THE CALUMET & ARIZONA MINING CO., WARREN, ARIZ.

INCLINED CUT-AND-FILL AND SEMISHRINKAGE

BY

H. M. LAVENDER

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MINING METHODS AT THE CAMPBELL MINE OF THE CALUMET & ARIZONA MINING CO., WARREN, ARIZ.[1]

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INTRODUCTION

This paper describing the inclined cut-and-fill and the semishrinkage methods of mining at the Campbell mine of the Calumet and Arizona Mining Co. at Warren, Ariz., is one of a series being prepared by the United States Bureau of Mines on mining methods, practices, and costs in the various mining districts in the United States. The major portion of the production from the Bisbee mines of the company is mined by these methods.

The total average yearly production for the last two years has been 496,612 tons of 5 per cent copper ore. The average employment in the mining department was 940 men.

With the exception of a small tonnage mined for the high iron and sulphur content, the ore is direct-smelting and is mined for its copper content and associated gold and silver values.

ACKNOWLEDGMENTS

This opportunity is taken to acknowledge the aid and assistance given by D. M. Rait, superintendent of mines, Fred Juliff, general foreman, J. W. Fisher, division foreman, L. L. Walton, division foreman, and the other members of the engineering and mining departments without whose cooperation this article would not have been possible.

HISTORY OF THE DISTRICT AND EARLY DEVELOPMENT

The first location recorded in the Warren district was in 1877, but it was soon abandoned. In 1878 George Warren and associates relocated the same group of claims from which, in 1880, the first copper was produced. The original producing company was known as the Copper Queen Mining Co.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgement is used: "Reprinted from U.S. Bureau of Mines Information Circular 6289."

2 - Consulting engineer, U. S. Bureau of Mines. Chief engineer, Calumet and Arizona Mining Co.

In 1881 Dr. James Douglas formed the Atlanta Mining Co., which in 1884 was combined with the Copper Queen Mining Co. to form the Copper Queen Consolidated Mining Co. of which the Phelps Dodge Corporation gained control. This organization which has acquired several other claims and companies has been operated continuously in the Warren district from that date.

In 1898 the Lake Superior & Western Development Co. was organized and started work on the Irish Mag claim. In 1901 ore was encountered on the 750 level of the Irish Mag and in 1901 the Calumet & Arizona Mining Co. was formed to take over the stock of the Lake Superior and Western Development Co.

By means of consolidation and purchase the present Calumet and Arizona Mining Co. has acquired an acreage of approximately 2,091 acres on which they have operated continuously to date.

GEOLOGY

In the Warren district the fundamental rocks are pre-Cambrian schists, separated by an unconformity from the overlying beds of the Cambrian, Devonian, Carboniferous, and Cretaceous formations.

The Cambrian formation consists of a quartzite about 430 feet thick and a thin-bedded, cherty limestone, known locally as the Abrigo limestone, about 770 feet thick.

The Devonian formation consists of about 340 feet of dark-gray fossiliferous limestone known locally as the Martin limestone.

The Carboniferous formations consist of 700 feet of thick-bedded limestone known locally as Escabrosa limestone, and at least 3,000 feet of limestone known locally as Naco limestone.

At the close of the Carboniferous period there was an intrusion of granitic magma accompanied by very extensive faulting and folding of the formations of the district. During this period the limestones were tilted toward the east; the average dip is about 25°. The mineralization of this district probably dates from the time of this disturbance.

The so-called limestone ore of this district occurs in fissures and brecciated areas in the limestone. The fissuring and brecciation, with the porphyry intrusions, evidently had a marked influence on the mineralization. Commercial ore has been found in the Cambrian, Devonian, Escabrosa, and lower beds of the Naco formation, with the Escabrosa producing the major part of the tonnage.

In addition, large areas of both low and high grade ore has been found in the porphyry itself. Sacramento Hill is an example of this type of deposit.

The greatest depth at which ore has been produced to date in the Warren district is on the 2200 levels of the mines of both the Shattuck-Denn Corporation and the Calumet & Arizona Mining Co.

PHYSICAL CHARACTERISTICS OF THE ORES AND ENCLOSING ROCKS

In the Warren district as a whole the ore shows a marked variation in physical characteristics, ranging from the soft oxide and porphyry ores to the exceedingly dense and silicious sulphides. As this paper deals only with the stoping methods in use in the Campbell area, the discussion will be confined to the ore occurrences of that locality.

The ore body as developed on the 1600, 1700, and 1800 foot levels averages about 500 feet in length and varies in width from 50 feet to 250 feet.

The footwall contact is fairly well defined, with the exception of fracture zones which have caused some displacement. The dip shows a marked variation from nearly 90° to as flat as 25°. Above the 1700 level the footwall roughly conforms to the normal dip of the limestone beds of the district.

The hanging wall is subject to a greater degree of variation than the footwall and changes from an economic silica-pyrite contact of low copper content to a sharply defined hanging wall conforming to the local bedding planes.

The greater portion of the ore consists of a mixture of pyrite, silica, and limestone as the gangue with chalcopyrite and bornite, the main visible copper-bearing minerals. Along the limestone contact and fracture zones are found native copper, cuprite, chalcocite, and other secondary minerals.

In comparison with the other extensive ore bodies of the company, the Campbell ore body, as far as stoping is concerned, may be classified as a fairly homogeneous body of sulphide ore of rather high silica content which lends itself readily to the requirements of the stoping methods now being used. The stoping requirements may be briefly stated as follows:

1. The back must be able to sustain itself with a minimum of support, when properly arched and trimmed over the area being mined.

2. Extraction must be complete in the section being mined.

3. Homogeneous deposit which reduces to a minimum any selective stoping in the section being mined.

4. As nearly as possible the prompt and complete filling of all stopes or sections as completed.

The hard character of the ore makes heavy blasting necessary in order to prevent excessive formation of bowlders.

METHODS OF PROSPECTING AND EXPLORATION

Due to the depth at which the Campbell ore body is encountered, all prospecting and exploration, with the exception of a small amount of diamond drilling below the 2200-foot level, has been by means of drifts and raises.

The ore body itself was discovered as a result of the company policy of exploring areas in the favorable ore horizons. The early exploration work outlined the boundaries of the ore body. Subsequent development work in this area has largely conformed to an early established sectionalized plan by which this ore body was divided into the pillar and stope sections as they now exist.

METHODS OF SAMPLING AND ESTIMATION OF TONNAGES AND VALUES

Samples are taken from all faces of both drifts and raises in mineralized ground by shift bosses after each round. These samples may be obtained by picking from the face, by catching the drillings from a drill hole, or by taking grabs from the muck pile or cars. In the event ore is encountered the same procedure is followed, except that the motorman or trammer handling the cars of ore, samples each car from a working place and consolidates this sample. The samples so obtained are sent to the assay office daily and from the returns made the material is classified.

The same procedure is followed for stope samples. In doubtful areas an effort is made to obtain sufficient face and drilling samples to determine the character of the material before blasting. In case of uncertainty as to the results, channel samples are taken under the supervision of the engineering department. The interval chosen is usually 5 feet, due care being taken to have the samples represent sections at approximately right angles to the bedding of the material in question.

The assay returns from all working faces are recorded in the engineering office; these data are used in the estimation of tonnage. An assay correction factor, based on comparison of mine and smelter returns, is applied. A factor of 9 cubic feet per ton of sulphide ore in place and of 12 cubic feet per ton of oxide ore in place is used in calculating tonnage. These factors have been arrived at by a comparison over a long period with smelter tonnage returns and also by actual determination of the specific gravity of the different materials.

