U.S. DEPARTMENT OF THE INTERIOR
HAROLD L. ICKES, Secretary

BUREAU OF MINES
JOHN W. FINCH, Director

Bulletin 381

LEAD AND ZINC MINING AND MILLING IN THE UNITED STATES CURRENT PRACTICES AND COSTS

By

CHAS. F. JACKSON, JOHN B. KNAEBEL, AND C. A. WRIGHT

UNITED STATES
GOVERNMENT PRINTING OFFICE
WASHINGTON : 1935

For sale by the Superintendent of Documents, Washington, D.C. Price, 15 cents (paper cover)

CONTENTS

	Page
Introduction	1
Acknowledgments	1
Object and scope of bulletin	1
Production statistics	2
Part 1. Geology of lead and zinc deposits	6
Types of deposits	
Mineralogy of lead and zinc	6
deposits	6
Character of outcrops	11
Changes with increasing depth	12
Localization of ore shoots	13
Geologic age of lead and zinc deposits	16
Mining districts	16
Southeastern Missouri district	17
Tri-State district	18
Eastern Tennessee district	23
Coeur d'Alene district, Idaho	24
Park City district, Utah	27
Tintic district, Utah	29
Butte district, Montana	32
Barker district, Montana	34
Pecos district, New Mexico	34
Santa Rita-Hanover region, New Mexico	36
Yellow Pine district, Nevada	39
Edwards district, New York	41
Franklin Furnace district, New Jersey	44
Part 2. Exploration, development, and mining	47
Exploration	47
Examples of exploration methods	48
Southeastern Missouri district	48
Tri-State district	50
Eastern Tennessee district	54
Edwards district, New York	54
Coeur d'Alene district, Idaho	57
Park City district, Utah	58
Tintic district, Utah	59
Butte district, Montana	61
Barker district, Montana	61
Part. 2. Exploration, development and mining—Contd.	
Exploration—Continued	
Examples of exploration methods—Continued	
Pecos district, New Mexico	61
Santa Rita-Hanover region, New Mexico	62
Battle Mountain district, Colorado	62
Summary of exploration practice	63
Sampling and estimation of ore deposits	65
Sampling methods	66
Examples of sampling and estimating practice	68
Southeastern Missouri district	68
Tri-State district	71
Edwards district, New York	74
Coeur d'Alene district, Idaho	75
Park City district, Utah	75
Tintic district, Utah	76
Barker district, Montana	77
Butte district, Montana	77
Pecos district, New Mexico	77
Santa Rita-Hanover region, New Mexico	78
Battle Mountain district, Colorado	78
Summary of sampling and estimating practice	79
Mine development	80
Shaft versus tunnel development	81
Shafts	82
Level development	86
Stope development	94
Cost of mine development	96
Stoping	98
Stoping methods	99
Classification of stoping methods	100
Open stopes	101
Heading-and-bench system	101
Summary of open-stope methods	107

	Page
Part 2. Exploration, development, and mining—Contd.	
Stoping—Continued	
Square-set stopes	107
Coeur d'Alene district, Idaho	110
Park City district, Utah	112
Tintic district, Utah	115
Butte district, Montana	117
Santa Rita-Hanover region, New Mexico	121
Battle Mountain district, Colorado	121
Summary of square-set stoping methods	122
Cut-and-fill stoping	123
Barker district, Montana	125
Pecos district, New Mexico	126
Battle Mountain district, Colorado	128
Coeur d'Alene district, Idaho	129
Summary of cut-and-fill stoping	130
Shrinkage stoping	131
Franklin district, New Jersey	134
Summary of shrinkage stoping	134
Top-slicing	135
Applicability of top-slicing	137
Cost of stoping	137
Cost of mining	141
Part 3. Methods of concentrating lead and zinc ores	146
Introduction	146
Acknowledgments	146
Brief summary and flow sheets of milling practice at various concentrators	147
Concentrator A — Eagle-Picher Lead Co., Netta mill, Picher, Okla	147
Concentrator B—Canam Metals Corporation, White Bird concentrator, Picher, Okla	147
Concentrator C—Bunker Hill & Sullivan Mining & Concentrating Co., West mill, Kellogg, Idaho	150
Concentrator D—Hecla Mining Co., crushing, sorting, and treatment plants, Gem, Idaho	150
Concentrator E—American Zinc Co., Mascot, Tenn	154
Concentrator F—Flat River, Mo	154
Part. 3. Methods of concentrating lead and zinc ores—Con.	
Brief summary, etc—Con.	
Concentrator G—Federal Mining & Smelting Co., Page concentrator, Kellogg, Idaho	156
Concentrator H—Federal Mining & Smelting Co., Morning concentrator, Mullan, Idaho	158
Concentrator I—St. Joseph Lead Co., Hughesville, Mont	159
Concentrator J—Treadwell Yukon Co., Tybo, Nev	160
Concentrator K—U.S. Smelting, Refining & Mining Co., Midvale concentrator, Midvale, Utah	161
Concentrator L—Chief Consolidated Mining Co., Eureka, Utah	162
Concentrator M—Eagle-Picher Lead Co., Montana concentrator, Ruby, Ariz	162
Concentrator N—Black Hawk Consolidated Mines Co., Hanover, N.Mex	165
Concentrator O—American Metals Co., Tererro, N. Mex	167
Concentrator P—St. Joseph Lead Co., Balmat, N.Y	168
Discussion of milling operations	170
General	170
Hand-sorting	170
Crushing	171
Grinding	172
Sizing and classifying	177
Gravity concentration	178
Jigging	178
Table concentration	179
Flotation	181
Flotation reagents	182
Dewatering of concentrates	184
Grade of products and recoveries	185
Disposal of tailings	187
Water supply	187
Power	188
Labor	189
Milling costs	191
Conclusion	195
Costs of mining and milling	195
Cost per pound of lead and zinc	195
Index	199

ILLUSTRATIONS

Fig.		Page
1.	Map of the United States, showing principal sources of lead and zinc ores, 1930	5
2.	Cross-section of geological formations from Flat River south to Simms Mountain, southeastern Missouri	17
3.	Typical mineralization in 10-foot face of ore, southeastern Missouri	18
4.	*A*, Plan of mine workings, Southeastern Missouri lead district, showing irregular outline of ore bodies; *B*, cross-sections through workings shown in *A*, showing ore horizons	19, 20
5.	Plan of Barr mine, Tri-State district, showing irregular outline of ore bodies	21
6.	*A*, Typical mineralization in face of brecciated ore body, Tri-State district; *B*, ore body of semisheet, ground type, Tri-State district	22
7.	*A*, Mascot (Tenn.) ore body as indicated by prospect drilling; *B*, vertical longitudinal section, Bunker Hill & Sullivan mines, Coeur d'Alene district, Idaho, Jan. 1, 1932	24
8.	Cross-section through Park Utah mine, Park City district, Utah, looking N. 80° W	27
9.	Bedded lead ore cut by "gash" fault, Silver King mine, Park City district, Utah	28
10.	*A*, Composite of Tintic Standard mine, Tintic district, Utah; *B*, ideal section on line N. 45° E. through no. 2 shaft, Tintic Standard mine	31
11.	Development of two main ore bodies, Pecos mine, Pecos district, New Mexico	35
12.	Theoretical geological section, Ground Hog mine, Santa Rita-Hanover region, New Mexico	38
13.	Plan and section of Yellow Pine mine, Yellow Pine district, Nevada	40
14.	*A*, Typical breast of ore, D–11 stope above 1,300-foot level, Edwards mine, Edwards district, New York; *B*, plan and sections of no. 3 ore body, Edwards mine, 700 to 1,300-foot levels	43
15.	Plan and sections of ore bodies, Franklin district, New Jersey	45
16.	*A*, No. 2 ore body as outlined by surface drilling at time of sinking no. 2 shaft, Mascot, Tenn.; *B*, generalized vertical longitudinal projection, part of Morning mine, Mullan, Idaho	55
17.	Method of forming polygonal areas around surface diamond-drill holes, No. 8 mine, Southeastern Missouri district	69
18.	Ore body in Tri-State district, showing drill holes, outline of estimated ore, and outline of ore as mined	72
19.	Typical shaft plan for long cage and skips	85
20.	*A*, Plan of level, Morning mine, Idaho; *B*, partly developed level in wide ore body, Bunker Hill & Sullivan mine; *C*, complete development of sill floor, showing ore chutes for higher floors, Bunker Hill & Sullivan mine	93
21.	Cross-section showing stope development by footwall raise, Bunker Hill & Sullivan mine	95
22.	*A*, Method of advancing breast and bench, No. 8 mine, Southeastern Missouri district; *B*, section showing mining with electric shovel in thick ore, No. 8 mine; *C*, method of driving heading and bench stope and turning pillars, Southeastern Missouri district	102
23.	*A*, Heading and bench stoping, Tri-State district; *B*, heading and bench stoping, showing use of "sprung" holes	103
24.	Method of carrying heading and bench stope, Mascot, Tenn	104
25.	General features of the several variations of open-stope mining at Mascot, Tenn	104
26.	*A*, Open stopes, Edwards mine (generalized); *B*, breaking and handling ore in open stopes, Edwards mine	105
27.	Typical square-set timbering	108
28.	Plan of typical floor, square-set stope, Bunker Hill & Sullivan mine	109
29.	Sections showing stages in square-set stoping, Bunker Hill & Sullivan mine	111
30.	Stringer sets used in stopes at Silver King Coalition mines, Utah	113
31.	*A*, Square-set method, Park Utah mine; *B*, scraping in flat square-set stopes, Park Utah mine	114

Fig.		Page
32.	*A*, First step in starting a stope, Tintic Standard mine; *B*, vertical section of stope, showing method of timbering; *C*, horizontal section through stope, showing use of diamond bracing	116
33.	Overhand square-set stoping, Black Rock mine, Butte, Mont	118
34.	Underhand square-set stoping, Black Rock mine, Butte, Mont	119
35.	Square-set-and-fill stoping method, Ground Hog mine, Vanadium, N.Mex	120
36.	Longitudinal and cross-sections of typical cut-and-fill stope, Block P mine, Hughesville, Mont	124
37.	*A*, Cut-and-fill stoping, employing alternate stopes and pillars, Gilman, Colo.; *B*, stoping with stringer sets and fill, Morning mine	125
38.	Section across vein through cut-and-fill stope, Pecos mine, New Mexico	127
39.	Generalized sketch showing method of stope mining at the Hecla mine	130
40.	*A*, Shrinkage stoping on drift pillars in narrow vein; *B*, sections of longitudinal stope, Franklin, N.J	132
41.	Successive stages in mining a transverse shrinkage stope, Franklin, N.J	133
42.	Top-slicing pillars, Franklin, N.J	136
43.	Flow sheet of Netta mill, Picher, Okla., concentrator A	148
44.	Flow sheet of White Bird concentrator, Picher, Okla., concentrator B	149
45.	Flow sheets of ore treatment at Bunker Hill & Sullivan mills, concentrator C	151
46.	Flow sheet of crushing and sorting plant, Hecla Mining Co., Gem, Idaho, concentrator D	152
47.	Flow sheet of ore treatment, Hecla Mining Co., Gem, Idaho, concentrator D	153
48.	Flow sheets of ore treatment, American Zinc Co. of Tennessee, concentrator E	154
49.	Flow sheets of various plant operations at a Flat River (Mo.) mill, concentrator F	155
50.	General flow sheet of Page concentrator, Federal Mining & Smelting Co., Kellogg, Idaho, concentrator G	157
51.	Flow sheets of treatment, Morning mill, Mullan, Idaho, concentrator H	158
52.	Flow sheet of crushing plant and mill, Hughesville concentrator, St. Joseph Lead Co., Hughesville, Mont., concentrator I	159
53.	Flow sheet of Tybo concentrator, Treadwell-Yukon Co., Ltd., Tybo, Nev., concentrator J	160
54.	Flow sheet of Midvale flotation plant, U.S. Smelting, Refining & Mining Co., Midvale, Utah, concentrator K	161
55.	Flow sheets of ore treatment, Chief Consolidated Mining Co., Eureka, Utah, concentrator L	163
56.	Flow sheets of ore treatment, Montana concentrator, Eagle-Picher Lead Co., Ruby, Ariz., concentrator M	164
57.	Flow sheet of Black Hawk Consolidated mill, Grant County, N.Mex., concentrator N	166
58.	Flow sheet of Pecos concentrator, American Metal Co., Tererro, N. Mex., concentrator O	167
59.	Flow sheet of Balmat 500-ton mill, St. Joseph Lead Co., St. Lawrence County, N.Y., concentrator P	169

LEAD AND ZINC MINING AND MILLING IN THE UNITED STATES; CURRENT PRACTICES AND COSTS[1]

By CHAS. F. JACKSON,[2] JOHN B. KNAEBEL,[3] and C. A. WRIGHT[4]

INTRODUCTION

This bulletin is one of a series issued by the Bureau of Mines on mining and milling of ores of the principal metals. They summarize the results of 4 years of field study in the most important mining districts of the United States and of a series of information circulars published by the Bureau dealing with operating methods and costs at a large number of individual mines and milling plants.

Because of the close association in nature of lead and zinc minerals and because a considerable proportion of the zinc produced in this country is obtained as a byproduct of lead mining and vice versa, it seems desirable to consider the mining and milling of lead and zinc ores in the same volume. Indeed, it would be difficult to consider them separately, either from the standpoint of production or technical control of operations.

Silver and, to a smaller extent, gold are commonly associated with lead and zinc ores, but the association is usually so intimate that these metals may be recovered with either the lead or zinc without introducing serious complications into mining or milling. This discussion will not go beyond milling in the processing of the ores.

ACKNOWLEDGMENTS

This bulletin is partly a compilation of information previously published in Bureau of Mines information circulars, in the transactions of technical societies, and in the technical press. Original authors are credited by footnote acknowledgments.

The section on geology has been reviewed by F. G. Wells of the United States Geological Survey, to whom special acknowledgment is given.

OBJECT AND SCOPE OF BULLETIN

The purpose of this paper is to combine in a single volume discussions of the winning of lead and zinc ores from the mines and of the milling of the ores with descriptions of practices at typical

---

[1] Work on manuscript completed August 1932.
[2] Principal mining engineer, U.S. Bureau of Mines.
[3] Assistant mining engineer, U.S. Bureau of Mines.
[4] One of the consulting engineers, U.S. Bureau of Mines (author of the section on milling).

properties in the United States. The rate of production of these ores and the location and importance of the several producing districts are discussed only briefly, since these subjects have been treated in considerable detail from a world-wide economic viewpoint in earlier publications.[5]

The geologic occurrence of the ores and types of ore bodies will be considered briefly; then the methods of exploring and developing the ore deposits, the mining and milling of the ores, and mine and mill operating costs at representative properties will be discussed.

PRODUCTION STATISTICS

Statistics on lead production of the world and of the United States have been compiled and published for 1801 to 1927, inclusive;[6] similar statistics covering the production of zinc have also been published.[7] These statistics show trends in production rates and changes in the distribution of production here and abroad. In 1930 the Bureau of Mines published a map of the United States[8] which showed graphically the production of the principal metals in 1928 and the production of the different States and mining districts with their relative importance, and included a table giving the tons of metal produced in each district and the value thereof. Except for iron, which is reported as iron ore, the figures are based upon short tons of metal produced and do not indicate the tonnage of ore mined to supply this production. The table shows that in 1928, 627,153 short tons of lead and 695,170 short tons of zinc were produced from mines having individual outputs valued at $100,000 or more during that year. These figures compare with a total production of 684,165 short tons of domestic refined lead in 1927, 695,830 tons in 1926, and 686,451 tons in 1925, and with a total production of primary zinc of 592,516 tons, 618,422 tons, and 572,946 tons, respectively, in 1927, 1926, and 1925. These were all high production years compared to 1921 and 1922, when production of lead was 395,287 and 481,689 tons, respectively, and production of zinc, 200,500 and 354,277 tons.

Prior to 1929 the Bureau did not obtain figures on the tons of crude ore mined at individual properties, but in 1930 incomplete returns were obtained for 1929. Figures for 1930 are somewhat more complete.

This report deals principally with the methods and costs of mining and milling the ore. These operations precede smelting and refining; hence, available statistics on the tonnages of lead and zinc ores mined have interest even though incomplete.

In 1929, 161 lead and zinc mines in the United States, with individual outputs valued at $100,000 or more, were reported to have produced 21,299,748 tons of ore yielding an average of 91.3 pounds of

---

[5] Pehrson, Elmer W., Summarized Data of Zinc Production: Econ. Paper 2, Bureau of Mines, 1929, 47 pp.
Smith, Lewis A., Summarized Data of Lead Production: Econ. Paper 5, Bureau of Mines, 1929, 44 pp.

[6] Smith, Lewis A., work cited.

[7] Pehrson, Elmer W., work cited.

[8] Pehrson, Elmer W., Map Showing Value of 1928 Production of Gold, Silver, Copper, Lead, Zinc, and Iron Ore of the United States by States and Districts Yielding $100,000 or More.

lead and zinc combined per ton of ore,[9] which is equivalent to 1,944,666,992 pounds (972,333 short tons) of lead and zinc. This figure is not comparable with the figures cited earlier in this report, which covered the total production of lead and zinc in the United States and were compiled upon a different basis.

In 1930, 19,360,710 tons of lead and zinc ore were mined from 160 properties producing more than 1,000 tons each. From data available it has not always been possible to classify a mine definitely as a lead mine, a lead-zinc mine, or a zinc mine, since the figures for some mines did not specifically indicate the metal from which most of the income was derived. The tonnages given are for freshly mined ore; re-treated tailings have been excluded as far as possible, although in a few instances small amounts of such material may have been included with freshly mined ore.

A few mines included may have derived the greater part of their income from silver and gold, but the tonnages involved were relatively small. The grouping of tonnages into three classes is believed, however, to be substantially correct; ores classified as lead ores are believed to contain lead or lead and silver as the predominant metals, those classified as zinc ores to be predominantly zinc, and those classified as lead-zinc ores to contain either lead or zinc in excess of the other metal but to yield important amounts of each. Thus, the ores of the Tri-State district which contain much more zinc than lead (averaging about 6 tons of zinc sulphide to 1 ton of lead sulphide) are classed as lead-zinc ores. The grouping in this paper is determined by the classification returned by the operating company, although in a few instances the relative amounts of lead and zinc were not clearly indicated.

Of a total of 19,360,710 tons, 7,901,553 tons are classed as lead ores (6,700,792 tons being mined in Missouri), 2,554,142 tons as zinc ores (New Jersey furnishing 690,348 tons and Tennessee 663,236 tons), and 8,905,015 tons as lead-zinc ore (5,691,541 tons being accounted for by the Tri-State zinc and lead district).

Table 1 shows the rank of the different States on the basis of tonnage of crude ore mined and not on the basis of production of the metals. The figures indicate rank, starting with 1 for the greatest production. The actual tonnages are not given, since in a few instances virtually all the tonnage from the State is produced by one company, which might not desire its production known; however, table 2 shows the tonnages of ore mined during 1930 by regions.

Although the figures may not be complete, the largest producers are included, and any tonnages not given are so small as to affect the totals only slightly.

---

[9] Wright, C. W., Mining Methods and Costs at Metal Mines of the United States: Inf. Circ. 6503, Bureau of Mines, 1931, 39 pp.

TABLE 1.—*Rank of the States as producers of crude ore in 1930*

State	Lead ore	Zinc ore	Lead-zinc ore	Total	State	Lead ore	Zinc ore	Lead-zinc ore	Total
Arizona	5			16	New Mexico		7	7	11
California	8			18	New York		5		12
Colorado	7	4	6	8	Oklahoma		9	1	2
Idaho	2		3	4	Tennessee		2		7
Kansas		6	2	3	Texas	8			15
Missouri	1	8	11	1	Utah	3		4	5
Montana	6		8	13	Virginia			5	9
Nevada	4		9	14	Washington			12	17
New Jersey		1		6	Wisconsin		3	10	10

NOTE.—1931 and 1932 were abnormal years when many mines were shut down and the production of others was greatly curtailed. Figures for 1930 are therefore the latest ones which are indicative of the relative productive capacities of the several States.

TABLE 2.—*Crude ore mined by regions in 1930*

Region	Lead ore	Zinc ore	Lead-zinc ore	Total
Rocky Mountain States, California, and Washington	1, 166, 451	339, 754	2, 567, 336	4, 073, 541
Central States	6, 735, 102	1, 307, 125	5, 101, 317	13, 143, 544
Eastern States		907, 263	1, 236, 362	2, 143, 625
Total	7, 901, 553	2, 554, 142	8, 905, 015	19, 360, 710

Figure 1 shows the principal source of lead and zinc ores in 1930, by regions.

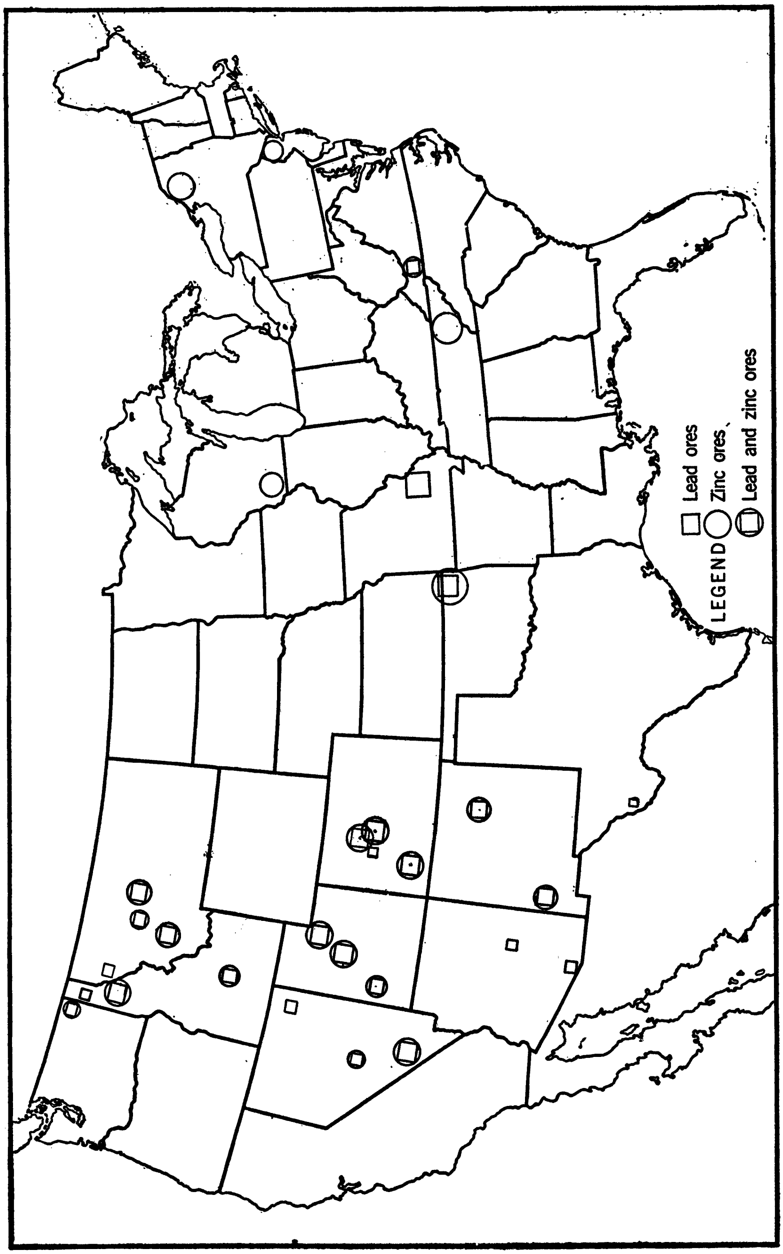

FIGURE 1.—Map of the United States, showing principal sources of lead and zinc ores, 1930.

Part 1.—GEOLOGY OF LEAD AND ZINC DEPOSITS

Methods and costs of mining and milling are influenced by the geology and mineralogy of the ore deposits and by the structure and other physical characteristics of the ores. Important considerations are: Type of the deposit and distribution of the ore shoots; size, shape, structure, and dip of the ore bodies; depth of cover; strength of the ore and enclosing rocks; nature and distribution of the valuable minerals in the ore bodies; nature of the associated gangue minerals; marginal characteristics of the deposits (whether ore limits are sharply defined or gradational); and other physical and chemical factors.

Although a general discussion of these subjects is pertinent to the present paper space will not permit treating the geology of lead and zinc deposits exhaustively. The discussion will therefore be confined to factors having an important bearing upon the methods and costs of exploring, developing, and mining the ore deposits and the milling of the ores.

TYPES OF DEPOSITS

The principal sources of lead and zinc ores in the United States are deposits of the following types:

1. Deposits in flat-lying or low-dipping beds of limestone, dolomite, or chert, typified by the deposits of the Tri-State district, southeastern Missouri, and eastern Tennessee. These deposits are the present source of the greatest production in this country in terms of crude ore mined.
2. Dipping tabular deposits in the form of fissure veins or occupying wide fault or shear zones, typified by deposits of the Coeur d'Alene district in Idaho, some Utah deposits, and those of other Rocky Mountain States.
3. Replacement deposits, usually in carbonate rocks and genetically related to igneous activity. The ores of the Leadville (Colo.), Tintic and Park City (Utah), and Edwards (N.Y.) districts are well-known examples of this type.
4. Deposits due to igneous metamorphism, usually remote from contacts. The geologic relations of these deposits are not yet thoroughly understood. In this country the best-known examples are the deposits of Magdalena, N.Mex., and the Franklin and Sterling Hill ore bodies in New Jersey.

MINERALOGY OF LEAD AND ZINC DEPOSITS

The commonest and commercially most important ore minerals of lead and zinc are the simple sulphides, galena and sphalerite, and the great bulk of production is derived from these minerals. The secondary carbonates and, to a lesser degree, sulphates of the two metals are an important source, however, and in some places such as the Tintic (Utah) and Yellow Pine (Nev.) districts they contribute most of the ore mined.

In a few districts the ore bodies are characterized by a very simple mineralization, with one metallic mineral present to virtual exclusion of all others. Thus, in southeastern Missouri the lead ores are generally composed of galena in an essentially nonmetallic gangue, with almost no silver, zinc, or other valuable metals; similarly, the deposits at Mascot, Tenn., and Edwards, N.Y., are mined for zinc and are noteworthy on account of the small percentages of lead associated with them.

As a rule, however, the primary ores of lead and zinc are rather closely associated mineralogically in various proportions. In the numerous deposits of the Western States silver is present almost universally; and still greater complexity of mineralization is found in many ores, some of which contain not only lead and zinc sulphides but also minerals of copper, iron, gold, and silver and occasionally vanadium, molybdenum, nickel, cobalt, arsenic, antimony, and other metals. Various combinations of minerals occur in these complex deposits, wherein lead and zinc may have paramount, secondary, or only minor commercial importance.

Some western deposits are worked profitably only because they contain appreciable quantities of silver in addition to lead or zinc. In the primary ore this silver is largely present as argentite (Ag_2S), intimately associated with the base-metal sulphides, but it occasionally occurs in the form of such minerals as argentiferous tetrahedrite and tennantite, proustite, pyrargyrite, and polybasite. Secondary processes resulting in the migration of silver downward, its reprecipitation at shallow depths in the form of native silver, cerargyrite, embolite, and similar minerals, and its deposition in the zone of sulphide enrichment immediately below water level in the form of argentite, stephanite, proustite, pearceite, and polybasite have enriched to a greater or smaller extent many deposits carrying only moderate amounts of silver in the primary ore. Many western mines now worked chiefly for lead and zinc were silver mines earlier in their history.

In the Franklin district, New Jersey, much zinc is produced from ores composed essentially of the oxides franklinite and zincite and the silicate willemite. Elsewhere these minerals have little or no commercial importance, and almost the entire lead and zinc production of the remainder of the United States is provided by galena and sphalerite or the oxidized minerals derived therefrom. The gangue of lead and zinc ores depends largely on the nature of the country rock, as well as on the mode of origin of the deposits.

In the Mississippi Valley region the ores are found in calcareous or dolomitic rocks and were deposited under conditions of shallow depth. The gangue minerals are chiefly calcite, dolomite, chert, or jasper, with some barite, marcasite, pyrite, and other minerals. Minerals formed characteristically under deep-vein zone (high-temperature) conditions are rare or lacking.

In deposits closely related to igneous rocks the gangue always contains some quartz. If limestone or dolomite is the country rock carbonate minerals are usually abundant and frequently predominate, but in veins in granitic or schistose rocks quartz is often the principal nonmetallic mineral. Deposits formed at considerable depth, like those of the Coeur d'Alene region, may contain such minerals of the high-temperature type as magnetite, tourmaline, and pyroxenes.

Chlorite, sericite, talc, and serpentine abound where the country rocks have been much altered by regional metamorphism or local shearing and brecciation, followed by chemical action (chiefly leaching and hydration).

The mineralogy of lead and zinc deposits has been discussed from a geological standpoint by many authors; bibliographies of their works appear in three recent volumes.[1] No attempt is made in this report to discuss individual minerals from a descriptive standpoint, since complete and exhaustive references on this phase of the subject are available in most libraries.[2]

To present a fairly complete picture of the mineralogy of lead and zinc deposits as briefly as possible with clarity, all the common species are listed, together with many of the rarer varieties, in table 3. Chemical composition and brief remarks as to occurrence, mode of formation, and economic importance are included. The names of minerals of particular economic importance or of widespread and abundant occurrence are printed in capital letters. The list is given under the four headings: Lead minerals, zinc minerals, associated metallic minerals, and nonmetallic gangue minerals; under each heading the various species are arranged in the order of Dana's classification.[3]

TABLE 3.—*Minerals occurring in lead and zinc deposits*

Mineral	Composition	Remarks
Lead minerals:		
Lead	Pb	Very rare mineral. Reported from mines in Idaho and Colorado.
GALENA	PbS	The commonest and most important lead mineral. Occurs almost universally, being found in all types of lead deposits. Often carries silver, sometimes gold.
Zinkenite	$PbSb_2S_4$	Rare minor mineral.
Jamesonite	$Pb_2Sb_2S_5$	Uncommon; may be argentiferous.
Bournonite	$(Pb,Cu_2)_3Sb_2S_6$	Uncommon.
Cotunnite	$PbCl_2$	Rare; occurs in oxidized portions of lodes.
Massicot	PbO	Rare; occurs in oxidized zone.
Minium	$2PbO.PbO_2$	Do.
Plattnerite	PbO_2	Do.
CERUSSITE	$PbCO_3$	Commonest oxidized lead mineral; important ore.
Phosgenite	$(PbCl)_2CO_3$	Rare mineral of the oxidized zone.
PYROMORPHITE.	$Pb_4(PbCl)(PO_4)_3$	Secondary mineral of considerable importance in a few districts.
Mimetite	$Pb_4(PbCl)(AsO_4)_3$	Secondary mineral of very slight economic importance.
Vanadinite	$Pb_4(PbCl)(VO_4)_3$	Secondary mineral mined chiefly for vanadium; unimportant for lead.
ANGLESITE	$PbSO_4$	Common secondary mineral; furnishes appreciable amounts of lead.
Leadhillite	$PbSO_4.2PbCO_3.Pb(OH)_2$	Unimportant oxidized mineral. Has been mined in Missouri, Nevada, and at Tintic, Utah.
Beaverite	$CuO.PbO.Fe_2O_3.2SO_3.4H_2O$	Rare oxidized mineral; was mined in Frisco district, Beaver County, Utah.
Plumbojarosite	$PbFe_6(OH)_{12}(SO_4)_4$	Rare oxidized mineral; no economic importance.
Crocoite	$PbCrO_4$	Uncommon; unimportant as a source of lead.
Wulfenite	$PbMoO_4$	Ore of molybdenum; unimportant for lead.

[1] Lindgren, W., Mineral Deposits: McGraw-Hill Book Co., New York, 3d ed., 1928, 1049 pp.

[2] Ries, H., Economic Geology: John Wiley & Sons, New York, 5th ed., 1925, 843 pp.
Emmons, W. H., Principles of Economic Geology: McGraw-Hill Book Co., New York, 1st ed., 1918, 606 pp.
Dana, E. S., Textbook of Mineralogy: John Wiley & Sons, New York, 3d ed., revised by W. E. Ford, 1922, 719 pp.
Brush, G. J., Determinative Mineralogy and Blowpipe Analysis: John Wiley & Sons, New York, 16th ed., revised by S. L. Penfield, 1898, 312 pp.

[3] Ford, Wm. E., Dana's Manual of Mineralogy: John Wiley & Sons, New York, 13th ed., 1915, p. 115.