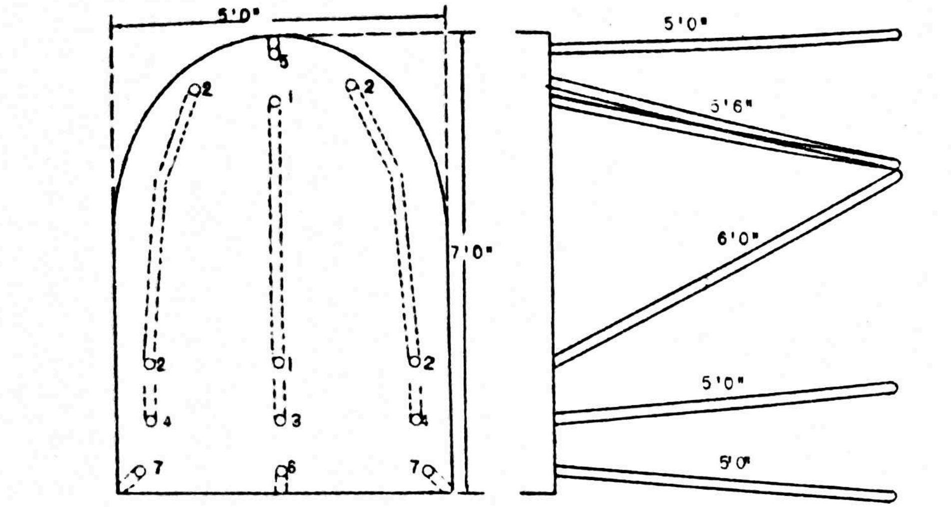
METHOD OF DEVELOPMENT AND MINING

Shafts

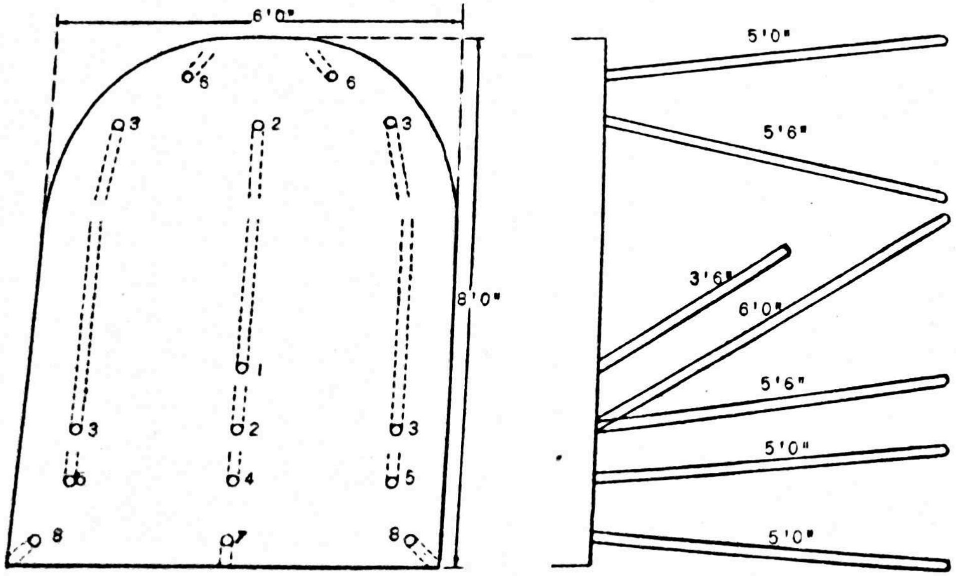
All shafts of the Calumet & Arizona Co. Bisbee mines are vertical. The Campbell ore body is served directly by two shafts. The Campbell shaft itself is used for supplies and ventilation. It is oval in shape and consists of two skip compartments, 5 feet by 4 feet 7-1/2 inches, and a large service compartment, 10 feet 8 inches by 5 feet 3 inches. The shaft is concreted to the 1600 level, timbered full size to the 2200-foot level, and is now being sunk to a depth of 2,350 feet. On completion of this operation the section from 1,600 to 2,350 feet will be concreted.

The Junction shaft is the main hoisting, ventilation, and pumping shaft. It is a 5-compartment concrete shaft from the surface to the 2,300 level.

The Campbell area is also connected to the Briggs and Cole shafts, which act as ventilation upcasts.



A.- Standard round, 5 by 7 foot drift



B.- Standard round, 6 by 8 foot motor drift

Figure 1.- Standard drift rounds in hard ground

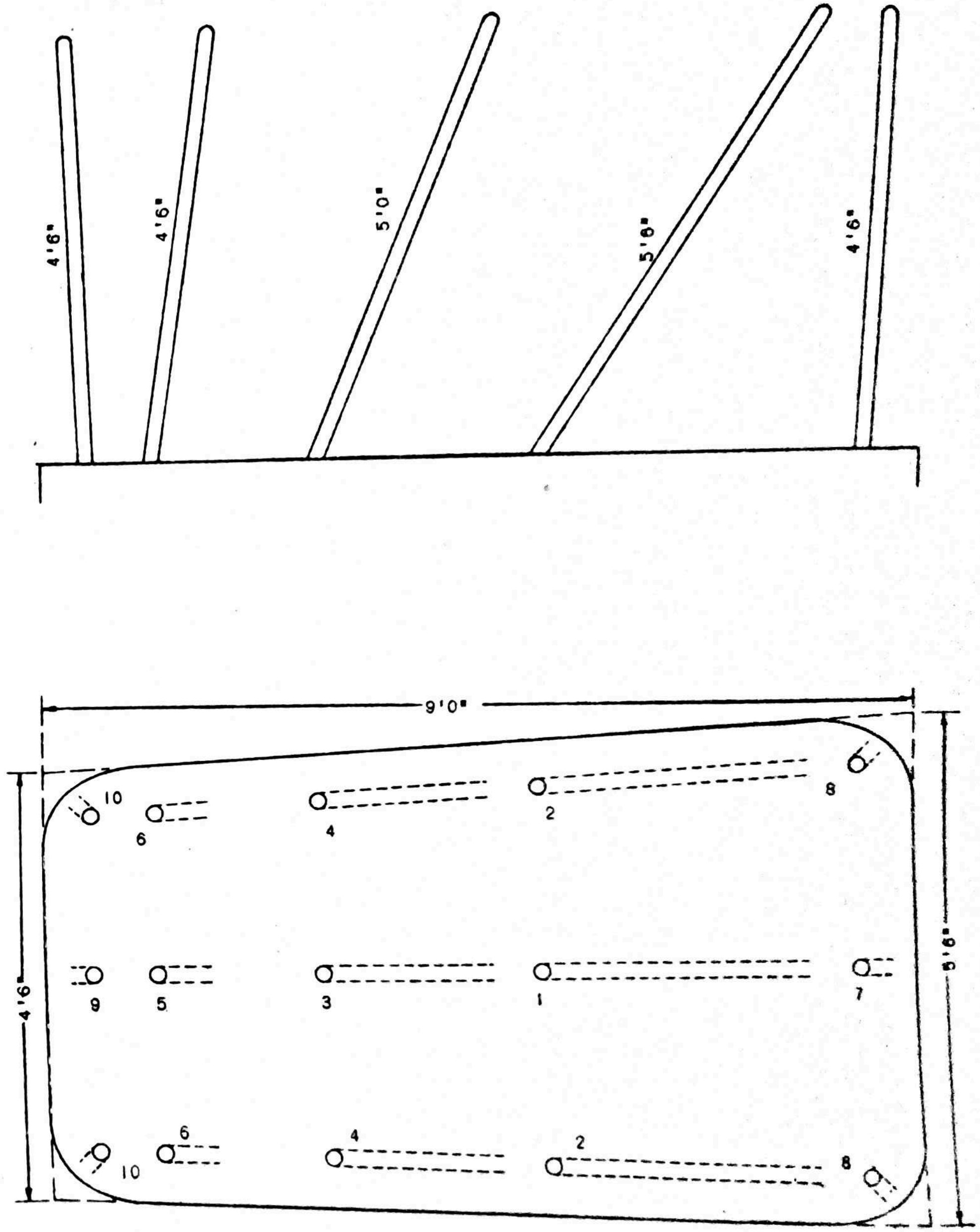


Figure 2.- Round used for stull raise, one end narrowed for wedging stull; numbers indicate firing order.

The Junction and Campbell shafts are approximately 2,300 feet apart and at the present time are connected on all levels from the 1300-foot level to the 2200-foot level, with the exception of the 1500 and 2100 foot levels.

Levels, Drifts, and Crosscuts

Levels are driven approximately 100 feet apart with stations cut at the same intervals. Motor haulage levels are established at every 200 feet. The even-numbered levels are the haulage levels, and the odd-numbered or intermediate levels are driven for prospecting, developing, and stoping.

Drifts and crosscuts are of two general classifications: A, Standard, averaging 5 by 7 feet in cross section; and B, motor, averaging 6 by 8 feet in cross section. The type of round drilled for each class is shown in Figure 1.

The ground in general may be classed as hard. Sixty per cent gelatin is generally used for blasting drift rounds.

Leyner-type drifters of two classes are used, the light type weighing about 125 pounds and the heavy type about 185 pounds. Drill columns and cross arms 3-1/2 inches in diameter are standard equipment, although on occasion a crossbar may be used. No mounted pluggers or hand-held pluggers are used in drifting in the Campbell area.

Hollow-round steel 1-1/4 inches in diameter, with the C&A double 5° and 14° taper bit is used in all drifting operations. Line oilers are also standard equipment.

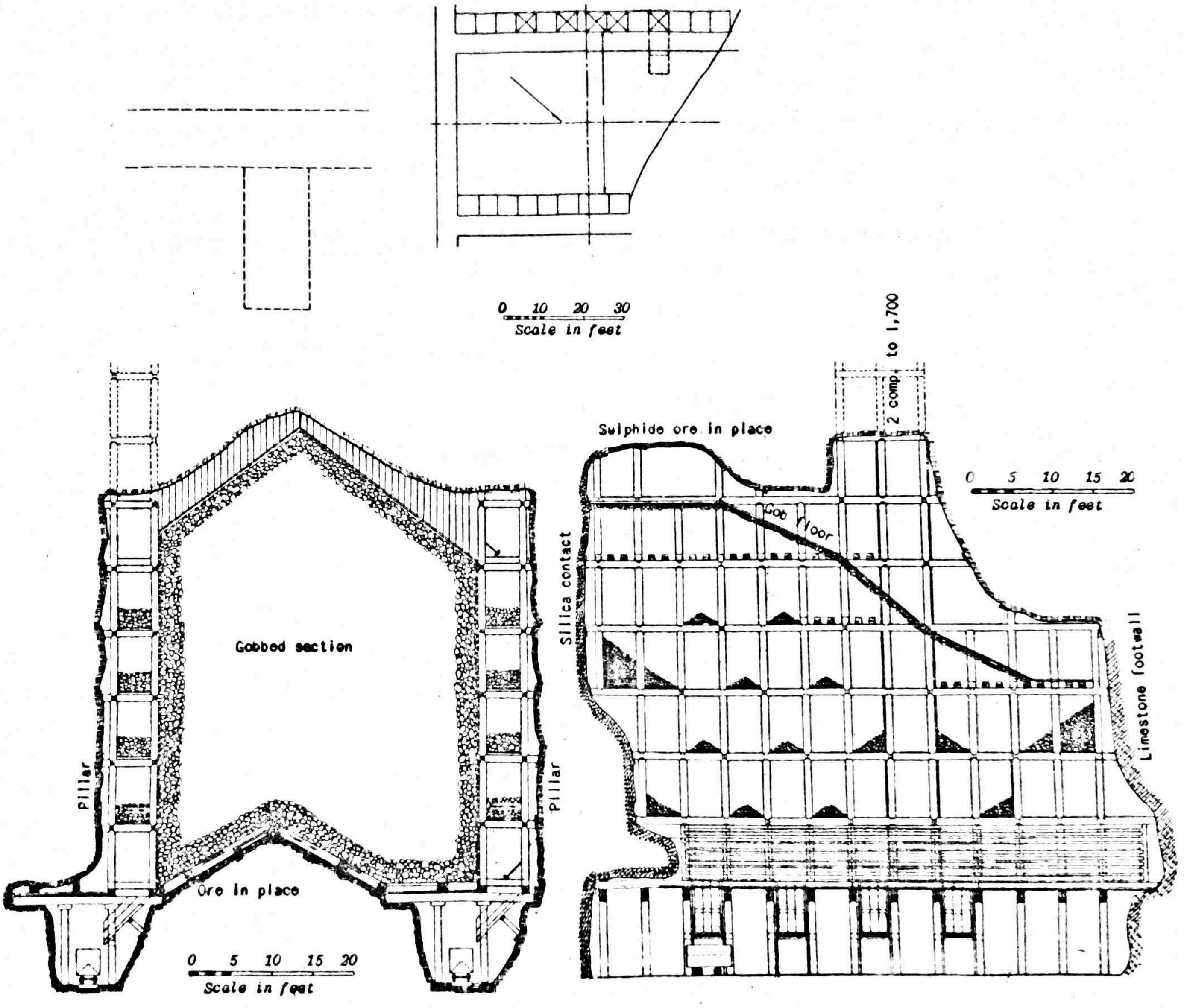
Raises

Raises are driven between levels for prospecting, developing, stoping, and ventilating purposes. Raises are of three general classes, depending upon the use to which the raise is to be put and upon the character of the ground, as follows:

1. Cribbed. Where ground conditions warrant and for extraction purposes, 4 inch, 6 inch, and 10 inch cribbing is used. Such raises are either single or of multiple compartments, each compartment being 4 feet 2 inches square in the clear.

2. Standard. Six or eight post raises are used principally for travel and supply purposes out of lead sets in stopes, and for main travel ways between levels.

3. Stull. This type is generally used where conditions permit. Cross-section and drilling practice are shown in Figure 2. Attention is called to the one narrowed end in which the manway is carried. The weight of broken rock in the chute tends to wedge the cross stull in place. This prevents stripping when blasting, due to the loosening of wedges.



Section A-A

Section B-B

Figure 3.- Inclined, double-lead set, cut-and-fill stope with raise in hanging wall

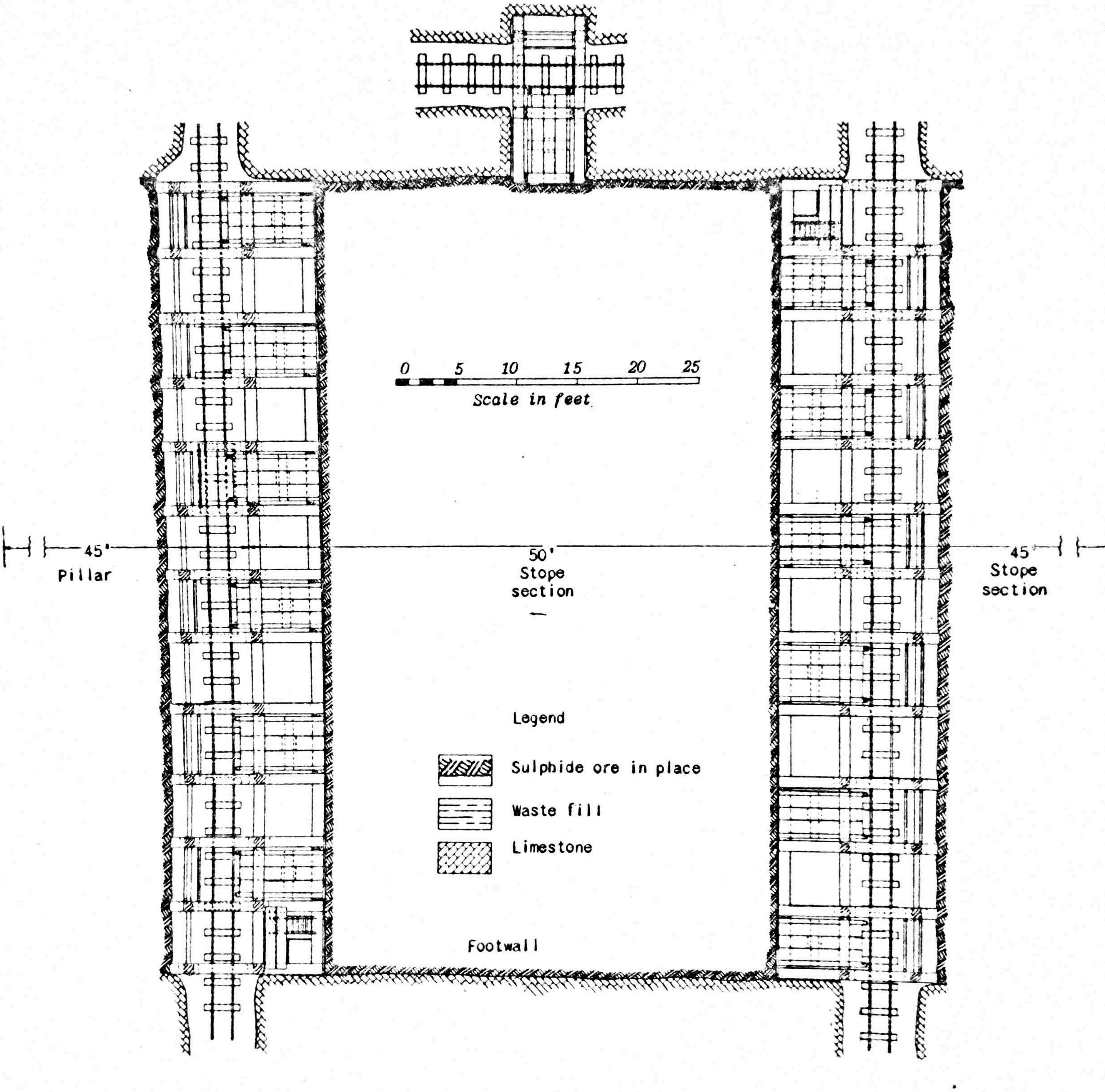


Figure 5.- Chute arrangement (plan on B-B, Fig. 4)

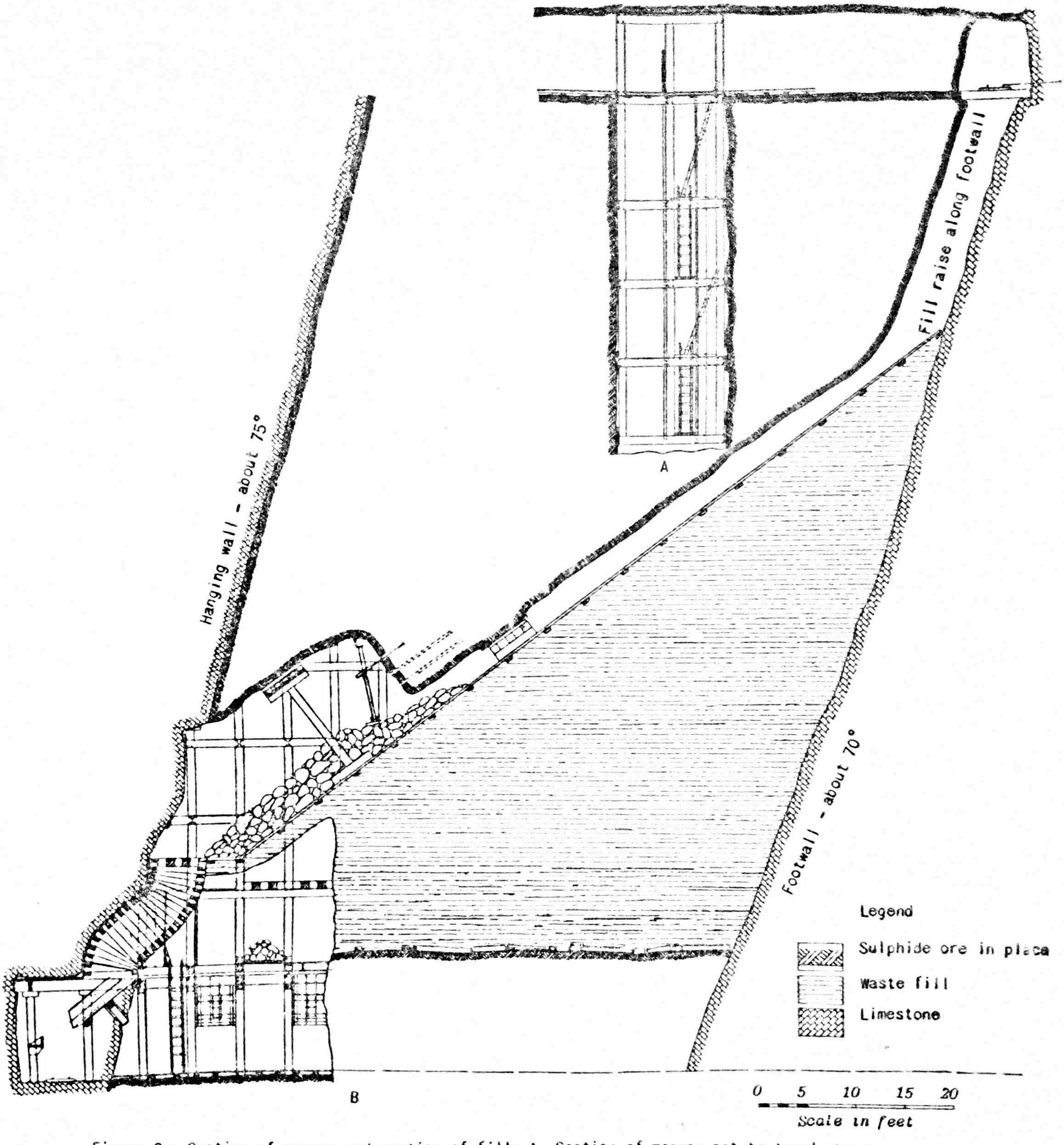


Figure 6.- Section of manway and section of fill: A, Section of manway set to level above projected on section C-C (Fig. 4); B, transverse section C-C (Fig. 4) showing section of fill and ground cut away