TABLE 3.—*Minerals occurring in lead and zinc deposits*—Continued

Mineral	Composition	Remarks
Zinc minerals:		
SPHALERITE OR BLENDE.	ZnS	Principal source of zinc production. Except in a very few deposits it is the only important primary mineral; it is rarely deposited by secondary processes. May carry valuable amounts of silver and sometimes gold.
MARMATITE	ZnS+10 percent Fe	Iron-bearing blende, often called "blackjack." An important variety of zinc sulphide.
Wurtzite	ZnS	Hexagonal form of ZnS. Usually derived from sphalerite and redeposited as a secondary mineral.
Zincite	ZnO	High-temperature mineral. Rare or unimportant except in Franklin district, N.J. May contain manganese.
FRANKLINITE	$(Fe, Zn, Mn)O \cdot (Fe,Mn)_2O_3$	Uncommon except in Franklin district, N.J., where it is an important ore; high-temperature mineral.
SMITHSONITE	$ZnCO_3$	Common secondary mineral, important commercially. Usually found in deposits in carbonate rocks. Also called "dry bone" ore.
HYDROZINCITE	$ZnCO_3.2Zn(OH)_2$	Secondary mineral, important in some districts.
WILLEMITE	Zn_2SiO_4	Important in Franklin district, N.J., rare elsewhere. May contain manganese and some iron.
CALAMINE	$(ZnOH)_2SiO_3$	Important secondary mineral, usually associated with smithsonite, hydrozincite, cerussite, etc., in carbonate rocks.
Goslarite	$ZnSO_4.7H_2O$	Secondary mineral, unimportant commercially.
Associated metallic minerals:		
GOLD	Au, alloyed with some Ag	Present in nearly all western lead-zinc ores; usually intimately associated with pyrite, galena, and/or blende. Often forms a valuable byproduct.
SILVER	Ag, alloyed with some Au	Present in the oxidized ores of many western lead and zinc deposits. Usually a secondary mineral.
Copper	Cu, often alloyed with silver	Sometimes occurs in the oxidized portions of veins containing chiefly lead-zinc ore.
Realgar	AsS	Secondary (supergene) mineral; not uncommon in oxidized zones.
Orpiment	As_2S_3	Secondary (supergene) mineral; sometimes found in oxidized ores.
Stibnite	Sb_2S_3	Primary mineral found in some western lead-zinc ores.
Molybdenite	MoS_2	Primary mineral found in some lead-zinc deposits, usually in minor quantities.
ARGENTITE	Ag_2S	Most important primary silver mineral, usually intimately mixed with galena or sphalerite; also, an important secondary mineral in enriched zones. Often provides the chief source of income from lead-silver ores.
Chalcocite	Cu_2S	Of secondary importance in lead-zinc ores. Usually occurs in zone of secondary enrichment.
Greenockite	CdS	Present in very intimate mixture with sphalerite in nearly all zinc deposits in amount up to 0.3 to 1 percent or more. Also as a yellow coating on blende, in Tri-State district and elsewhere.
Millerite	NiS	Sometimes occurs in small amounts with lead-zinc ores.
Niccolite	NiAs	Sometimes occurs in small amounts with zinc-lead ores.
Pyrrhotite	Fe_nS_{n+1}	Found in the gangue of some lead-zinc deposits, particularly those of the deep-vein zone or high-temperature types.
Bornite	Cu_5FeS_4	Occurs in some veins of lead and zinc, as at Butte.
CHALCOPYRITE	$CuFeS_2$	Primary mineral of wide-spread occurrence. Found in most lead-zinc deposits, sometimes in commercial quantities.
PYRITE	FeS_2	Very common primary mineral. Occurs in nearly all lead-zinc ores. Crystallizes in the isometric system.
Marcasite	FeS_2	Orthorhombic form of FeS_2. Found in deposits of low-temperature type, as those of the Tri-State district; easily altered.
Arsenopyrite	FeAsS	Primary mineral frequently associated with lead-zinc ores formed under deep-vein zone conditions.

TABLE 3.—*Minerals occurring in lead and zinc deposits*—Continued

Mineral	Composition	Remarks
Associated metallic minerals—Continued		
Pyrargyrite	Ag_3SbS_3	Dark-ruby silver ore. Source of silver in many argentiferous lead-zinc veins. Often primary, sometimes secondary in enriched zones
Proustite	Ag_3AsS_3	Light-ruby silver ore. Source of silver in many argentiferous lead-zinc veins. Often primary, sometimes secondary in enriched zones.
Tetrahedrite	$Cu_8Sb_2S_7$	Often contain considerable silver. Important constituents of many silver-lead ores. Characteristically of primary origin.
Tennantite	$Cu_8As_2S_7$	
Stephanite	Ag_5SbS_4	Occurs in silver-lead veins, both as a primary and a secondary mineral.
Polybasite	Ag_9SbS_6	Found in lead-silver lodes; usually a primary mineral.
Pearceite	Ag_9AsS_6	Sometimes an important source of silver in complex ores; both primary and secondary.
Enargite	Cu_3AsS_4	Occurs in some complex lead-zinc ores.
Cerargyrite	$AgCl$	Horn silver. Always a secondary (supergene) mineral confined to the oxidized and enriched upper parts of silver-bearing lodes.
Embolite	$Ag(Cl,Br)$	Always secondary (supergene) minerals confined to the oxidized and enriched upper parts of silver-bearing lodes.
Bromyrite	$AgBr$	
Magnetite	Fe_3O_4	Uncommon in lead-zinc deposits; occurs in some deposits of the high-temperature type.
Specularite	Fe_2O_3	Uncommon in lead-zinc ores. Usually confined to high-temperature lodes.
LIMONITE	$2Fe_2O_3 \cdot 3H_2O$	Common mineral in oxidized parts of lead-zinc ore bodies. With quartz it is the chief constituent of gossans.
Brochantite	$CuSO_4 \cdot 3Cu(OH)_2$	Rare; occurs in the oxidized parts of some copper-bearing lead-zinc deposits.
Malachite	$CuCO_3 \cdot Cu(OH)_2$	Occurs in oxidized zones of many lead-zinc bodies which contain some copper.
Azurite	$2CuCO_3 \cdot Cu(OH)_2$	Occurs in oxidized zones of some copper-bearing lead-zinc ore bodies.
Nonmetallic gangue minerals:		
FLUORITE	CaF_2	Common gangue mineral in lead-zinc deposits; in a few cases it may occur in paying quantities.
QUARTZ	SiO_2	Almost always present in greater or less amount in the gangue; especially abundant in vein deposits.
CHERT	SiO_2	Also called flint; common in many deposits in carbonate rocks, especially in the Mississippi Valley districts.
JASPER	SiO_2	Impure form of silica; especially common in some of the Mississippi Valley mines.
CALCITE	$CaCO_3$	Almost always present in lead-zinc ores; often the chief constituent of the gangue.
DOLOMITE	$(Ca,Mg)CO_3$	Common and wide-spread as a gangue mineral in lead-zinc deposits.
SIDERITE	$FeCO_3$	Common gangue mineral; often the most abundant mineral in the Coeur d'Alene deposits.
Rhodochrosite	$MnCO_3$	Common gangue mineral, especially in vein deposits, as at Butte, Mont.
Aragonite	$CaCO_3$	Orthorhombic $CaCO_3$; found in vugs, cavities, etc., in some deposits in limestone.
Witherite	$BaCO_3$	Occasionally found in limestone deposits.
Feldspar group	Polysilicates of Al with K, Na, or Ca.	Occur in the gangue of some lead-zinc deposits of the deep-vein or high-temperature types. The commonest varieties are orthoclase ($KAlSi_3O_8$) and albite ($NaAlSi_3O_8$).
Pyroxene group	Metasilicates of Ca, Mg., Fe, and Mn.	Occur in some high-temperature or intermediate vein-zone deposits. Commonest forms are augite $Ca(Mg,Fe)(SiO_3)_2$ with $(Mg,Fe)(Al,Fe)_2SiO_6$, diopside ($CaMg(SiO_3)_2$), rhodonite ($MnSiO_3$), and hedenbergite ($CaFe(SiO_3)_2$).
Amphibole group	Metasilicates of Ca, Mg, Fe, Mn, Na, K, and H.	In some deposits formed at high temperatures. Commonest varieties are tremolite ($CaMg_3(SiO_3)_4$); actinolite ($Ca(Mg,Fe)_3(SiO_3)_4$); and hornblende ($Ca(Mg,Fe)_3(SiO_3)_4 + Na_2Al_2(SiO_3)_4 + (Mg,Fe)_2(Al,Fe)_4Si_2O_{12}$).
Garnet group	Orthosilicates of Ca, Mg, Fe or Mn, and Al, Fe, or Cr.	Occur in some high-temperature deposits.
Andalusite	$(AlO)AlSiO_4$	In a few deposits of high-temperature origin; in some of the Coeur d'Alene ore bodies.
Sillimanite	Al_2SiO_5	

TABLE 3.—*Minerals occurring in lead and zinc deposits*—Continued

Mineral	Composition	Remarks
Nonmetallic gangue minerals—Continued		
Tourmaline	Complex; in general, $H_9Al_3(B \cdot OH)_2Si_4O_{19}$.	In some high-temperature deposits; found in the Coeur d'Alene ores.
Mica group	Hydrous silicates of Al, K, Na, Mg, Fe, etc.	Found in many ores, usually in small amounts. Commonest forms are muscovite (sericite) ($H_2KAl_3(SiO_4)_3$) and biotite, $(H,K)_2(Mg,Fe)_2(Al,Fe)_2(SiO_4)_3$.
CHLORITE GROUP	Essentially $H_8Mg_5Al_2Si_3O_{18}$	The chlorites are common in ore deposits, especially as constituents of gouge and altered country rock.
Serpentine	$H_4(Mg,Fe)_3Si_2O_9$	Important gangue mineral in some deposits (as Edwards, N.Y.) and common in minor amounts in others, being an alteration product of earlier minerals.
Talc	$H_2Mg_3(SiO_3)_4$	Common in a few deposits, as at Edwards, N.Y. Often occurs in gouge and altered country rocks.
Glauconite	Hydrous silicate of Fe and K	Common in some partly altered rocks containing lead-zinc ores.
Kaolinite	$H_4Al_2Si_2O_9$	Principal component of clay; common in oxidized zone of ore deposits, in fault breccia, etc.
BARITE	$BaSO_4$	Common gangue mineral in lead-zinc deposits.
Anhydrite	$CaSO_4$	Occurs in some lead-zinc deposits, usually in limestone and some distance below the surface.
Gypsum	$CaSO_4 \cdot 2H_2O$	Common; may form by hydration of anhydrite or by decomposition of pyrite in limestone rocks; sometimes a primary mineral.
Jarosite	$K_2Fe_6(OH)_{12}(SO_4)_4$	Uncommon; occurs in secondary ores in Tintic, Utah, and elsewhere.

CHARACTER OF OUTCROPS

The outcrops of lead-and-zinc ore bodies differ rather widely in aspect and general character, according to variations in the type of mineralization of the primary ores, the climate, the topography, and the general geologic conditions. The two most important factors are the relative susceptibility to weathering and erosion of the ore and of the enclosing country rock and the nature and visual appearance of the minerals in the ore after they have been altered by oxidation and other weathering processes.

If the ore contains a quartz gangue and occurs in limestone or other easily eroded rock the outcrop is apt to be conspicuous, especially in regions of rugged topography and arid climate; under such conditions erosion tends to keep pace with weathering, and the disintegrated surface material is removed about as rapidly as it is formed, leaving the more resistant outcrop in bold relief. If the ore also contains much pyrite or other iron sulphide the outcrop will often consist of a conspicuous mass of honeycombed quartz stained with yellow or brown limonite; it is then called a gossan or "iron hat." Massive sulphide ore containing little quartz often has a heavy gossan outcrop composed chiefly of iron oxide. In some places there may be enough copper in the primary ore to color the outcrop green.

In districts like Missouri and Wisconsin, where the flat-lying ore bodies are not more resistant than the silicified and dolomitized enclosing carbonate country rock, the climate is relatively moist, and the terrain is flat, outcrops are often inconspicuous and are frequently masked by a thick overburden of soil and unconsolidated material.

Frequently ore bodies are less resistant to erosion than the surrounding rocks and are represented at the surface by small gullies or depressions. In flat country such depressions may be occupied by swamps, while in hilly country they catch loose rocks, boulders, and soil from the slopes above, the outcrops thus becoming in many instances effectually covered.

Zinc blende is, relatively speaking, easily dissolved by surface waters and is not commonly found in the outcrop except in regions of very rapid erosion or where glaciation has planed off the surface. Furthermore, its oxidization products (principally smithsonite, calamine, and hydrozincite) have usually been transported down the structure and dissipated or redeposited in depth; therefore, few zinc deposits appear as such at the surface, usually being opened at depth in the mining of lead. Some notable exceptions to this condition, however, are Leadville, Tintic, and the Cerro Gordo vein in California.

Galena is one of the most inert common sulphides and is not readily attacked by weathering agencies. It is frequently found at or near the surface in lode deposits and has even remained in a relatively unaltered condition in the disintegrated and weathered material forming the overburden of the deposits. Mines have been located in Mississippi Valley districts as a result of lumps of galena being plowed up by farmers. In numerous western camps the finding of galena in pieces of float on the slopes below hidden outcrops has led prospectors to trace out the original source in place.

As with other types of deposits, float is one of the most valuable surface indications of lead and zinc ore, whether it contains traces of these metals or is merely honeycombed iron-stained quartz from which the valuable minerals have been leached.

In numerous instances bodies of lead and zinc ore do not outcrop but occur "blind", apexing below the surface. Such deposits must be found by surface drilling or by underground prospecting from workings in known ore, using the best geological information obtainable.

CHANGES WITH INCREASING DEPTH

Changes near surface.—Changes in lead and zinc deposits near the surface are indicated by the character of the "leached zone." Zinc is usually removed entirely by downward migration in solution or is present in greatly reduced amounts, except sometimes in glaciated regions or regions where erosion is very rapid and keeps up with weathering. Lead, being much less soluble, is apt to persist near the surface and to be concentrated with respect to the original ore by removal of zinc, carbonates, and other soluble minerals and by alteration of pyrite to limonite with consequent reduction in volume. When oxidized to cerussite or anglesite, lead ore usually does not move far from its original position, except as it is lowered in the disintegrated surface capping.

Changes in the oxidized zone.—Below the leached zone and above the ground-water level there is frequently a zone in which oxidation has taken place without extensive removal of the metals. In lead and zinc deposits this zone contains such oxidized minerals as smithsonite, calamine, hydrozincite, cerussite, anglesite, pyromorphite, and partly oxidized galena; if silver was present in the primary ore it may be found in the oxidized zone in the form of native silver, cerargyrite (horn silver), embolite, and similar minerals.

Where oxidized, lead usually is found near its original location. Zinc, on the other hand, tends to migrate in the oxidized zone and is redeposited as the carbonate or other oxidized mineral around the edges and bottom of the original deposit. In some formations, which are permeable or which contain cavities, breccia zones, or solution channels, it may move considerable distances before being reprecipitated. It is thus often characteristic of lead-zinc deposits that the metals tend to become separated in the zone of oxidation.

Complete oxidation is generally not found below water level, unless the water level has been raised after the primary ores have been altered.

Oxidation is deepest with ores containing carbonates and depends further on the depth of the water table, the climate, and the degree of fracturing, brecciation, or other conditions of permeability.

Changes below water level.—Lead is altered very little below water level; but zinc and iron sulphides will, under some conditions, dissolve at depth below the water table, as they are attacked by acids in the absence of air or other oxidizing agents.

The phenomenon of secondary sulphide enrichment, which occurs at and below water level and is of such paramount importance in many copper deposits, is of small moment in lead and zinc deposits insofar as these metals themselves are concerned. With respect to the silver associated with many such ores, however, sulphide enrichment has considerable economic significance, and in certain districts it has resulted in so great a concentration of silver in the secondary zone as to mean the difference between unsuccessful and highly remunerative exploitation of the deposits. Silver dissolved in the upper portions of the deposit is carried downward in solution and precipitated below water level as argentite or various minerals of the long series of sulphosalts, such as proustite, polybasite, stephanite, and pearceite. Lead is almost never deposited as the secondary sulphide and does not move far in any form.

Zinc is sometimes redeposited in the zone of enrichment as secondary blende, more often as the hexagonal sulphide wurtzite. Since it is soluble in acid solution in the absence of oxygen, however, and since many lead-zinc ores occur in carbonate rocks, the enrichment of zinc ores is effected most frequently by reprecipitation as zinc carbonate.

The enrichment zone below water level may be of comparatively small vertical extent or it may extend to considerable depths, depending on the degree of fracturing in the rocks. In some districts secondary ores occur at greater depths than 2,000 feet.