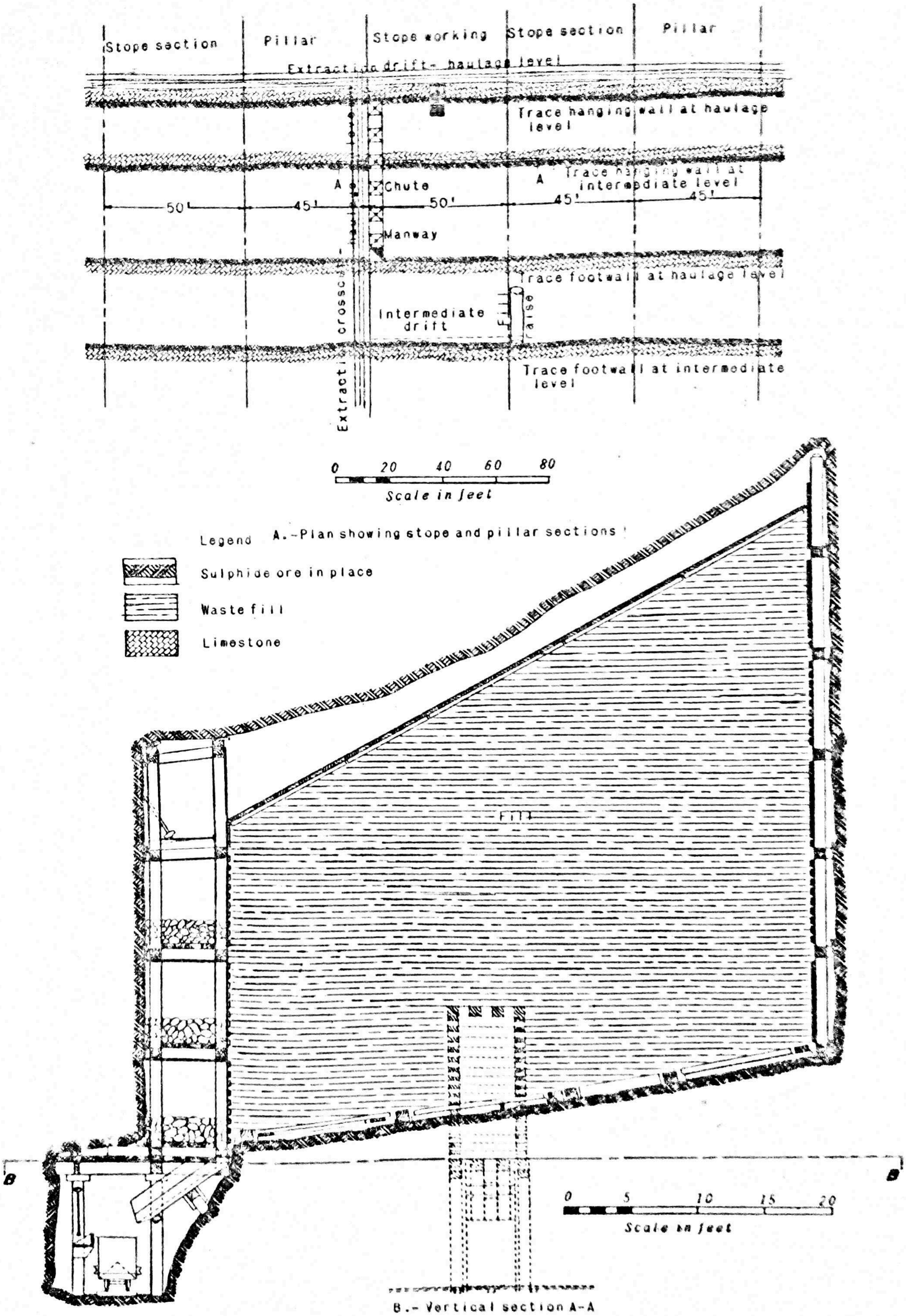


Figure 7.- Inclined, single-lead, single cap, cut-and-fill stope with fill raise in footwall

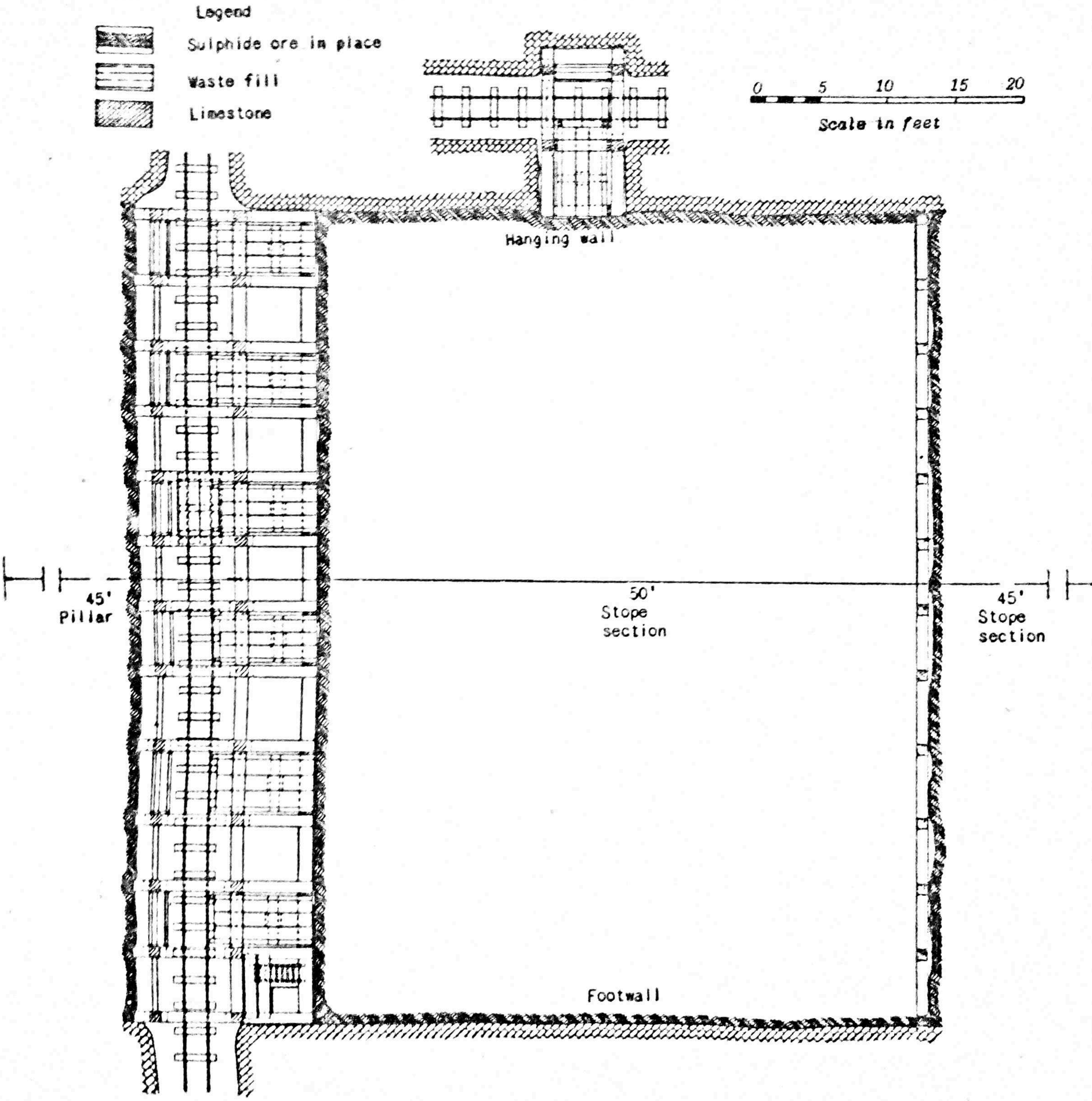
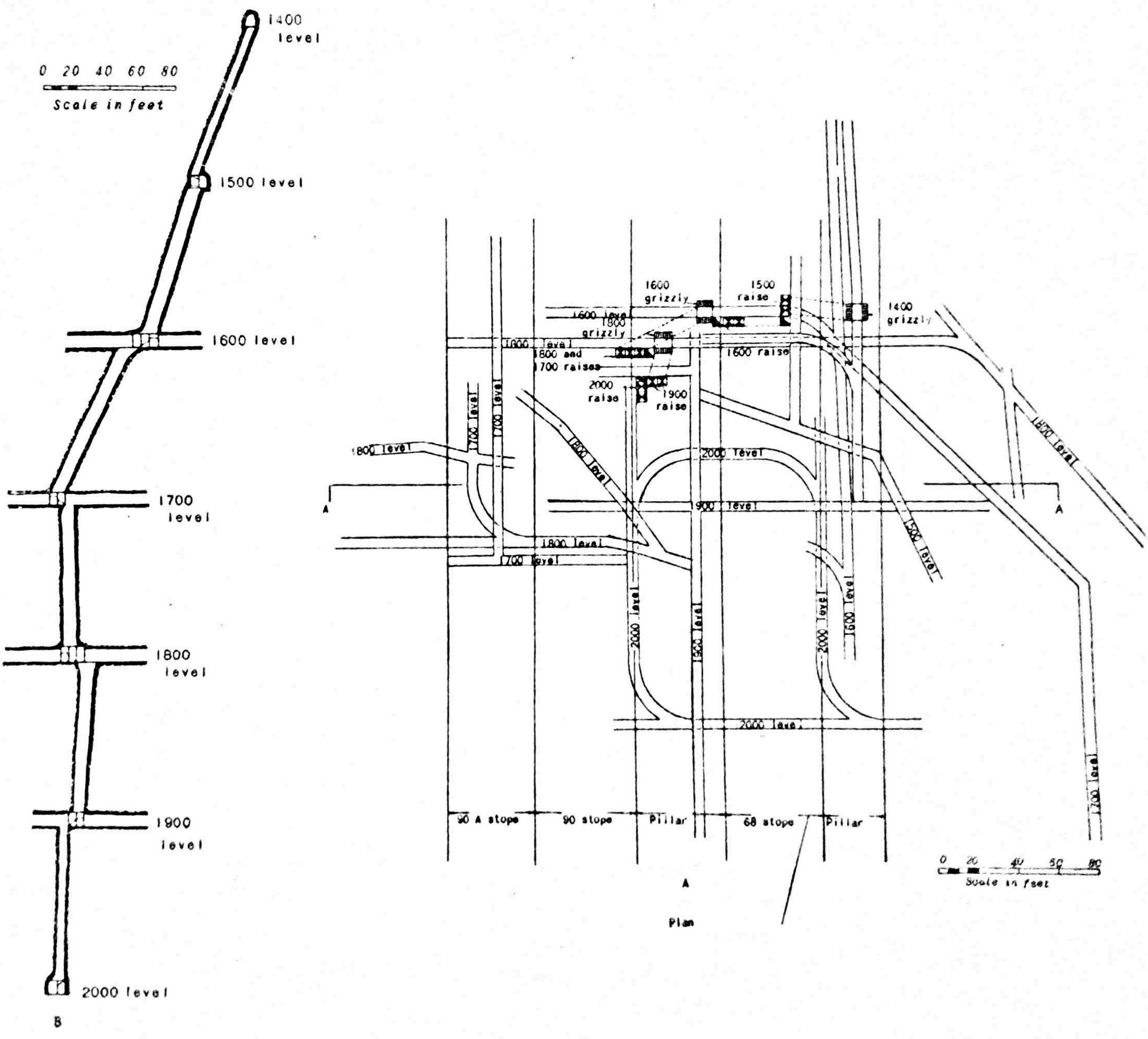