LOCALIZATION OF ORE SHOOTS

The term "ore shoots" is applied in this paper in its broader sense to portions of mineral deposits in which valuable minerals are concentrated and abundant enough to be commercially exploited.

The subject has been exhaustively treated by Lindgren,[4] Ries,[5] Emmons,[6] and many others. Bibliographic references to the most important writings are given in the works of these authors. In the present paper only the broader aspects of the subject and those having a direct bearing on problems of prospecting, development, and mining will be discussed.

Favorable beds.—In many districts it has been learned by mining experience that ore shoots are largely confined to certain beds or members of the formations present in the area and are poor or lacking in others. Because these formations have such physical characteristics as porosity, wide-spread fracturing, or other features favorable to the passage of mineralizing solutions, or such chemical properties as easy solubility, they tend to localize ore shoots. When a phenomenon of this kind becomes firmly established in a district, the problem of exploration and prospecting is greatly simplified, and efforts can be directed to intensive investigation of favorable areas. Thus, in southeastern Missouri drill holes are put down to the known productive horizon, and no money is wasted in sinking them deeper into underlying beds known to be barren. In some Utah districts where the ores are irregular replacements in limestones, underground exploration is confined largely to beds known as favorable. In some camps, finding faulted favorable beds is a necessary prerequisite to finding ore.

Dolomitization.—Hewett and others [7] have pointed out that many lead-zinc replacement deposits in carbonate rock formations occur in dolomitized areas. In some districts the ore bodies are so characteristically confined to dolomitized portions of the beds that areas not so altered can be disregarded in exploration. While dolomization in itself is not believed essential in providing conditions favorable to ore deposition, the fact that the solutions which caused it probably gained access to the original limestone beds through the same feeder channels that facilitated subsequent introduction of ore-bearing fluids and moreover penetrated as far or farther into the surrounding rocks than the ore solutions did, designates the dolomitized areas as limiting areas beyond which ore is unlikely to be found.

Structural control.—The influence of geologic structures in localizing ore shoots has long been known. The most important of these are faults, brecciated and shear zones, contacts, joints and sheeted areas, intersections of fissures, and folds.

Faults often serve as conduits through which ore-bearing solutions were introduced. In some districts, such as the Coeur d'Alene region, very large and extensive faults do not contain ore but were responsible for smaller faults nearby in which ore is found. In other districts the reverse is true. In still others, faults filled with clayey

---

[4] Lindgren, Waldemar, Mineral Deposits: McGraw-Hill Book Co., New York, 3d ed., 1928, 1049 pp.

[5] Ries, H., Economic Geology: John Wiley & Sons, New York, 5th ed., 1925, 843 pp.

[6] Emmons, W. H., Principles of Economic Geology: McGraw-Hill Book Co., New York, 1918, 606 pp.

[7] Hewett, D. F., Geology and Ore Deposits of the Goodsprings Quadrangle. Nevada: U.S. Geol. Survey, Prof. Paper 162, 1931, 172 pp.

Fowler, F. M., and Lyden, J. P., Ore Deposits of the Tri-State District, published as a section in The Story of the Tri-State District, a souvenir booklet for the joint convention of the American Mining Congress and the American Institute of Mining and Metallurgical Engineers: Joplin Printing Co., Joplin, Mo., September 1931, 43 pp.

gouge acted as dams to solutions moving along other channels and were instrumental in localizing deposition of ore at the intersections of these channels with the fault.

Brecciated and sheared zones not only afford access to mineralizing agencies but also provide abundant open space or crushed, altered, and easily replaced material, or both, in which conditions favor deposition. Many of the largest deposits occur in such ground.

Formational contacts, such as those between sedimentary beds and between beds and dikes or intrusive masses, may provide channels for movement of solutions; moreover, because of differences in the physical or chemical nature of contiguous rocks they may induce the deposition of ore. In addition, when one of the rocks involved in a contact is igneous, its genetic relation to the ore as a probable or possible source of the mineralizing solutions becomes a factor. Although many lead and zinc deposits are of the low or moderate temperature type, some grade with depth into the deep-vein or high-temperature replacement types, and a few zinc deposits are of distinctly contact metamorphic or related origin. Thus, the possibility that an igneous contact may have an important relation to ore shoots must be considered carefully in working out the ore habits of a district.

Joint fissures and sheeted areas (closely spaced parallel fractures) may act as ore channels. Because joints are usually of much more limited extent than faults they are more important in providing space for deposition of ore introduced through other channels than in serving as main conduits. Joints, often enlarged by solution, when mineralized constitute the "gash" veins of Wisconsin and other regions. The borders of many large ore bodies show tongues of ore making off along joints; these structures are important in the formation of large bodies in that they serve as points from which replacement processes are easily started. Sheeted zones or groups of closely spaced joints act similarly, and they are host to important ore deposits.

Intersecting fissures frequently localize ore shoots. Penrose [8] early pointed out their importance. Ore may be deposited as a result of the mingling of unlike solutions at fissure intersections. Such intersections may result in fracturing or weakening of nearby rocks, especially if the fissures meet at sharp angles, rendering them more hospitable to the deposition of ore. Where a fissure intersects a favorable bed, ore may form in the fissure where it traverses the bed; where fissures intersect impervious beds or dikes, ore bodies are sometimes produced by the damming effect of the impervious formations.

Folds sometimes influence the localization of ore shoots. Their crests often represent areas of shattering or opening of the beds, thus affording access to solutions. The Edwards (N.Y.) district affords an example of this sort of structural control, where minor folds on the sides of a major structure have probably opened the beds of crystalline dolomite enough to provide the easiest path for solutions, which in moving up the dip have deposited long lenses of ore along the folds. Folds in conjunction with other structures

[8] Penrose, R. A. F., Jr., Some Causes of Ore Shoots: Econ. Geol., vol. 5, 1910, pp. 97–133.

are important in some districts. Thus, some large ore bodies of the Coeur d'Alene district occur near the intersection of zones of shearing with the axis of a large fold.

The structural features mentioned not only are factors in controlling the location of primary ore shoots, but also may influence the place and mode of deposition of secondary ores; in working out the criteria for ore bodies in a given district the investigator should constantly bear this fact in mind to prevent confusion and the development of incorrect theories, since in the different periods of the geological history of a district changes frequently occur which may entirely alter the effect produced by various structures. Generalities that apply to the effect of geologic structures on ore shoots in one district do not necessarily apply in another. Small differences, or different combinations of conditions, may produce highly divergent results, and it is only by a thorough and detailed study both of the general geology and the local details of individual veins or mines in a particular district that reliable conclusions can be reached as to the meaning of structural features and their relations to ore occurrence.

GEOLOGIC AGE OF LEAD AND ZINC DEPOSITS

Lead and zinc are rare in pre-Cambrian ores in the United States. Most of the deposits that occur in rocks of this age, as, for example, the Coeur d'Alene ores, were deposited in later periods. A few deposits of zinc are pre-Cambrian; of these the ores of the Franklin (N.J.) and Edwards (N.Y.) districts are the outstanding examples in the United States. They were formed under high-temperature conditions.

The most important lead and zinc metallization took place in the Paleozoic period when the great deposits of the Mississippi Valley region were formed.

Most western deposits associated with igneous rocks are of late Cretaceous or early Tertiary age.

Lead ores are characteristic in calcareous or dolomitic rocks and were generally deposited by relatively cold solutions at moderate depth. They are sometimes mined from deposits of the contact metamorphic or deep-vein zone type but are abundant in only a few of these, nearly always being subordinate to other metals.

Zinc likewise displays a preference for carbonate rocks and, while usually associated with lead, is more abundant than lead in high-temperature deposits. Its occurrence at Franklin, Edwards, and elsewhere under these conditions has been mentioned. It is characteristic of lead-zinc veins that zinc becomes proportionately more abundant in depth and lead less so. Many mines originally worked for lead have become essentially producers of zinc as they attained greater depth.

MINING DISTRICTS

In describing the ore deposits of the various districts, the authors have freely drawn upon Bureau of Mines information circulars dealing with mining methods and costs, publications of the United States Geological Survey, and the technical press. Information from such sources has been supplemented by the authors' field observations.

SOUTHEASTERN MISSOURI DISTRICT

As indicated by its name, this district is in southeastern Missouri, the present productive area occupying parts of St. Francois, Iron, and Madison Counties.

The ores occur in sedimentary rocks of Middle Cambrian age, chiefly in the Bonneterre limestone, which rests upon the porous LaMotte sandstone. The Bonneterre consists of about 400 feet of magnesian limestone, some of it argillaceous and interbedded with dark gray and black shale layers. Near the base of the formation where the shale layers are most numerous considerable glauconite is also found. The Bonneterre formation is capped by the Davis shale, which is about 160 to 200 feet thick. Figure 2 is a pictorial cross-section of the geological formations and shows the nearly horizontal position of the beds, which are displaced by two prominent faults; locally, the beds dip as much as 20°.

Galena is the principal ore mineral, but at a few places there is enough sphalerite to make it profitable to produce both lead and zinc

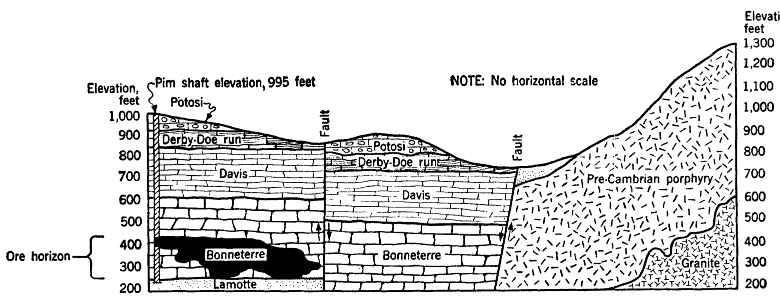

FIGURE 2.—Cross-section of geological formations from Flat River south to Simms Mountain, southeastern Missouri.

concentrates. At other places no zinc is found, and on the whole the district may be said to produce straight lead ores. The ore is found principally in the lower 100 or 150 feet of the Bonneterre limestone, although it occurs at various horizons from top to bottom of the formation. In some places the workable horizons are separated by barren or very low grade rock, and in others the deposits are workable over continuous heights of 200 or more feet.

The galena occurs disseminated through dolomitic limestone and shaly beds, in horizontal sheets along bedding planes, in vugs and cavities, filling or lining the wall of joints and crevices, and as aggregates of cubes in channels and joints, and it appears to favor the darker coarse-textured beds and the more shaly parts of the formation. The shale layers, which are usually thin, are often impregnated with galena, while the lime beds on either side may be only lightly sprinkled with the mineral. (See fig. 3.)

Systems of roughly parallel fracture zones occur in the upper horizons, individual fractures being thin, leached, and whitened by the action of surface waters. These are often mineralized and frequently have workable ore associated with them.

So-called water channels cut across the beds in various directions but strike predominantly in one direction in any given area. These channels sometimes start from the surface or in the overlying shale, pass into the limestone, and then gradually die out at depth. Others pass from the Bonneterre into the underlying LaMotte sandstone. The larger water channels seldom carry ore in commercial quantities; however, sheets of galena and aggregates of crystals are often found in them, embedded in reddish or brownish mud and decomposed dolomite, the usual filling of these channels.

Figure 4, *A*, is a plan of an ore body which shows the irregularity of outline typical of these deposits. Figure 4, *B*, gives cross-sections of the same ore body showing the different workable horizons indicated by the mine workings. In this mine stoping widths range from a few feet along some of the upper narrow fractures to several hundred feet. Some ore bodies are 700 to 800 feet wide by over 1,200 feet long and range from a few feet to over 200 feet in thickness. In some other parts of the district the ore bodies are almost continuous for several miles in length. The strength of the formation in different localities and in different sections of the same mine varies considerably, but on the whole the formation is very strong and will often stand over spans of 100 feet between walls or pillars. The wall rocks have the same character as the ore; and the ore-bearing areas pass laterally into barren rock, in some places abruptly and at other places gradually.

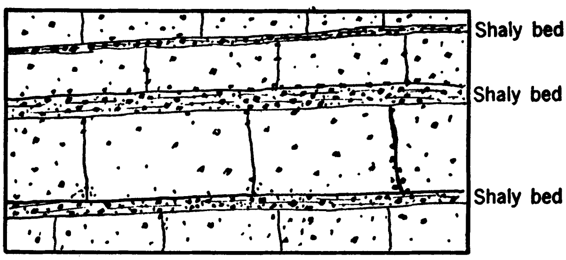

FIGURE 3.—Typical mineralization in 10-foot face of ore, southeastern Missouri.

The first ore mined in the district came from shallow surface diggings and was found as concentrations in pockets and crevices. Most of the present production comes from disseminated deposits, the bottom of the ore being usually less than 750 feet below the surface.

The ores are relatively low grade, yielding an average recovery of about 3.5 percent metallic lead, although often they are richer.

The mineralogy of the ores is simple, consisting of galena in a limestone gangue with varying amounts of calcite; small percentages of pyrite, marcasite, and chalcopyrite; and traces of cobalt and nickel, probably in the form of sulphides. The ores are classed as soft lead ores and contain only a small quantity of silver.

TRI-STATE DISTRICT

This district comprises a section as much as 30 miles wide in places and nearly 100 miles long on the northwest flank of the Ozark uplift and occupies parts of Oklahoma, Kansas, and Missouri; the principal ore production has come from an area about 35 miles long by 10 miles wide.

Virtually all the ore deposits are in the Boone formation, Lower Mississippian age. This formation, believed to have been a limestone originally, is now made up of flat-lying beds of limestone, dolomite, and chert; nodule beds; and one or more oölite beds. Fowler and Lyden [9] state that the exploited part of the Boone formation has been divided into 16 distinct beds, ranging in thickness from 4 to 55 feet, and that 6 of these provide the principal ore-bearing horizons, 2 others carry some ore, while ore may occur in certain others in zones of intense shattering. The principal ore beds are characterized by nodules of distinctive size and shape.

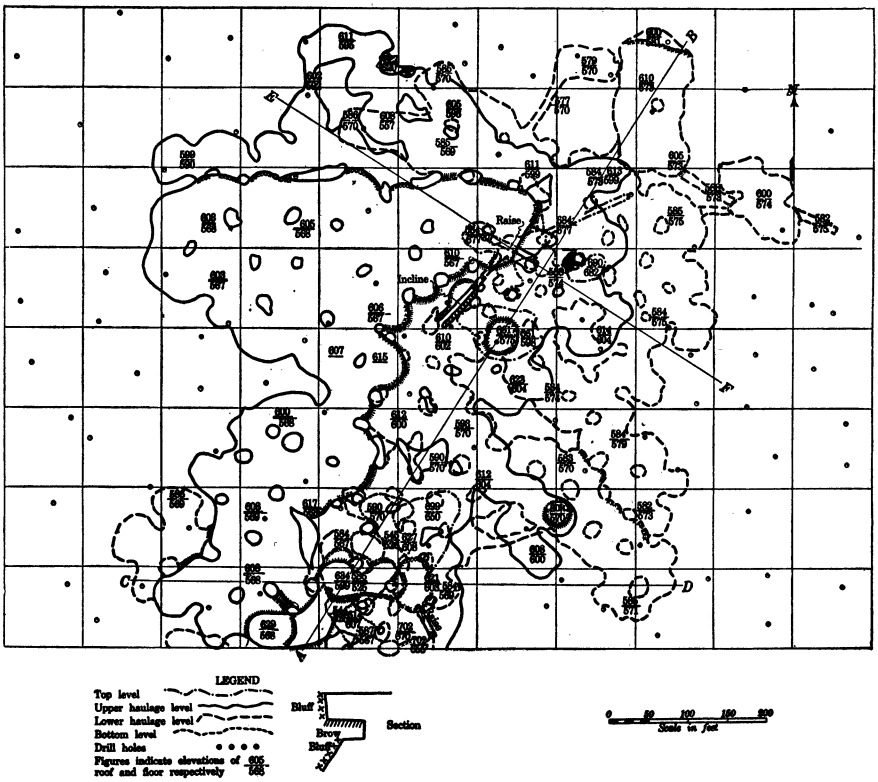

FIGURE 4A.—Plan of mine workings, Southeastern Missouri lead district, showing irregular outline of ore bodies.