Figure 8.- Chute arrangement, (plan on B-B, Fig. 7)



Elevation of raises projected on A-A

Figure 9. - Waste storage raises

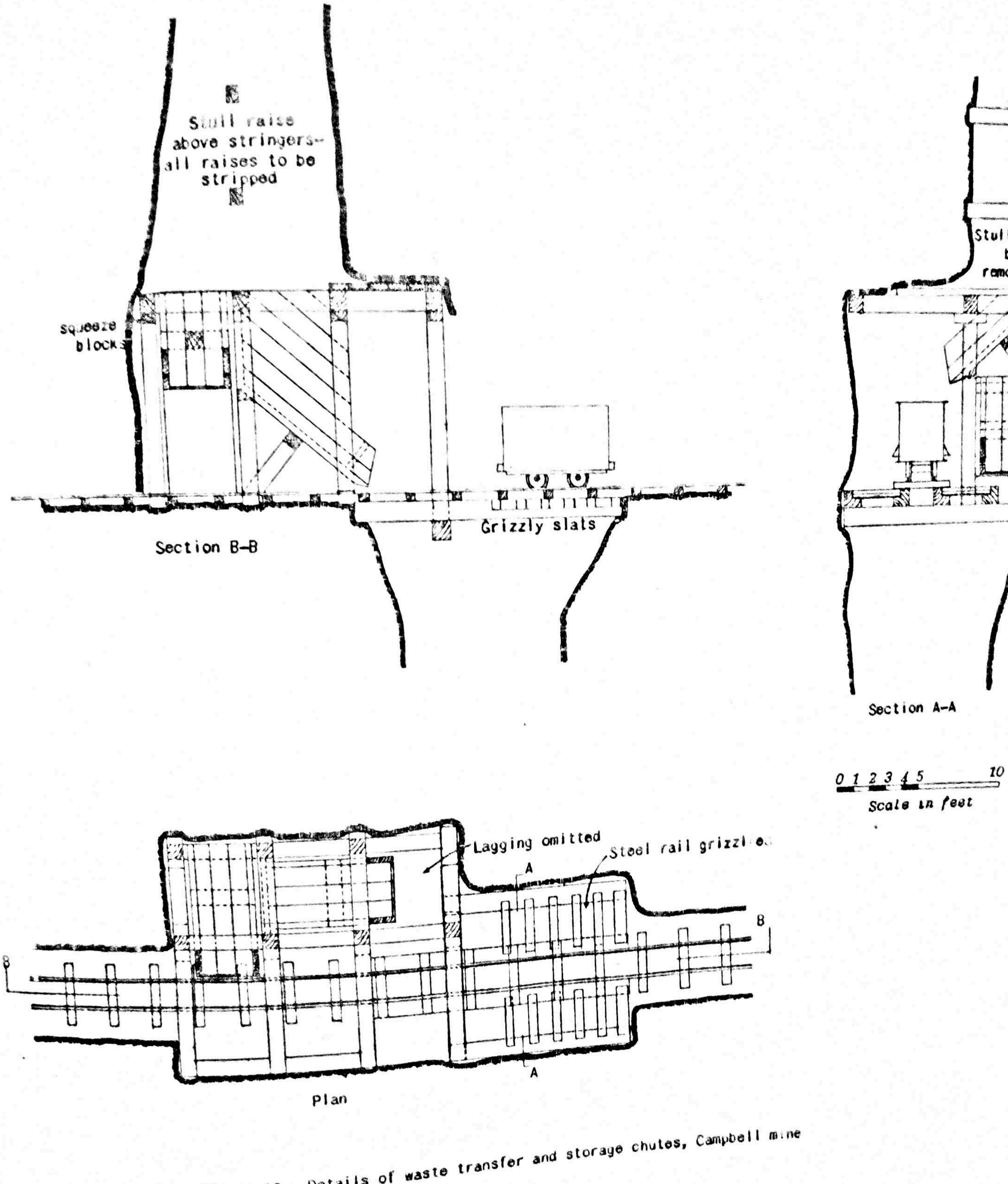


Figure 10.- Details of waste transfer and storage chutes, Campbell mine

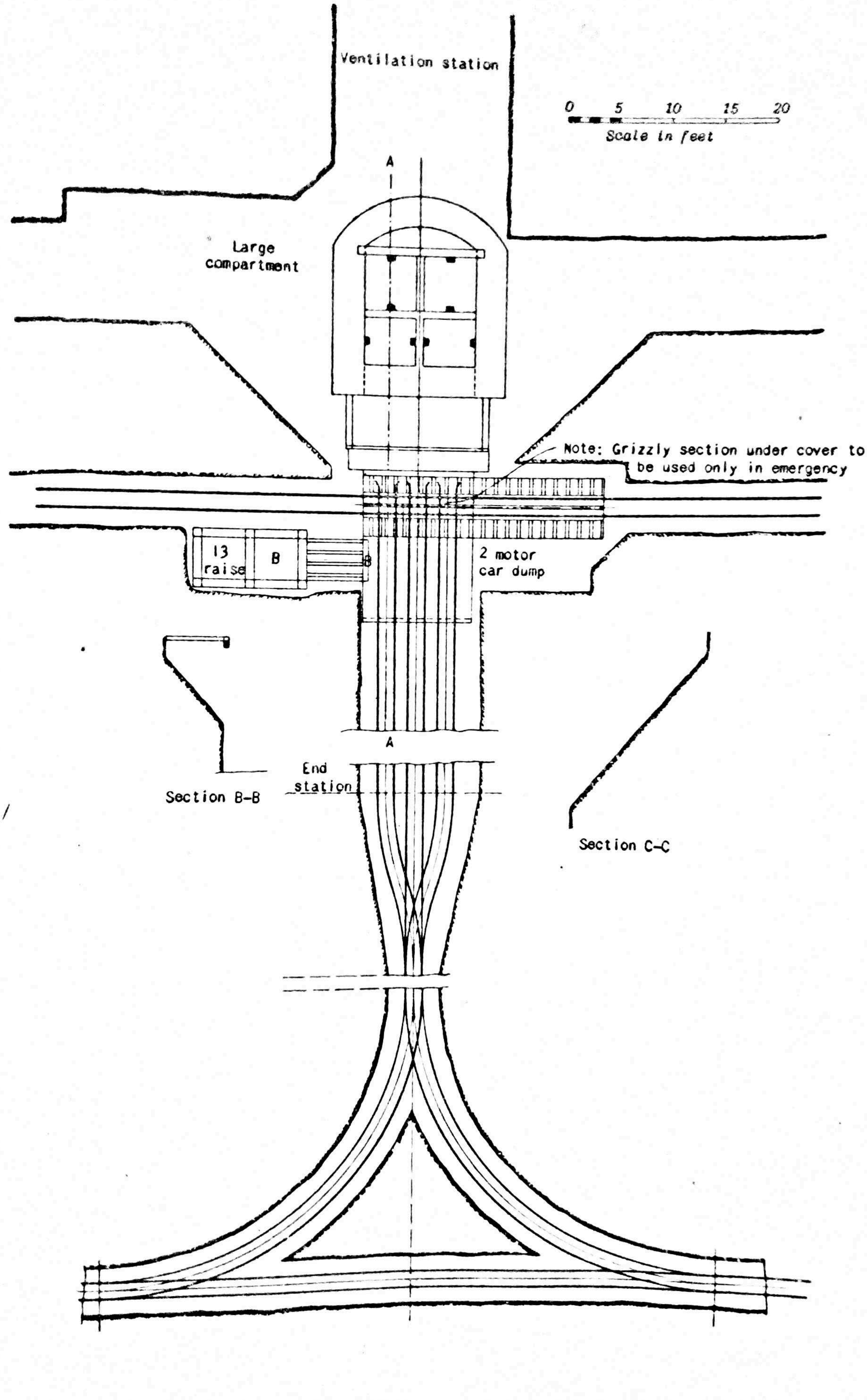


Figure 11.- Loading station, 2200 level

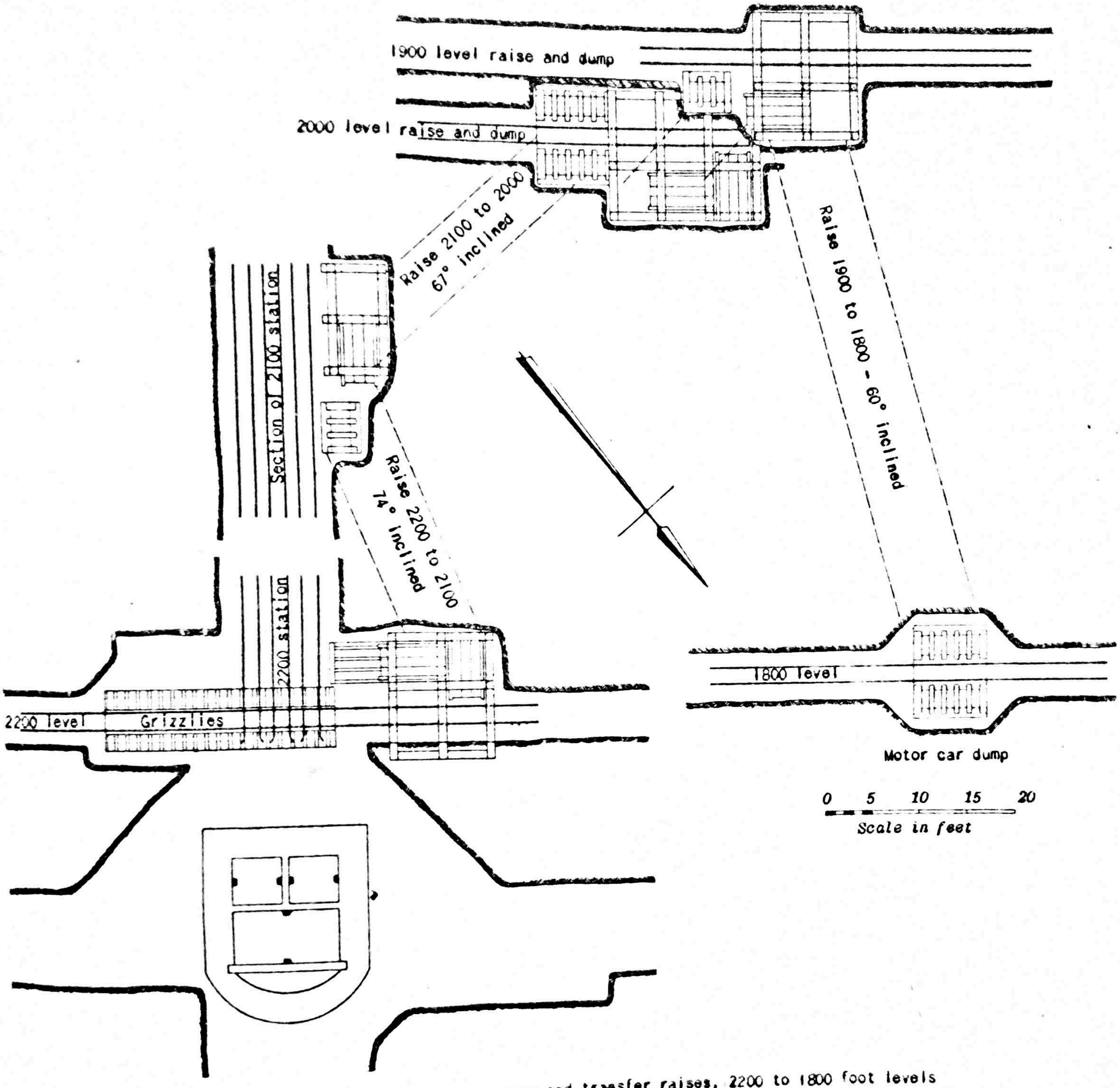


Figure 12.- Location of ore storage and transfer raises, 2200 to 1800 foot levels

In general the plan adopted was to have two adjoining stope sections, each from 45 to 50 feet wide, separated by a pillar about 45 feet in width from the next two stoping sections. Figures 4, A and 7, A, show this general arrangement. Where the width of the ore body was not excessive, the stopes were to be mined from hanging wall to footwall. Where, as above the 1600 level, the width of the ore body is greater, the sections are cut off by either a single cap or regular lead, and new sections are established conforming to the established section lines.

In laying out the stope and pillar sections, it was planned to work both stope sections by cut-and-fill methods. The pillar was to be recovered by means of a Mitchell slice running at right angles to the long stope dimensions. While work at present has not progressed to the point of mining the pillar sections, the success of the cut-and-fill method in this section has made it probable that a serious attempt will be made to use the cut-and-fill in mining the pillars.

In vertical extent, it was planned to take up each section in 200-foot blocks where the continuity of ore would permit; the interval chosen is the distance between haulage levels. Thus one series of stopes will be from the 2200 level to the 2000 level, another from the 2000 level to the 1800 level, and so on in similar blocks for the full vertical extent of the ore. Pillars are carried throughout the ore body in one vertical section.

In this method of stoping, the waste demands for filling are subject to extreme variations. Therefore ample storage capacity is desirable, and provided for by a series of storage raises which directly or by means of a short transfer can in a brief time supply a large amount of waste to a given stope. Figure 9, A and B, shows a system of waste storage raises which are planned to meet the conditions previously outlined. Figure 10 shows the construction details of the waste transfer and storage chutes. Waste is obtained from development headings. An attempt is made to balance the stoping and developing work. No waste is handled in the shaft except between levels and from isolated areas.

At present all ore produced in the Campbell area is trammed to the Junction shaft for hoisting. Provision is made at the Campbell shaft for an ore pocket below the 2200 level, which will be connected to a series of ore raises from the 1800 level. The total capacity of both raises and pocket will be about 1200 tons of sulphide ore.

Figure 11 shows the shaft loading station arrangement on the 2200 level, and Figure 12 shows the general plan of the transfer raises from the 2200 level to the 1800 level.

Stoping Details

Double Lead Sets.--In the illustrations shown (see figs. 3 to 6), crosscuts are driven in the pillar and adjoining stope sections along the section lines. The ground is then cut out for stringer sets.