All the ore deposits are related to structural features and occur in zones of shearing, shattering, and brecciation and in places where there has been considerable silicification of the original limestone. They lie for the most part within a horizon 200 feet thick in the Boone formation, although a few small, high-grade deposits have extended upward into the overlying Chester formation. The ore deposits occur at shallow depths, the bottom seldom being more

[9] Fowler, F. M., and Lyden, J. P., Ore Deposits of the Tri-State District; published as a section in "The Story of the Tri-State District", a souvenir booklet for the joint convention of the American Mining Congress and the American Institute of Mining and Metallurgical Engineers: Joplin Printing Co., Joplin, Mo., September 1931, 43 pp.

than 350 feet and usually not over 300 feet below the surface. The top of the favorable formation is usually 95 to 175 feet below the surface.

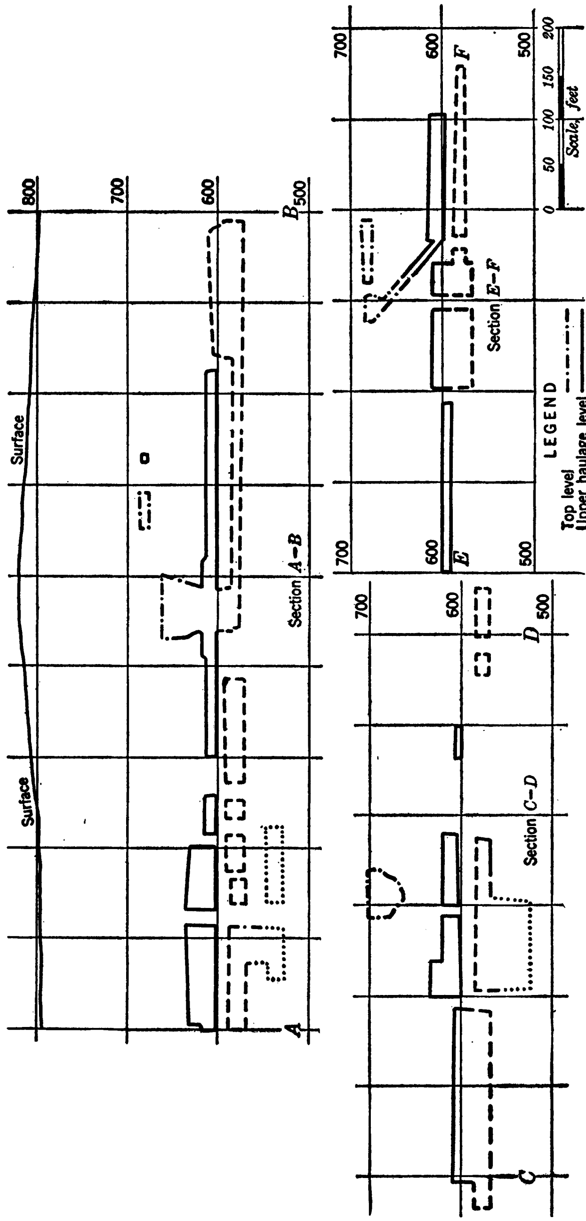

FIGURE 4B.—Cross-sections through workings shown in A, showing ore horizons.

According to Fowler and Lyden[10] a favorable "reservoir" for ores may be found in (1) porous formations which can be replaced, (2) cavernous formations, (3) shattered or brecciated zones or open

[10] Fowler, F. M., and Lyden, J. P., work cited.

fissures, and (4) any combination of these. The formations between the ore beds generally are massive limestone, dolomite, or chert. Where shattering has been intense enough the horizons between the ore beds are sometimes mineralized. In a few places, such as along major shear zones, the mineralization has a vertical range of approximately 150 feet. In these places the ore generally is confined to the shear zones and does not extend beyond into the contiguous strata.

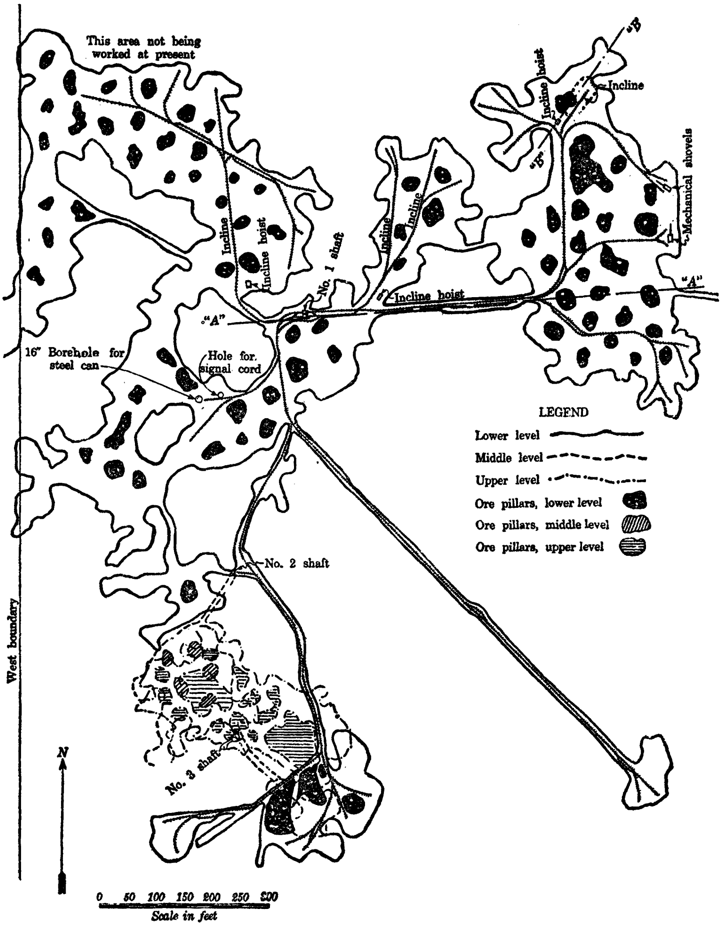

FIGURE 5.—Plan of Barr mine, Tri-State district, showing irregular outline of ore bodies.

The ore beds are 5 to 22 feet thick. Only part of the bed may contain commercial ore, mineralization depending upon the extent to which structural conditions are favorable. These favorable zones may be the top, bottom, or flanks of domes, synclines, or other flexures. The lateral boundaries of the ore bodies are very irregular (fig. 5), and the line of demarcation between ore and waste may be very sharp. In many instances, however, the transition is gradual.

There are three different types of ore bodies: (1) The brecciated and boulder-ground type, the most important; (2) the sheet-ground type found in the Grand Falls chert, a member of the Boone formation; and (3) the soft-ground type, found in the upper part of the Boone formation and in the overlying Chester formation. The latter type is no longer important as a source of ore, as it was mined

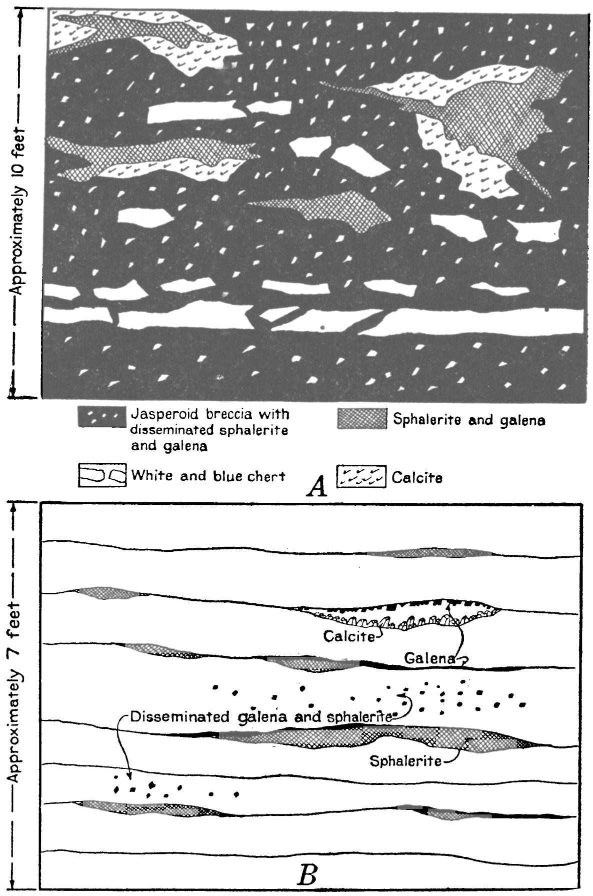

FIGURE 6.—*A*, Typical mineralization in face of brecciated ore body, Tri-State district; *B*, ore body of semisheet, ground type, Tri-State district.

out in the early days; such ore usually contained a higher percentage of lead than the other types.

The brecciated and boulder ground contains the most important ore bodies in the Kansas-Oklahoma area. The ore minerals, sphalerite and galena, occur as seams and veinlets in cracks or fissures in the beds, along bedding planes, disseminated in jasperoid breccia, as irregular patches between boulders, or cementing boulders and badly brecciated material. Figure 6, *A*, shows typical mineralization of a face of ore. The gangue, consisting principally of country rock, often contains considerable calcite and dolomite. Marcasite is found in nearly all parts of the Picher-Miami district, while pyrite and chalcopyrite occur sparingly. In some deposits of this type the mining thickness attains 60 or more feet.

The sheet-ground deposits are thin but often extend with remarkable persistence over wide areas. They are usually of low grade, the ore minerals occurring as lenses along bedding planes and lining vugs and small caves in the chert. Figure 6, *B*, shows typical mineralization in an ore body of the semi-sheet-ground type.

Grade of ore.—Statistics from 1917 to 1929, inclusive, show that 6,504,369 tons of zinc concentrates and 1,221,823 tons of lead concentrates were recovered from 117,401,237 tons of mine rock. In the Tri-State district it is customary to express recovery figures in terms of concentrates rather than in terms of metallic content. No precise figures on the metallic content of the ores are available; however, the selling prices of zinc and lead concentrates are based upon 60 percent metallic zinc in the zinc concentrates and 80 percent metallic lead in the lead concentrates with penalties for lower percentages. The foregoing figures indicate an average recovery of 5.54 percent zinc concentrates and 1.04 percent lead concentrates; if it is assumed that shipments for the years in question were approximately of the base grades, which is believed to be approximately correct, an average of 3.324 percent metallic zinc and 0.624 metallic lead would be the indicated recovery. About 1 percent metallic zinc is lost in the tailings; thus the crude ore averaged about 4.32 percent metallic zinc.

EASTERN TENNESSEE DISTRICT

The principal developed deposits of zinc ore in this district are in a belt about 40 miles long and 1 mile wide in Knox and Jefferson Counties and in Claiborne County about 35 miles north of Knoxville. Similar deposits found in southwestern Virginia may represent an extension of the same belt.

The ore bodies at Mascot, 14 miles northeast of Knoxville, have been the most extensively developed, and a brief description follows.

The rocks are limestones, dolomites, and calcareous shales, striking 50° to 75° east of north and dipping 18° to 22° southeast. The ore-bearing horizon is found in the Knox dolomite [11] about 1,000 feet structurally below the top of that formation, and as far as now known the ores are confined to beds of this horizon.

The ore occurs largely as veinlets and seams of sphalerite with secondary dolomite in the dolomitic limestone. Locally there are thin shale partings between beds, but these are not common.

Prior to 1900 a small output of oxidized ore was obtained from shallow pits at the outcrop, but sulphides occurring at greater depth down the dip of the formation are now being mined. Figure 7, *A*, is a plan of the ore body and mineralized areas as indicated by prospect drilling at Mascot. The thickness of the ore varies greatly; abrupt changes from a few feet to 50 or more are not uncommon, and in places the ore is 150 feet thick.

The ore is of low grade, averaging 2.90 percent zinc for 1929.[12] The sphalerite is exceptionally pure, however, and is not con-

---

[11] Smith, Arthur, Geologic Atlas of the United States: U.S. Geol. Survey, Folio 75, Maynardville, Tenn., 1901, 6 pp., 4 maps.

[12] Strachan, C. B., Milling Methods of the American Zinc Co. of Tennessee, Mascot, Tenn.: Inf. Circ. 6379, Bureau of Mines, 1930, 13 pp.

taminated with other metallic minerals. A complete percentage analysis of the ore follows:

Calcium carbonate	48.11	Iron oxide	1.33
Magnesium carbonate	35.36	Zinc sulphide	5.43
Silica	8.73		
Alumina	1.04	Total	100.00

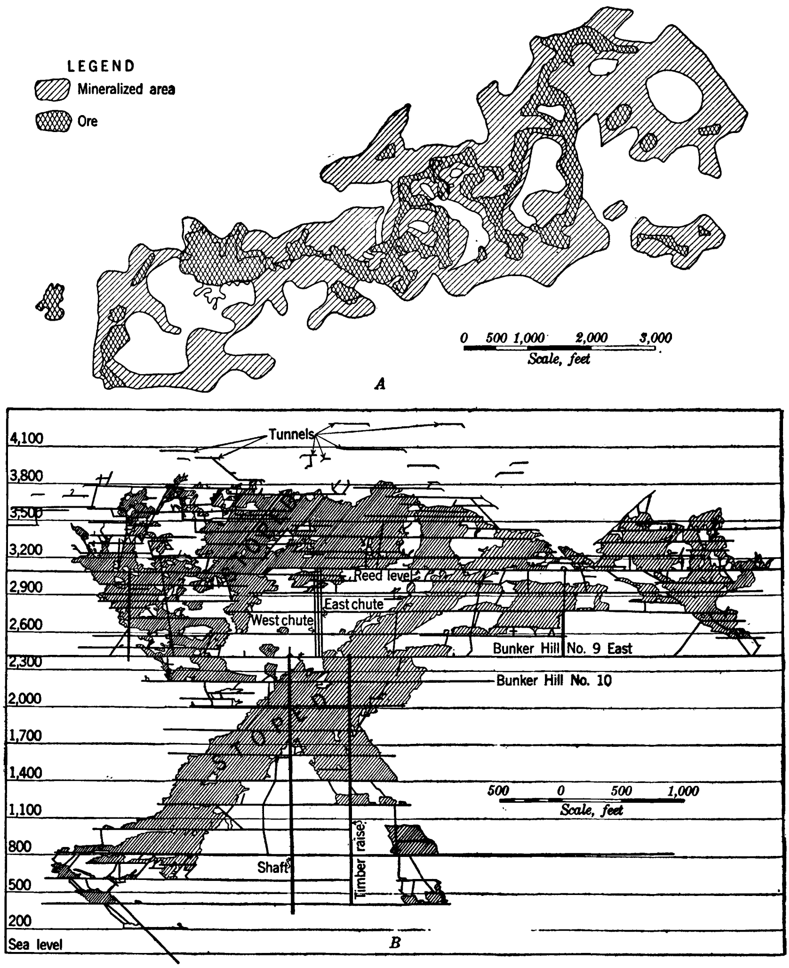

FIGURE 7.—*A*, Mascot (Tenn.) ore body as indicated by prospect drilling; *B*, vertical longitudinal section, Bunker Hill & Sullivan mines, Coeur d'Alene district, Idaho, Jan. 1, 1932. Stoped areas show outlines of ore bodies.

COEUR D'ALENE DISTRICT, IDAHO

The Coeur d'Alene region, in Shoshone County, northern Idaho, includes the following local organized mining districts: Beaver, Evolution, Hunter (Mullan), Leland (Burke), Pine Creek, Placer Center (Wallace), Slate Creek, Summit, and Yreka (Wardner).

The region lies on the western slope of the Coeur d'Alene Mountains, which are the result of deep dissection of a high plateau sloping gently north and northwest. The topography is characterized by narrow westward-draining valleys lying between steep ridges, and rapid changes in relief constitute a prominent feature of the region. Altitudes range from about 2,150 feet in the valleys at the western edge of Shoshone County to about 6,900 feet at the tops of some of the highest peaks. Heavy forests, or later underbrush where fires and logging have denuded the slopes of first-growth timber, add to prospecting difficulties.

The rocks of the district consist chiefly of the Belt series of pre-Cambrian sediments, largely quartzites, shales, and some calcareous beds, which have been laid down conformably to a thickness of about 17,000 feet. The sedimentary section, given by Ransome and Calkins,[13] follows:

Generalized tabular section of Algonkian rocks in the Coeur d'Alene district

No.	Name	Approximate thickness, feet	Description
6	Striped Peak formation	1,000+	Sandstones, siliceous, generally flaggy to shaly; colors mostly green and purple; characterized by shallow-water features, such as ripple marks, sun cracks, etc. Top removed by erosion.
5	Wallace formation	4,000	Thin-bedded, bluish and greenish, more or less calcareous shales, underlain by rapidly alternating thin beds of argillite, calcareous sandstone, and impure limestone; these underlain in turn by gray-green siliceous argillites. Ripple marks, sun cracks, etc., throughout.
4	St. Regis formation	1,000	Indurated shales and more or less flaggy sandstones; colors mostly green and purple; characterized by shallow-water features.
3	Revett quartzite	1,200	White quartzites, generally rather thick bedded; interstratified with subordinate quantities of micaceous sandstone.
2	Burke formation	2,000	Light-gray, flaggy, fine-grained sandstones and shales, mostly greenish, with a variable amount of purple quartzitic sandstone and white quartzite. Shallow-water features throughout.
1	Prichard slate	8,000+	Mostly argillite, blue-black to blue-gray, generally showing distinct and regular banding. Considerable interbedded gray indurated sandstone. Upper portion characterized by numerous alterations of argillaceous and arenaceous layers and by shallow-water features. Base not exposed.
		17,200+	

Monzonites and syenites have been intruded into some of the sediments as irregular masses, dikes, and sheets and are regarded as the probable source of the solutions that deposited the ore. Many small diabase and lamprophyre dikes occur, but these have no apparent connection with ore formation.