Posts 9 feet 2 inches long are used, the faces of which are 30 inches from the center line of track. The bottom end is sunk 6 inches below the rail elevation. Between the top of the posts and the stringers are placed 6 by 10 inch crush blocks. The stringers, which are 10 by 12 inches in section, average 13 feet in length and are laid with the greater dimension vertical. This gives a distance of 9 feet 2 inches from rail to stringer. The sets are 5 feet apart center to center. After the stringers are placed, chutes are built in alternate sets and mining operations started above the stringers. Simultaneously with the cutting out for the lead sets, mining operations are started in the stope between the lead sets. In a flat-bottomed stope, the ore is allowed to pile up to form a natural slide toward each lead; the back of the first cut is shaped for the initial fill. Meanwhile work on the fill raise has been pushed, the raise holing to the level above and frequently to the next haulage level, a vertical distance of 200 feet. The ore from this raise is worked to the chutes in either lead. On completion, the raise is stripped and serves for both fill and ventilation purposes. Since the natural angle of repose of the fill is approximately 37°, the back of the stope roughly follows that angle. The character of ground determines the height to which the initial cut is taken.

Simultaneously with these operations, the placing of the lead sets is started along the stope boundary lines. Flooring 3 or 4 inches thick is placed over the stringers, and sills are laid on these. The outside row of posts in each lead set conforms to the stope section line. This leaves the extraction crosscuts both in solid ground with the weight of gob on the solid ground beneath, thus relieving the crosscuts of this weight and enabling them to be maintained with a minimum of repairs.

Standard 10 by 10 inch posts 7 feet 4 inches long are used in the outside of each row of leads. The posts of the inside row of fill side are also 10 by 10 inches but are 7 feet 7 inches long. This difference of 3 inches is to allow for the compression in the timber on the fill side due to the setting of the fill, and thus helps to maintain relatively level floors.

The leads are carried sufficiently in advance of the fill to provide ample height for gob lagging and also to give access to the working floor for the succeeding cut. A manway set is usually carried in each row of leads.

On completion of the lead sets and the section between leads, the stope is cleaned out by hand-mucking and drag scrapers. The bottom of the stope is then covered with a timber mat, which is sometimes laid on sills placed for the purpose. The mat consists largely of scrap timber gathered in the near vicinity.

The lead sets are lagged off on the fill side with either split lagging spaced from 3 to 4 inches apart or with 2 by 12 inch planks 10 feet long, laid edge to edge. Care is taken to have the outer grain of split lagging placed on the side away from the fill. The apparent advantage of the solid lagging is in the tendency of the gob in settling to slide along the smooth surface, thus avoiding a part of the compression on the fill side of the lead sets.