The rocks of the area have been subjected to intense folding and faulting, so that the structure is often very complex. The Osburn fault, which crosses the district from east to west, is estimated to have caused a horizontal displacement or shift of the rocks of the south side, relative to the north, of 12 miles. A downthrow of the south side is believed to range from 1,000 to 10,000 feet. Other large faults are known, but none approaches the Osburn in magnitude. The major faulting was accompanied by the development of less pronounced fault fissures and zones of shearing, and it is in these

[13] Ransome, F. L., and Calkins, F. C., The Geology and Ore Deposits of the Coeur d'Alene District, Idaho: U.S. Geol. Survey Prof. Paper 62, 1908, p. 25.

structures that most of the ore deposits are found. According to Umpleby and Jones [14] the principal deposits are situated near the intersections of a broad zone of extensive faulting with an axis of anticlinal uplift and igneous intrusion.

The ores are found chiefly in the Burke and Revett quartzites and in the Prichard slates, but they are also developed in the other rocks of the Belt series.

The ore bodies are mainly fissure veins or lodes and were formed largely by replacement of the brecciated and sheared rock in the fissures or sheeted zones. In most deposits the quartzite was first replaced by siderite, which in turn was replaced by quartz, sphalerite, and silver-bearing galena. Gradation into deposits of the contact-metamorphic type is noted in a few mines.[15]

Mineralogically, the ores usually consist of galena, blende, pyrite, and chalcopyrite in variable amounts in a gangue of siderite and quartz. The oxidized upper portions of the lodes carry principally quartz, limonite, and the lead carbonate cerussite. Oxidized zinc minerals have no commercial importance, and most of the oxidized lead-silver ore has now been mined out.

Favorable conditions for deposition of ore were developed by fissuring, sheeting, and fracturing of the rocks in broad zones of faulting, and ore deposition seems to have depended on suitable physical structure rather than on the chemical or lithologic nature of the enclosing rocks.

As depth is attained in many mines zinc becomes a more abundant constituent of the ore and lead less so. Some mines have been developed to depths more than 5,000 feet below the outcrops without serious diminution in the grade or width of ore. Outcrops are frequently inconspicuous.

The ore shoots in the Coeur d'Alene district are characteristically persistent and of large size and have produced some of the great mines of the world, such as the Bunker Hill & Sullivan, Morning, and Hecla and Star. Thus, the main ore zone in the Morning mine is 1,500 to 2,000 feet long and is practically continuous in depth for 5,000 feet below the outcrop, with the bottom level still in ore.[16] This vein carries stoping widths of 6 to 30 feet, averaging about 13 feet.

The Bunker Hill ore bodies occur as wide, irregular masses within a sheared area 1,000 feet wide by 6,000 feet long, bounded by major faults. According to Brown [17] the ore dips 40° to 50° and usually has one fairly well defined wall. Figure 7, *B*, is a vertical longitudinal section of the main part of the Bunker Hill & Sullivan mine, as of January 1, 1932.

Most ore bodies in the district have irregular borders, caused by the mineralization making off into the fractured wall rocks, but some fill relatively tight fissures and exhibit more clear-cut boundaries. Intense fracturing in the ore zones and alteration of the rocks, with

[14] Umpleby, J. B., and Jones, E. L., Geology and Ore Deposits of Shoshone County, Idaho: U.S. Geol. Survey Bull. 732, 1923, p. 17.

[15] Umpleby, J. B., and Jones, E. L., work cited, pp. 32–38.

[16] Wethered, C. E., and Coady, L. J., Mining Methods at the Morning Mine of the Federal Mining & Smelting Co., Mullan, Idaho: Inf. Circ. 6238, Bureau of Mines, 1930, p. 2.

[17] Brown, U. E., Mining Methods of the Bunker Hill & Sullivan Mining & Concentrating Co., Kellogg, Idaho: Inf. Circ. 6407, Bureau of Mines, 1931, p. 2.

development of more or less gouge, are responsible in many mines for heavy ground conditions which have greatly influenced mining methods in the district.

The grade of ore mined varies at different properties and is affected by economic conditions from year to year. At most mines the ore has the following range in average tenor: Lead, 3.5 to 11 percent; zinc, 1 to 12.5 percent; and silver, 1.0 to 20.0 ounces per ton.

PARK CITY DISTRICT, UTAH

The Park City district[18] is about 25 miles southeast of Salt Lake City and lies on the eastern slope of the Wasatch Mountains at an altitude ranging from 7,000 feet at Park City to well over 8,000 at some nearby mines. Production of silver-lead-zinc ore, as well as some gold and copper, has been continuous since 1872.

The rocks in which the ores chiefly occur are the Carboniferous Wasatch limestone, Weber quartzite, and Park City limestone; also the Thaynes formation, made up of Triassic limestones, sandstones, and shales. Figure 8 is a typical cross-section through the Park Utah

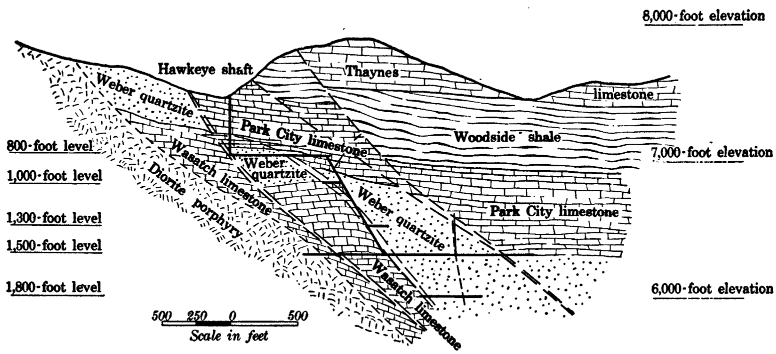

FIGURE 8.—Cross-section through Park Utah mine, Park City district, Utah, looking N. 80° W.

mine showing these formations. The sediments have been intruded by diorite and diorite porphyry, which metamorphosed the enclosing stratified rocks and developed typical contact-metamorphic silicate minerals. The ores are genetically related to these igneous rocks and were deposited in the sediments by hot ascending waters.

The rocks have been uplifted and subjected to considerable folding and faulting. A system of northeast-southwest fault fractures provided channels by which solutions penetrated the sediments and deposited ore (as fissure-lode deposits in the Weber quartzite and to some extent in the overlying limes and as irregular bedded replacement deposits, chiefly in certain favorable beds of the Park City and Thaynes members). Later movement along the northeast-southwest fractures faulted the ore and caused much brecciation within the deposits, and still later fracturing along northwest-southeast-trending lines gave access to downward percolation of surface

[18] Boutwell, J. M., and Woolsey, L. H., Geology and Ore Deposits of the Park City District, Utah: U.S. Geol. Survey Prof. Paper 77, 1912, 231 pp.

waters which were instrumental in the oxidation and enrichment of the upper ore zones.

The ore bodies are of two general types, which frequently grade into each other. The fissure type, found principally in quartzite but to some extent in other rocks as well, is characterized by widths of 3 to 80 feet and dips of 40° to 55° in the Park Utah mine, where lodes of this type supply the bulk of the production.[19] The ore bodies are lenticular and range in length from a few feet to 900. The ore itself consists of a series of parallel sulphide bands alternating with bands of altered limestone.

The bedded replacement type of ore is well typified at Park City in the Silver King Coalition mines,[20] where bedded ores provide 95 percent of the tonnage. At this property the ore bodies are re-

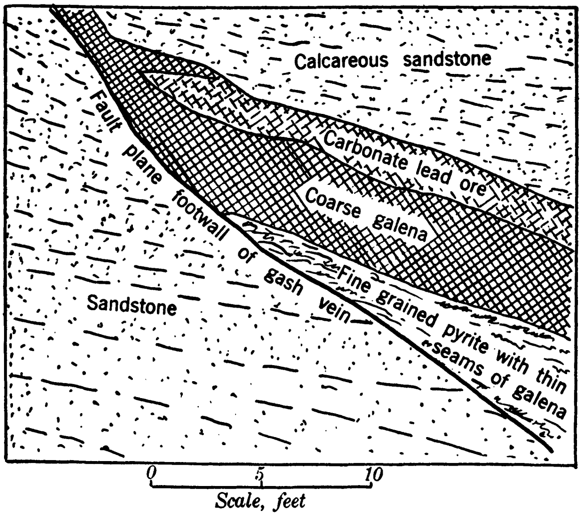

FIGURE 9.—Bedded lead ore cut by "gash" fault, Silver King mine, Park City district, Utah (after U.S. Geol. Survey).

stricted to certain favorable limestone beds or to contact horizons between limestone and quartzite. The bodies are roughly lenticular, although great irregularity of the walls is characteristic, and offshoots may extend hundreds of feet away from the parent ore bodies. Figure 9 shows bedded lead ore cut by a gash fault in the Silver King mine. Some bedded deposits at the Silver King properties have been followed on the strike for more than 10,000 feet without notable change in size or grade.

The lode deposits at this mine lie along the northeast-southwest main fracture zone rather than in minor fissures; in this respect they show marked contrast to the Coeur d'Alene veins. The lodes evidently occupy channels through which the solutions that gave rise to the bedded ores were introduced.

---

[19] Hewitt, E. A., Mining Methods and Costs at the Park Utah Mine, Park City, Utah: Inf. Circ. 6290, Bureau of Mines, 1930, p. 2.

[20] Dailey, M. J., Mining Methods and Costs of the Silver King Coalition Mines Co., Park City, Utah: Inf. Circ. 6371, Bureau of Mines, 1930, pp. 5–6.

The principal ore minerals of the Park City deposits consist of galena, pyrite, blende, tetrahedrite, and occasionally other copper minerals, in a gangue of quartz, chert, calcite, fluorite, rhodochrosite, and country rock. Silver and some gold are associated with the other metallic minerals. The upper portions of the lodes are oxidized, carrying characteristic secondary carbonates, oxides, and other alteration products of the sulphide minerals. At the Park Utah mine important production is obtained from the oxidized ores in the upper levels where direct-smelting siliceous silver ores are mined. The lower levels yield lead-zinc-silver sulphide ore of milling grade. Along fracture zones oxidation sometimes proceeds to considerable depth, being noticeable in places at 1,200 feet.

The mineralization of the district brought about alteration of the immediate walls of the deposits. Fault movement has produced more or less gouge and has resulted in a good deal of fracturing and brecciation of the wall rocks. These conditions combine to produce heavy ground characteristics that greatly affect mining operations.

According to Lindgren,[21] the grade of concentrating ore at Park City is about as follows: Lead, 6 to 8 percent; zinc, 6 to 8 percent; iron, 6 to 10 percent; and silver, 9 ounces per ton.

Gold and copper occur in minor amounts, although with depth the lodes carry proportionately more copper and zinc. The ore at lower levels in the lode deposits is of somewhat lower grade, but as the lodes are wider at the upper levels the total metallic content per vertical foot is probably undiminished.

Geological conditions in the Park City mines are responsible for heavy flows of water in the workings and render the mining problem more difficult than it would otherwise be.

TINTIC DISTRICT, UTAH

The Tintic district,[22] Juab and Utah Counties, Utah, about 70 miles south of Salt Lake City, is in the East Tintic Mountains, which attain elevations of more than 8,000 feet.

The geology of the district and of the Tintic Standard mine has been well summarized by Wade,[23] who states:

> From early Cambrian throughout the entire Paleozoic and into the Mesozoic time, there was a long period of continuous deposition of sedimentary rocks. These deposits finally reached a great thickness, probably 25,000 or 30,000 feet, consisting of quartzite, shales, and limestones.
>
> Following this period of deposition and at about the end of Jurassic time the entire section was uplifted and thrown into a series of folds and faults from compressive forces. From this great movement there resulted in Tintic a huge synclinal fold, the axis of which extends north and south, dipping slightly to the north. In the west limb of this syncline the beds stand almost vertically and in places are slightly overturned until they have a steep westerly dip; in the easterly limb the beds are flatter, having a westerly dip of only 20° to 40°.
>
> The forces that caused this syncline also resulted in much faulting. Many of these faults were along the bedding planes with north-south strikes, and others were transverse to the bedding. The vertical throw was considerable,

---

[21] Lindgren, Waldemar, Mineral Deposits: McGraw Hill Book Co., 3d ed., 1928, pp. 661–663.

[22] Butler, B. S., Loughlin, G. F., Heikes, V. C., and others, The Ore Deposits of Utah: U.S. Geol. Survey Prof. Paper 111, 1920, pp. 396–418.

[23] Wade, J. W., Mining Methods and Costs at Tintic Standard Mine, Tintic District, Utah: Inf. Circ. 6360, Bureau of Mines, 1930, pp. 2–5.

in some instances reaching as much as 1,500 feet. This folding and faulting fractured the beddings and created loose open channels which later were readily accessible to mineral-bearing solutions.

During the period of folding and faulting, and for a long time afterwards, erosion forces were active; and a great deal of the sedimentation was carried away, leaving the surface topography of the Tintic district comparatively rugged, with several high peaks and ridges, such as Eureka Peak and Pinion Peak. After this period of erosion there remained exposed some 6,000 feet of Cambrian quartzite and shales and about 7,000 feet of Paleozoic limestones.

The next and most important period influencing the ore deposits of this district was one of great volcanic activity, which began about early Tertiary time and affected the whole intermountain region. During this period the Tintic district was covered with many successive flows of lava, consisting mostly of rhyolite porphyry with some andesite and latites. The crowning event of this period was the intruding into the south end of the district near Diamond of a huge monzonite stock, which was forced up through the sedimentaries. Along with the great intrusion, or later but from the same source, came the mineral-bearing solutions that flowed out readily into the loose open channels already created in the soluble limestones, thereby making the replacement ore bodies from which many millions of dollars worth of metals have been produced.

The Tintic Standard mine is located on the east limb of the great Tintic syncline mentioned above. It comes very low in the general geological pile of Tintic, with the exception of surface flows being entirely in the Cambrian quartzites, limestones, and shales. Some 5,000 feet of this Cambrian quartzite underlies the shale and limestone series which produce the ore bodies of this mine.

During the general seismic activity following the period of deposition of the sedimentaries three major premineral normal faults were formed in the Tintic Standard property. The first, called the South Fault, strikes about N. 80° E. and dips northerly about 45°, the downthrow block being on the north. The second, called the North Fault, strikes northwesterly and southeasterly and dips southwesterly about 45°, the downthrow block being on the south side. Thus, these two faults, dipping toward each other, form a V-shaped trough narrow on the east end and wider to the west with the downthrow block between them. A third, called the East Fault, later in occurrence than the first two, striking about N. 30° E. and dipping 70° westerly with the downthrow block on the west side, cuts this trough near its east or narrow end. (See fig. 10, *A*.) On the upthrow side of each of these faults, or surrounding the trough, the top of the normal quartzite bedding is at about the 600 level of the mine; while on the downthrow sides, or within the trough, it is at about the 1,400 level, giving for all three faults a vertical throw of approximately 800 feet. The sedimentary beddings within the trough normally strike northerly and southerly and dip easterly 20° to 40°.

There then existed, prior to the deposition of mineral, a trough bounded on two sides and one end by major faults, with faulted faces of quartzite from the 600 level to the 1,400 level and with a quartzite bottom made up of normal beds dipping easterly so that the trough is deep and narrow at the east end and wider and more shallow to the west.

The limestones and shales in the trough and overlying the quartzite, which are called the Ophir series, have contained practically all of the Tintic Standard ore bodies found to date. This series is composed of alternate lime and shale beds as shown, capped by a thick shale bed which seems to have had some damming effect on the ascending mineral-bearing solutions.

Extending across the deep, narrow east end of the trough and striking N. 30° to 50° E. are several vertical fissure zones through which mineral-bearing solutions came from some unknown source at depth and, upon reaching the much broken and shattered soluble limestone beds, found easy access and many points of attack for replacement. The line of least resistance for these ascending solutions was along the major faults, so that the large replacement ore bodies are found in the Ophir limes adjacent to faulted faces of the quartzite.

The final geological event that influenced the molding of these ore bodies of the Tintic Standard mine was the occurrence of a fourth system of normal faulting, which is mostly postmineral. This system strikes about N. 20° W. and dips steeply to the southwest, forming a succession of step faults which have cut the ore bodies and raised the northeast segments successively higher as shown in figure 10, *B*, an ideal section through the ore zones at right angles to the postmineral fault system last mentioned.

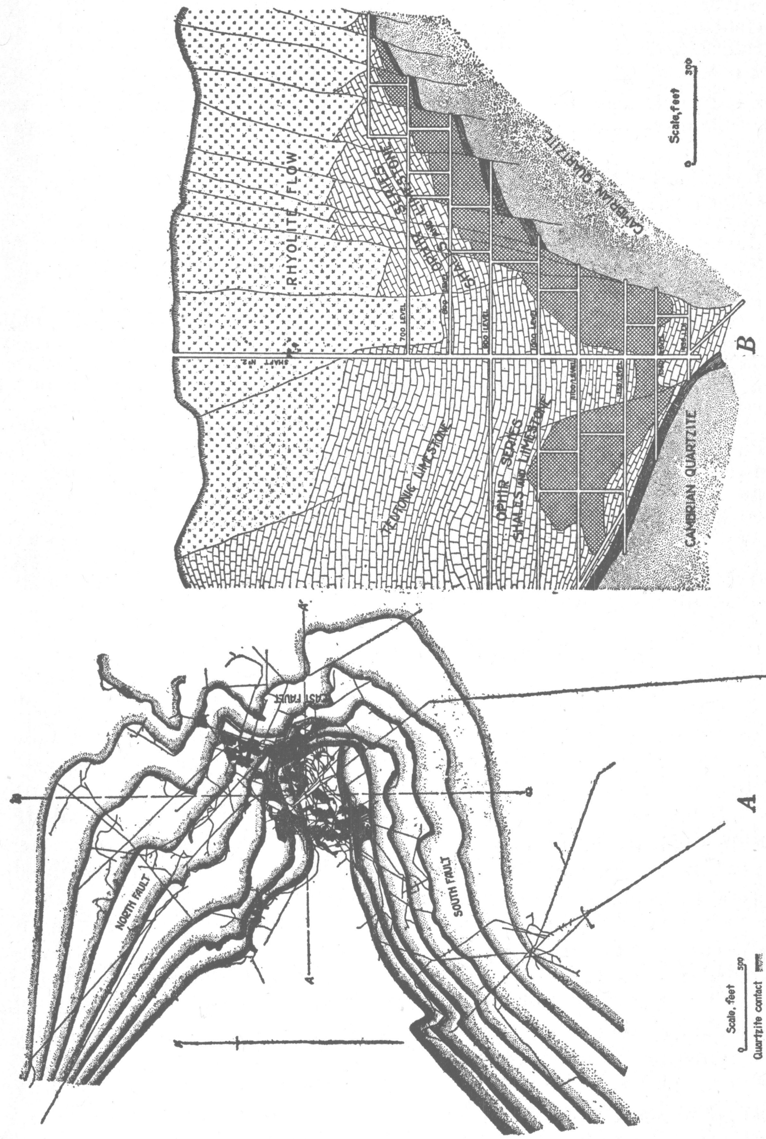

FIGURE 10.—*A*, Composite of Tintic Standard mine, Tintic district, Utah; *B*, ideal section on line N. 45° E. through no. 2 shaft, Tintic Standard mine.

As stated before, the major ore bodies of the Tintic Standard mine are found in the Ophir limestone on or near its contact with faulted faces of Cambrian quartzite, a firm, hard, massive rock. Along these fault faces, both premineral and postmineral faulting has been extensive.

The principal revenue of the mine is from lead-silver ores, with a quartz-barite gangue. The average silica content of clean, well-sorted lead ore is

49 percent. The following analysis is typical of the average of lead-ore shipments since the opening of the mine:

Lead	percent	25.04
Copper	do	.31
Silver	ounces	30.29
Gold	do	.039
Silica	percent	49.03
Iron	do	7.18

Occurring with lead ore in large masses and representing an earlier phase of mineralization than the lead is a siliceous ore of the following average content:

Lead	percent	4.85
Copper	do	.36
Silver	ounces	17.24
Gold	do	.037
Silica	percent	66.96
Iron	do	11.50

Both of these ores are completely oxidized above the 1,000 level; while below that level the sulphide content of the ore gradually increases, until on the 1,450 level the ores are practically all sulphide. Extensive postmineral faulting has brecciated a considerable part of this ore into fine sandy material, intermixed with masses of heavy lead ore. The hanging-wall rocks are the shales and limestones. The hanging-wall shales have been altered by mineralizing solutions to a soft, gray, clayey material. The limestone areas over the ore bodies are altered by the introduction of silica and magnesium silicates. These altered limestone areas are brecciated and have the appearance of the collapsed breccia filling in limestone caves.

The individual ore bodies, when adjacent to the quartzite fault faces, have steep dips ranging from 45° to vertical; when distant from the quartzite fault faces, they are tabular replacements of flat-lying limestone beds. The tabular commercial ore bodies range in height from 6 to 200 feet.

BUTTE DISTRICT, MONTANA

The Butte district, Silver Bow County, Mont., was originally worked for placer gold. Rich silver lodes were later exploited, but with the advent of matte smelting copper became important and is the metal that has made Butte famous as one of the greatest mining camps of all time. Copper has accounted for most of Butte's enormous output, valued at almost $2,000,000,000, with silver next in importance. Much zinc has been produced, however, from the lower parts of the veins originally mined for silver.

The veins of the Butte district are steep-dipping fissures which fall into three general groups of intersecting systems, one east-west, one northwest-southeast (later), and one northeast-southwest in trend. The first two include important ore-bearing veins, while the last is usually barren of values except for "drag" ore from other veins which the "fault veins" of this system intersect.

The country rock is a quartz-monzonite variant of the Boulder batholith, cut by granite porphyry and aplite dikes.

The metals show well-developed zonal distribution. Copper, with some silver and a little gold, is found in the center of the district. Bordering the central zone is a belt of more complex ore in which the content of silver increases and sphalerite is found, whereas silver and silver-zinc veins virtually free from copper occur on the north, west, and south periphery of the district. The silver veins are often capped by bold siliceous outcrops honeycombed and stained with manganese and generally are quite thoroughly leached. Below the leached gossan was usually a zone of silver enrichment, while below water level the zinc content increases until at depth mines originally worked for silver have become primarily producers of zinc. The principal minerals of these veins include quartz, pyrite, rhodonite, sphalerite, galena, rhodochrosite, and argentite. In contrast to the copper ores of the district, which are formed largely by replacement of country rock in and adjacent to strong fissures and contain little quartz, the silver-zinc ore bodies are essentially vein fillings, although replacement also took place. The veins are sometimes distinctly banded, and large amounts of quartz are present.

The veins of the Butte district have been intricately faulted and offset in at least two directions. They are consequently characterized by heavy gouge on the walls and in the bands within the veins, and much sericite and other greasy minerals have formed in the altered granite. These conditions make support of mine workings difficult. Faulted ore has always been a problem in the district; the faulting has also been responsible for much expensive apex litigation.

Much has been written on the geology of the Butte district,[24] and the subject is too involved for further discussion here. McGilvra and Healy [25] state as follows regarding the geology of the Black Rock mine, which has been one of the principal zinc-silver producers:

The Black Rock mine is on the eastern extremity of the Rainbow vein, which has a general east and west strike and an average dip of 80° to the south. The width of the vein ranges from a few feet to 120 feet. Stoping widths range from 1 to 12 sets, or 70 feet. The bulk of the production has been derived from this vein. Other veins occur, some of them of commercial importance. These veins are invariably narrow, averaging about 6 feet in width.

The country rock is granite, which has been intensely altered in proximity to the vein. The principal ore minerals are sphalerite and galena. The gangue consists mainly of altered granite, quartz, and pyrite. The vein structure presents a banded arrangement of ore and waste parallel to the walls.

The ore ranges from soft crumbly sphalerite to hard flinty material composed of quartz and sphalerite.

Both walls of the vein are usually talc seams, which cause considerable difficulty in mining operations. When a portion of the vein is undercut the block of ore above tends to slide downward on the slippery talc walls. Within the vein there are slips at right angles to the walls. When being stoped the ore often breaks off to these slips.

The granite country rock adjacent to the vein is altered, in many places to such an extent that it may be crumbled in the hand. The degree of alteration of the granite is more pronounced near the wider portions of the vein and becomes less intense as the distance from the vein increases.

In general, the ore and enclosing rocks are not strong enough to support themselves even over small spans. Artificial support is necessary as soon as possible after an opening is made.

---

[24] Weed, Walter H., Geology and Ore Deposits of the Butte District, Montana: U.S. Geol. Survey Prof. Paper 74, 1912, 262 pp.
Sales, Reno H., Ore Deposits of Butte, Mont.: Trans. Am. Inst. Min. Eng., vol. 46, 1914, pp. 3–106.
Lindgren, W., Mineral Deposits: McGraw-Hill Book Co., New York, 3d ed., 1928, pp. 696–700.

[25] McGilvra, D. B., and Healy, A. J., Methods of Mining at the Black Rock Mine, Butte & Superior Mining Co., Butte District, Mont.: Inf. Circ. 6370, Bureau of Mines, 1930, p. 2.

BARKER DISTRICT, MONTANA

The Unorganized Barker mining district, in the Little Belt Mountains, Judith Basin County, Mont., is 66 miles southeast of Great Falls. The elevation of the shaft collar at the Block P mine, the principal property, is 5,985 feet. According to Vanderburg,[26] the most important ore bodies of the district occur in a fissure vein in a syenite stock or chimney which has been intruded by dikes of rhyolite. In plan the vein is crescent-shaped and has been exposed by underground workings for 4,000 feet laterally and 1,200 feet vertically below the shaft collar. The vein ranges in dip from 65° to 88° and has well-defined walls.

The ore exhibits a banded character and occurs in lenticular bodies within the vein. Ore bodies are usually 1 to 4 feet wide and about 400 feet long, although some have been stoped for 750 feet. The mining method is influenced by the narrowness of the vein and the banded structure of the ore; the latter occurs sometimes as a single band within the vein and sometimes in several bands alternating with waste or lean material. The vein walls are generally quite firm, although in places they are somewhat blocky due to slips and joints.

The ore minerals [27] consist of galena, marmatite (iron-bearing blende), sphalerite, and pyrite, considerable silver being associated with the sulphides of lead and zinc. Small amounts of chalcopyrite and cupriferous pyrite are present. The gangue, composed in the main of altered syenite and rhyolite, carries also some calcite, quartz, barite, rhodochrosite, and marcasite. During 1929 the average assay of the ore milled was as follows: Lead, 6.21 percent; zinc, 5.01 percent; and silver, 0.09 ounce per ton.

Oxidation of the ore minerals has not taken place to any extent at the horizons which have been mined recently. The rich ores mined near the surface soon after the discovery of the deposits in the late fifties, which were valuable for their lead and silver, were doubtless more or less oxidized and enriched. With increasing depth zinc has become relatively more abundant.

PECOS DISTRICT, NEW MEXICO

The Pecos or Cooper district, San Miguel County, N. Mex., lies about 8,000 feet above sea level and is 17 miles east-northeast of Santa Fe. The only operating mine in the district is the Pecos property. The geology has been summarized briefly by Matson and Hoag [28] who state:

> The Pecos mine is in the heart of the main mass of the pre-Cambrian rocks of New Mexico. In the neighborhood of the mine, however, the granite is exposed only in the canyons of the Pecos River and its tributary creeks; elsewhere it is covered by extensive areas of the Magdalena limestone formation of Carboniferous age. The pre-Cambrian granite has been intruded by a dark, fine-grained diorite. An extensive northeast-southwest shear zone has

---

[26] Vanderburg, Wm. O., Mining Methods at the Block P Mine of the St. Joseph Lead Co., Hughesville, Mont.: Inf. Circ. 6416, Bureau of Mines, 1931, pp. 2–3.

[27] Vanderburg, Wm. O., Milling Methods at the Hughesville Concentrator of the St. Joseph Lead Co., Hughesville, Mont.: Inf. Circ. 6447, Bureau of Mines, 1931, p. 2.

[28] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Inf. Circ. 6368, Bureau of Mines, 1930, pp. 2–3.

altered the igneous rocks into parallel bands of schist. This zone, which is from a few feet to several hundred feet wide, contains rocks of all degrees of metamorphism and varying from acid to basic. The ore bodies are found in the shear zone, replacing the schist. In only a few places have they been found to reach up to the bottom of the limestone and in no case to extend into it.

The mineralization consists of a mixture of sphalerite, galena, chalcopyrite, and pyrite, carrying appreciable values in gold and silver, associated with the products of metamorphism and alteration, such as talc, hornblende, mica, and

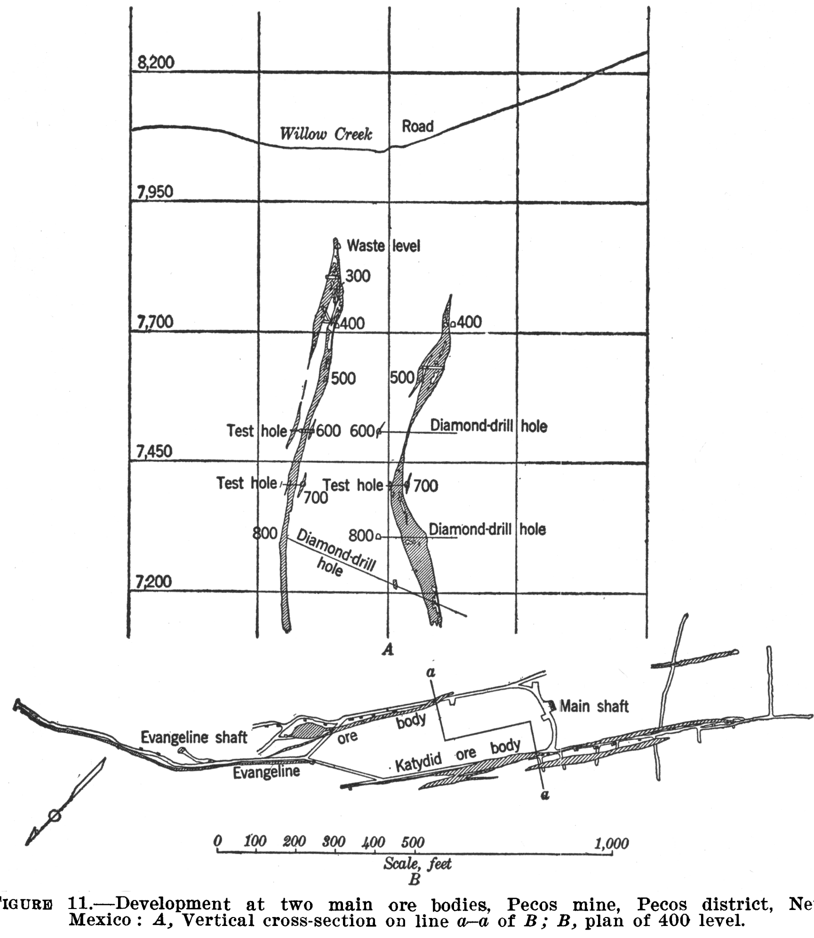

FIGURE 11.—Development at two main ore bodies, Pecos mine, Pecos district, New Mexico: *A*, Vertical cross-section on line *a–a* of *B*; *B*, plan of 400 level.

chlorite schist. Near the surface the ore bodies have been oxidized but to only shallow depths. No secondary enrichment is observable. In the Evangeline ore body lead appears to increase in proportion to zinc as depth is gained.

From January, 1927, to January, 1930, the mine has produced 584,158 tons of ore averaging 16.06 percent zinc, 3.73 percent lead, 1.02 percent copper, 3.39 ounces of silver, and 0.109 ounce of gold per ton.

The ore bodies have been developed for a length of 2,000 feet and from the surface to the 1,200 level. They consist of irregular, disconnected, often overlapping lenses of sulphides in the shear zones. The lenses often pinch out abruptly, either horizontally or vertically. Figure 11 illustrates the development of the two main ore bodies, the Katydid and the Evangeline, and some minor parallel occurrences.