Waste is then dropped through the fill raise until it is within 2 to 3 feet of the back of the stope. As far as possible further handling of the fill is avoided when the transverse angle of repose roughly approaches 37°. A little flattening of the cone on the sides toward the lead sets is necessary.

On the completion of fill, 2 by 10 inch sills are laid flush with the fill; 2 by 10 inch or 2 by 12 inch flooring 10 feet long is then placed on these. Care is taken to avoid hollow places between flooring and fill. Flooring operations are usually started about 10 feet up the slope above the lower edge of the fill and are continued to the top. The spill of waste from these operations rolls to the bottom and is covered in flooring the last section.

If any loose or blocky conditions of the back have developed which can not be corrected by barring-down, small cribbed bulkheads may be erected for support.

Before mining operations are started, 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 and the lead sets. With the filling of subsequent cuts and the raising of lead sets, these grizzlies unless badly worn are moved to the new grizzly floor above. In the event of excessive wear they are frequently left in place, and the openings are closed by means of other similarly worn grizzlies. A checker board is then formed which prevents the cutting out of lead-set timbers and protects the chutes by checking the unrestricted fall of the ore from grizzly floors above. Flooring 4 inches thick is also used to the same end. Figure 3 shows this feature.

With the completion of the flooring, mining operations are resumed. A cut is started at the bottom and broken ore is drawn off as produced. The average height of cut is about 12 feet. Sufficient ore is left undrawn to permit the back to be reached for both barring-down and mining. This also protects the floor while blasting and prevents undue breakage of floor boards. If bad ground develops, the new back may be supported by means of an umbrella stull as shown in Figure 6, B.

After a cut is completed and the ore is drawn into the chutes, the floor is taken up. In making the final cleanup before pulling the floors an especial effort is made to clean the section thoroughly. The last operation is to sweep the floor with heavy brooms, by which a considerable quantity of fines containing a high copper content is obtained.

Each cycle of cut and fill is approximately the same; the lead sets are raised with the main cut.

For stoping drifter-type machines are largely used in drilling 7 to 8 foot holes. The use of this type of machine is advantageous as the back can be readily shaped by drilling holes parallel to the floor angle. Also the drifter is very successful in drilling in the generally hard ground encountered in stopes of this class.

A limited number of stopers, both hand-rotated and self-rotated, are used, principally in driving for lead sets and in raises. Pluggers are used for plugging boulders. As far as possible this work is done on the mining floor, although when necessary plugging is also done on the grizzlies.

In blasting operations No. 8 detonators, safety fuze and 60 per cent strength gelatin dynamite are mostly used. Along some limestone contacts where the softer oxide ores occur 40 per cent gelatin may be used. Main blasting operations are permitted only at the end of the shift. A limited amount of bowlders may be blasted at the noon or supper hour.

Sometimes in order to avoid the necessity of mucking in flat places a cribbed raise is brought up from the hanging-wall end. Either 6 or 10 inch cribbing may be used, depending upon the height to which it is expected to carry the raise.

A chute and manway set may also be carried to the level above from one of the lead sets. This serves as a travel, ventilation, and supply raise.

In the stope shown in Figure 3 the fill raise was started on the hanging wall and one of the crosscuts is inside the stope section. This was one of the earliest stopes started and was made to conform with then-existing raises and crosscuts. Otherwise the procedure followed is exactly the same as outlined in the previous discussion. As will be noted, the floor in this stope is of the rafter type. The purpose of this is twofold: First, to avoid excessive mucking on the first cut due to a flat place in the bottom of the stope; second, additional head room is available beneath the flooring for driving a crosscut to serve as a supply and ventilation outlet for the section below when mining up to this level.

In Figures 7, A and B, and 8 is shown a single-lead, single-cap stope with bottom slightly inclined from lead set to single cap. As will be noted in Figure 7, A, the position of the footwall raise has been moved to a position outside the stope boundary line. It is planned to have the one fill raise serve both sections. The necessity for two extraction drifts is also obviated, and in addition a single row of caps and posts replaces a double row. It is also considered that better filling of the stope may be obtained with less attendant shrinkage of the fill, thus improving mining conditions in the adjacent sections.

With the exceptions noted, the work is carried on in the same cycle as previously described.

Semishrinkage Stoping.--The ground requirements of the semishrinkage method as used in the Bisbee mines of the Calumet & Arizona Mining Co. are similar to those of the cut-and-fill methods, with the possible exception that back conditions must be uniformly good.

Figure 13 is a drawing of a stope using this method in actual operation. The stope is bounded on one side by a fault, on another by a lead set, a single cap forms the third side and the footwall forms the bottom of the stope. The footwall contact represents roughly the normal bedding of the limestone at that point.

In starting operations and to the completion of the flooring of the first fill, the procedure is the same as for the cut-and-fill stoping. On completion of the first cut, mining operations are started as before, but instead of making one 12-foot cut, successive cuts are taken; the excess ore is drawn off and the lead sets are raised with the progress of mining.

Both lead sets and single cap are anchored to the adjacent solid ground by means of cables fastened to anchors in holes drilled into the solid. This prevents any tilting of the sets after the removal of the ore and before the fill is placed.

In blasting, the general practice in cut-and-fill stopes is followed, with the exception that particular care is taken in blasting all bowlders after each blast so that in the final draw, chute and grizzly trouble may be reduced to a minimum.

By using semishrinkage stoping methods a large reduction in flooring expense is made, as some cuts reach a height of 40 feet above the fill floor before being drawn. A considerable breakage of flooring results from the blasting in stopes. Although a large percentage of the cracked and broken floor boards may be used for lagging and the remainder for timber mats, this breakage represents a distinct loss in all cuts. By reducing the number of floors laid, which is one feature of the semishrinkage method, a saving is made not only in the labor incidental to this work, but also in the material itself.

Although grizzly floors are carried up with the advance of the lead sets, they are used only for the ore derived from the advance of the lead. As is shown in Figure 13, the excess is drawn off on the grizzly floor one set above the intersection with the fill. This prevents any movement of broken ore along the floor and thus reduces wear on the boards.

Above the grizzly floor the 3-inch lagging is placed vertically, thus eliminating the tendency to remove additional lagging in order to temporarily facilitate drawing operations.

Before the final ore-drawing operations are started in a stope, the back is closely inspected and very carefully barred down. If minor cracks have developed which may cause any weakness of the back the affected area is supported by a bridge consisting of timbers fastened by anchors in holes drilled into the solid back on each side of the place in question.

PER CENT EXTRACTION

In the Campbell area complete recovery is attempted. A slight loss of fines occurs through the cracks in the flooring, but by careful cleaning up and sweeping an attempt is made to eliminate this loss. A small amount of dilution may occur at the edges of the stopes where the contact is an economic one, as with silicious low grade. Usually the contacts are rather sharply defined so that very little dilution results.

COMPARISON OF DIFFERENT METHODS USED

The semishrinkage method has the advantage over the cut-and-fill method in that there is less flooring expense. Also, due to the longer cycle of operations, a better chance for specialization is afforded, thus increasing the efficiency of the labor. The semishrinkage method has the disadvantage that additional care is necessary in the final trimming of the back and that the sets must be anchored to the solid. A larger proportion of ore is tied up over a longer period in the semishrinkage stopes and waste demands are greater over a short period to avoid excessive intervals of idleness during the filling stage.

UNDERGROUND TRANSPORTATION

Locomotives of 3-1/2-ton, 250-volt, d. c. trolley type are used on the main haulage levels. Storage-battery locomotives of 1-1/2 tons are used on the intermediate levels and for gathering purposes on the haulage levels. These locomotives are transferred between levels on the cages and are serviced at convenient underground charging stations.

Two types of cars are in service. On the haulage levels 38-cubic-foot, saddle-back, side-dump cars equipped with automatic couplers are mainly used for ore haulage. Flatbottomed, end-opening cars of 16 cubic feet are used on both the intermediate levels and the haulage levels. Most of the cars have roller bearings.

All ore is hauled to the Junction shaft, and there is dumped into pockets and hoisted to the surface in 92-cubic-foot skips which average about 5-1/2 tons of sulphide ore per load.

BONUS SYSTEM

At the Bisbee mines the bonus system is employed in a part of the development work and for some stoping.

In the case of development work, standards are set by the mine foreman and approved by the superintendent of mines. Bonus is paid on the measurements made by the engineering department. No penalty clauses for lack of making a standard are included, as daily wages are guaranteed.

In stoping, the bonus basis is set on tonnage per man shift. This is determined from stope tonnage records.

VENTILATION

The Campbell area is ventilated by two fans having a capacity of 70,000-cubic-feet per minute; one located near the shaft on the 1500 level operates against a 3-inch water head; the other located near the shaft on the 2200 level operates against a 3-1/2-inch head.

Part of the air from these fans at the present time is diverted to other areas. The Campbell shaft is downcast and the workings through the ore body are upcast.

Auxiliary equipment electrically driven is used to serve areas off the main air courses.

FIRE HAZARDS AND SAFETY WORK

Shafts serving the producing areas are either completely concreted or will be within a short time.

High-pressure water lines are provided for protection near the heavily timbered stope sections.

With the general adoption of the mining methods being used, the amount of timber in the areas has been greatly reduced. The fill consists mainly of limestone. Care is taken to avoid filling in any one stope extensively with the low-grade sulphides encountered in development that might have a tendency to heat and cause unduly high temperatures.

Chute blasting is done with permissible powder. All fuze is capped and cut in the respective lengths and delivered daily, together with powder, to the underground magazines on the various levels of the mines. The powder and fuze are lowered into the mines between shifts.

Provision is made in the ventilation system for reversal of the operating shafts if occasion should demand.

Underground watchmen are provided during the graveyard and Sunday shifts. They patrol the stope areas after blasting and make inspections for any unusual condition.

Trained helmet crews are maintained from among the underground force. Training periods are held once a week. There are two helmet stations, one on the surface and one underground. Both are provided with oxygen-breathing apparatus and the necessary auxiliary equipment.

First-aid instruction is given on occasion of the visits of the Bureau of Mines instruction car.

A safety inspector is also on duty at all times. He makes recommendations for necessary measures to secure as high a degree of safety as possible.

COSTS

Since the discussion in this paper has been confined to the methods used in an area producing a minor part of the total tonnage, it is deemed advisable to include in the costs only the direct charges which relate to the methods under discussion.

It is further considered that direct charges more nearly represent the information desired for comparison, as the local conditions are reflected in the general charges and as such are influences that do not effect the methods discussed.

Table 1. - Direct charges, Calumet and Arizona Mining Co., Bisbee mines.

68 stope, 1800 level (fig. 3)

Double lead, inclined cut and fill, rafter type bottom
Period mined: May, 1926, to January 1, 1930.
Height mined: 170 feet at the high end.

	Labor	Explosives	Timber	Air	Total	Tons ore
	$32,065.38	$6,999.45	$8,027.57	$11,234.42	$58,326.82	51,970
Cost per ton	$0.62	$0.135	$0.155	$0.215	$1.125	-
Per cent of cost	55	12	14	19	100.0	-

90 stope, 1800 level (figs. 7, A and B, and 8)

Single lead, single cap, inclined cut and fill.
Period mined: January 1, 1928, to January 1, 1930.
Height mined: 80 feet at the high end.

	Labor	Explosives	Timber	Air	Total	Tons ore
	$14,582.12	$2,094.67	$2,771.88	$4,213.90	$23,662.57	20,894.0
Cost per ton	$0.70	$0.10	$0.13	$0.20	$1.13	-
Per cent of cost	62	9	11.5	17.5	100.0	-

Table 1. - Direct charges, Calumet and Arizona Mining Co., Bisbee mines--Continued.

442 stope, 1600 level (fig. 13)

Single lead, single cap, semishrinkage
Period mined: June, 1928, to January 1, 1930.
Height mined: 80 feet at the high end.

(a) First cut-and-fill section.

	Labor	Explosives	Timber	Air	Total	Tons ore
	$3,322.69	$813.87	$1,058.05	$1,434.0	$6,628.61	4094.0
Cost per ton	$0.81	$0.20	$0.26	$0.35	$1.62	-
Per cent of cost	50	12	16	22	100.0	-

(b) Shrinkage section. (Ore drawn in shrinking).

	Labor	Explosives	Timber	Air	Total	Tons ore
	$8,662.32	$2,326.39	$2,413.98	$4,000.95	$17,404.64	8198.0
Cost per ton	$0.106	$0.28	$0.30	$0.49	$2.13	-

(c) Estimated cost of pulling broken ore in stope. (Based on previous shrinks).

	Labor	Explosives	Timber	Air	Total
Cost per ton	$0.255	$0.035	$0.05	$0.015	$0.31

Table 1. - Direct charges, Calumet and Arizona Mining Co., Bisbee mines--Continued

All stopes

(a) Campbell ares - Two year period.
January 1, 1928, to January 1, 1930.

	Labor	Explosives	Timber	Air	Total	Tons ore
	$214,567.57	$33,350.70	$50,025.15	$62,052.60	$359,996.02	280,527.0
Cost per ton	$0.765	$0.12	$0.175	$0.22	$1.28	-
Per cent of cost	60	9.5	13.5	17	100.0	-

(The cost shown here represents the entire production of the Campbell area during the two year period from January 1, 1928, to January 1, 1930. During that time a large number of stopes were started or were in the initial stage. This higher proportional expense, together with ore now tied up in shrinkage operations, is directly reflected in the costs. The stoping in this area is almost exclusively inclined cut and fill and semishrinkage.)

(b) Upper Junction mine costs. Two year period.
January 1, 1928, to January 1, 1930.

(The ground of the Upper Junction mine is not as hard as the Campbell and Lower Junction areas; sections are smaller and a considerable part of the stoping is in finishing out sections in old ore bodies. Square-set, cut-and-fill, and Mitchell-slice methods are all used in this section.)

Table 1. - Direct charges, Calumet and Arizona Mining Co., Bisbee mines -- Contined

(b) Upper Junction mine costs -- Continued

	Labor	Explosives	Timber	Air	Total	Tons ore
	$420,073.34	$38,947.32	$117,491.51	$60,205.60	$636,717.77	427,571.0
Cost per ton	$0.98	$0.095	$0.275	$0.14	$1.49	-
Per cent of cost	65.5	6.5	18.5	9.5	100.0	-

(c) Lower Junction and Campbell mine costs.

Two year period - January 1, 1928 to January 1, 1930.

	Labor	Explosives	Timber	Air	Total	Tons ore
	$474,016.34	$58,186.24	$105,667.91	$104,779.0	$742,649.49	605,540.0
Cost per ton	$0.785	$0.095	$0.175	$0.175	$1.23	-
Per cent of cost	64	7.5	14.25	14.25	100.0	-

(Generally hard ground. Almost exclusively cut and fill and semishrinkage.)

Table 1. - Direct charges, Calumet and Arizona Mining Co., Bisbee mines--Contined

All stopes

(d) Junction and Campbell areas combined.

	Labor	Explosives	Timber	Air	Total	Tons ore
	$894,089.68	$97,133.56	$223,159.42	$164,984.60	$1,379,367.26	1,033,111
Cost per ton	$0.865	$0.095	$0.215	$0.16	$1.335	-
Per cent of cost	65	7	16	12	100.0	-

Note: The labor charge also includes bonus.

The charge for explosives includes powder, caps, and fuze.

The air charge includes:
1. Cost of compression.
2. Air and water lines, hose, and connections.
3. Drilling machines and repairs.

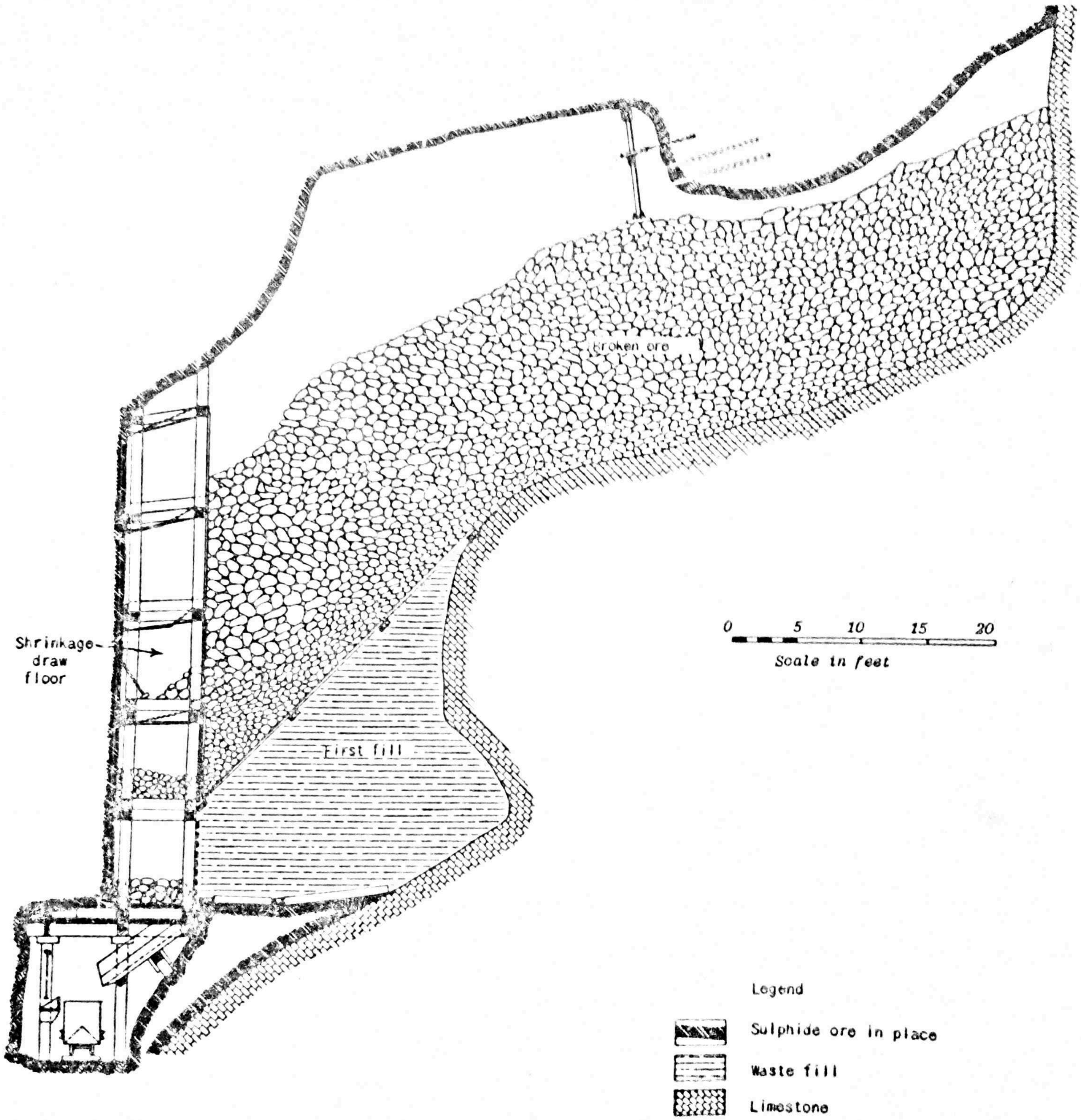


Figure 13.- Semishrinkage stope