The mineralization sometimes replaces all of the schist, in which case the ore body may have diorite foot and hanging walls. In other places only certain bands of the schist are replaced, and both walls may be of schist. Some movement apparently accompanied or followed deposition of the ores, but there has been no important displacement of the ore bodies.

The ore and walls are both generally very loose and require close timbering. The ground in many places runs freely, and serious caves are likely to follow. If any initial movement can be prevented, however, support is not difficult, as the ground is neither heavy nor swelling when dry. The presence of water makes the ground very difficult to support, because of its talc content. For this reason it is important to provide drainage by developing levels below the stoping areas.

Some interesting notes on the geology and early developments of the district were given by Lindgren [29] in 1910. A more detailed discussion of the geology of the Pecos mine has been published recently by Stott.[30]

SANTA RITA-HANOVER REGION, NEW MEXICO

The important mineralized area of northeastern Grant County, N. Mex., includes several organized mining districts from which gold, silver, copper, lead, zinc, and iron, as well as some vanadium, tungsten, molybdenum, and other metals have been produced.

According to Graton [31] the principal rocks of the region consist of heavy flows of Tertiary rhyolite, andesite, and basalt underlain by Palezoic limestones and Cretaceous shales and sandstones. The strata have been considerably faulted and have undergone some folding, usually local. Later granite porphyry and quartz monzonite porphyry have invaded the country rocks and doubtless have been the source of most of the ore.

Several classes of ore deposits are represented, including fissure veins, limestone replacements, disseminated copper ores, and contact metamorphic types, but the replacement variety has furnished most of the lead and zinc. Some of the fissure veins, however, have been mined for these metals, and some zinc ores occur in deposits of contact metamorphic or related origin. It should be noted that the replacement and contact types often grade into one another, these in turn passing into vein types, so that frequently no definite differentiation is possible.

Many deposits were worked originally for the precious metals contained in the zones of oxidation and enrichment. The base metals, especially zinc, have become more prominent with depth. Conspicuous gossans have marked the outcrops of a good many of the sulphide bodies.

Mineralogically, the ores exhibit a wide degree of variation. Thus, fissure veins in porphyry may contain pyrite, sphalerite, and chalcopyrite in varying proportions, with some galena, specularite, gold and silver minerals, and molybdenite, all in a gangue of quartz and altered country rock. The more important zinc bodies (those of replacement type occurring in crystalline limestone near

[29] Lindgren, Waldemar, Graton, L. C., and Gordon, Charles H., The Ore Deposits of New Mexico: U.S. Geol. Survey Prof. Paper 68, 1910, pp. 110–114.

[30] Stott, Charles E., Geology of the Pecos Mine: Eng. and Min. Jour., vol. 131, Mar. 23, 1931, pp. 270–275.

[31] Lindgren, W., Graton, L. C., and Gordon, C. H., The Ore Deposits of New Mexico: U.S. Geol. Survey Prof. Paper 68, 1910, pp. 295–348.

porphyry contacts) contain massive sphalerite with some pyrite, galena, and chalcopyrite, associated with pyroxene and other silicate minerals.

In a district characterized by so wide a range in geological conditions and types of deposits the grade of ore naturally varies considerably at different mines. At the Black Hawk concentrator near Hanover the ore treated has come chiefly from the Combination shaft but has been augmented by additional supplies from the Ground Hog, Lucky Bill, and other mines. For 2 years after the summer of 1928, according to Wright,[32] the average ore milled assayed about as follows: Silver, 2 ounces per ton; lead, 2.5 percent; copper, 0.5 percent; and zinc, 12 percent.

Clean ore will run considerably higher than the average shown, but contamination by limestone and iron pyrite has brought the average down. During the latter part of 1930, however, the grade of ore was about 6 percent lead and 15 percent zinc, with correspondingly higher silver and copper values, probably due largely to an increase in the proportion of the ore furnished by the Ground Hog mine.

Regarding the mineralogic character of the ore treated at this plant, Wright states further:

> The metallic minerals are galena, spalerite, chalcopyrite, and pyrite. The association of the silver with the various metallic sulphides is not known. Pyrite is present in amount about equal to the sphalerite. The lead, zinc, and copper minerals are coarsely crystalline and do not require particularly fine grinding to effect a separation.
>
> The gangue consists of limestone, hedenbergite, and garnet. The ore is not hard to crush; but part of the gangue is a tough hedenbergite which breaks along radial cleavage surfaces, and large thin pieces are common in the ball mill feed. The garnet is not present in sufficient amount to cause excessive wear on the crushers.

Although the majority of the zinc and zinc-lead ore bodies of this region occur as irregular deposits replacing limestone, an important exception is provided by the Ground Hog mine. In writing of operations at this property, Richard [33] describes the geology briefly as follows:

> The lead-zinc mines of the Central mining district are mostly in limestone. To the present time no other ore body similar to the Ground Hog has been found in the district. Before the extrusion of the Tertiary volcano the ground now containing the ore bodies was severely faulted, developing a series of parallel fractures having a general northeasterly strike and a dip of about 50° to the southeast. Subsequent to a part of this faulting came the intrusion of quartz diorite porphyry and granodiorite, the latter known locally as "bird's-eye porphyry", probably in the form of dikes which followed the fault zone. More faulting along old planes of weakness followed. Apparently, subsequent intrusions have consisted of siliceous and ore-bearing solutions.
>
> The result of these events has been to form an ore body which may be considered as filling a fault fissure. (See fig. 12.) It has foot and hanging walls consisting in many places of fault gouge several feet thick. In some places the vein is very siliceous, but in most parts it is massive sulphide. The ore presents a roughly banded structure. The ore minerals do not extend into the wall rocks to any appreciable extent.
>
> Postmineral faulting has not been found to be extensive, and the vein is fairly continuous without major displacements.

---

[32] Wright, Ira L., Milling Methods and Costs at the Black Hawk Concentrator, Hanover, N.Mex.: Inf. Circ. 6359, Bureau of Mines, 1930, p. 2.

[33] Richard, F. W., Mining Methods and Costs at the Ground Hog Unit, Asarco Mining Co., Vanadium, N.Mex.: Inf. Circ. 6377, Bureau of Mines, 1930, pp. 2–3.

Below the 400-foot level the mineralization is practically all primary, consisting of galena, chalcopyrite, and sphalerite, with quartz and pyrite as gangue minerals.

The main ore shoots occur as irregularly shaped, connected lenses along the contact between the diorite porphyry and a granodiorite or "bird's-eye porphyry" dike. The ore has been followed for approximately 600 feet along the strike on the 500-foot level and extends from that level to the 300, where

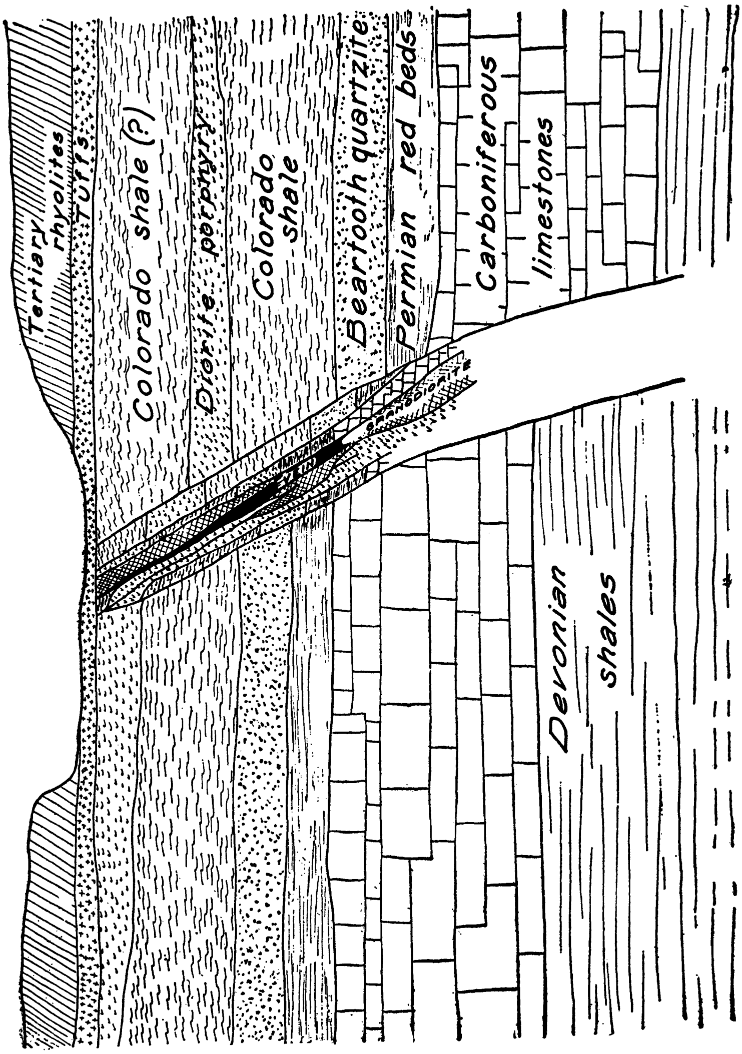

FIGURE 12.—Theoretical geological section, Ground Hog mine, Santa Rita-Hanover region, New Mexico.

it apexes. Its width varies from 3 feet to 25 feet normal to the dip. The average dip is about 50°, being steeper than that at the north end of the claim and flattening toward the south end to such an extent that broken ore will not run freely by gravity.

Postmineral faulting, though of slight displacement, has fractured the ore and wall rocks so that they make treacherous mining ground, which must be studied carefully and watched closely to prevent disastrous cave-ins. The ore is a heavy sulphide of little natural strength and in general requires more support than do the walls. The dike forming the hanging wall in most places is from 15 to 90 feet thick. It will stand if not opened over too great distances; if too large an area is unsupported it breaks into large slabs or blocks. The footwall consists normally of a thick gouge, which may cause trouble by sloughing away, thus weakening the back and increasing the amount of waste to be handled. Frequently, however, the vein material next to the footwall consists of a low-grade siliceous ore which can be left in place and which forms a strong and satisfactory wall.

Perhaps the worst condition occurs where the vein has formed in the diorite slightly removed from the dike. The thin shell of diorite then forming the hanging wall, being badly fractured and crushed, is very difficult to support. It has a tendency to "ravel" away from the edge of a block or stull headboard and requires close and prompt filling.

YELLOW PINE DISTRICT, NEVADA

The Yellow Pine (Goodsprings) district is situated in southwestern Clark County in southeastern Nevada. Elevations near the mines range from less than 3,000 feet in the southwestern part of the district to 8,504 feet at Potosi Mountain (Olcott Peak).

The geology and ore deposits of the area have recently been studied in detail by Hewett,[34] from whose report the following brief description has been abstracted.

Sedimentary rocks having a total thickness of 13,000 feet are exposed in the district, more than 7,600 feet being carbonate rocks. Middle Cambrian formations, resting on pre-Cambrian gneissic granite (which does not outcrop), are overlain by 8,500 feet of Paleozoic sediments, 7,000 feet being limestones and dolomites. Four thousand feet of lower Mesozoic rocks, chiefly conglomerate, sandstones, and shales, with 600 feet of limestone as a basal member, overlie the Paleozoics. Tertiary flows and tuffs and Pleistocene gravels were the last stratified rocks to be formed. Sills and large masses of granite porphyry intruded the sediments near large thrust faults; smaller dikes of porphyry and lamprophyre are found, chiefly near major fault zones.

The rocks have been folded and faulted extensively, several periods of displacement being clearly discernible. Large thrust faults nearly parallel to the bedding of the sediments had as their corollary development of fracture zones and zones of brecciation. Minor thrusts accompanied major movements and followed them after an intervening period during which the granite porphyry sills and dikes were intruded. Ore was deposited during the closing stages of this period of igneous intrusion and in the interval immediately afterward. Some normal faults containing ore indicate that this type of movement was initiated before the close of the period of mineralization. Other normal faulting occurred still later and caused some displacement of the ore bodies.

Large masses and thick layers of pure limestone were dolomitized after most of the thrust faulting but before ore deposition. Most of the ore is found in brecciated dolomitized limestone, and Hewett concludes that where this form of alteration has not been developed the probability of finding ore bodies is poor.

The district has produced ores of gold, copper, zinc, lead, and other metals, with zinc predominating. Many deposits are grouped

---

[34] Hewett, D. F., Geology and Ore Deposits of the Goodsprings Quadrangle, Nevada: U.S. Geol. Survey, Prof. Paper 162, 1931, 172 pp.

about intrusive masses of granite porphyry, but others are found several miles from any known outcrops of such igneous rocks. Most zinc and lead deposits occur in lower Mississippian beds and, except

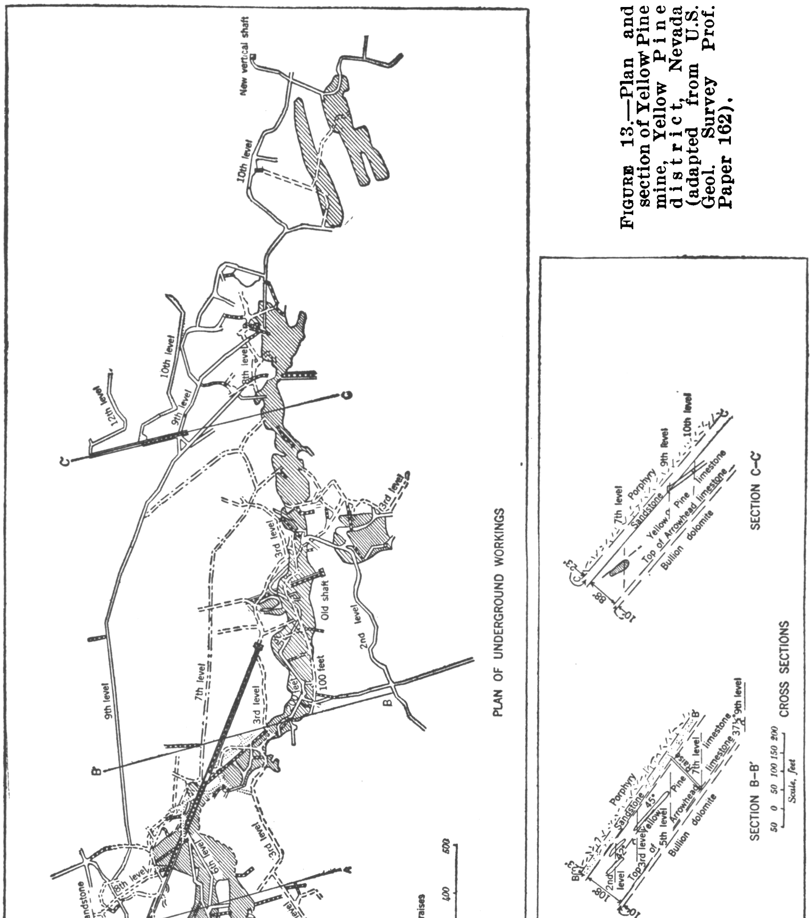

FIGURE 13.—Plan and section of Yellow Pine mine, Yellow Pine district, Nevada (adapted from U.S. Geol. Survey Prof. Paper 162).

for the Yellow Pine and Prairie Flower ore bodies (which underlie a porphyry sill), are not found near outcrops of intrusive rocks Figure 13 shows a plan and cross-sections of the Yellow Pine mine.

The bodies of zinc and lead ores are generally characterized by an association of the minerals of both metals, although a few deposits carrying zinc are devoid of lead and a few lead ores contain no zinc. The primary mineralization doubtless consisted essentially of zinc and lead sulphides; but weathering and alteration of the ore minerals have proceeded to so great a degree that sphalerite is rarely found, and galena, although common enough, is less abundant than its secondary derivatives. The chief zinc mineral is earthy hydrozincite (the hydrated carbonate); smithsonite ($ZnCO_3$) and calamine (the hydrous zinc silicate) are less important but are wide-spread. Lead occurs chiefly as the carbonate (cerussite); but the sulphate (anglesite) and small amounts of mixed vanadates of lead, zinc, and copper are widely distributed in the district.

Most of the important ore bodies are of the flat tubular or bedded type, being localized in the breccia zones that have been produced by thrust faulting parallel, or nearly parallel, to the bedding. The ore occurs filling fractures and voids in the breccia and, to a smaller extent, replacing the dolomitized limestone. Apparently the mineralizing solutions were introduced through the steeper crosscutting fractures, spreading thence laterally along the breccia zones. A few ore bodies occur in and adjacent to the steeper fractures which cut the beds obliquely, and two large deposits are contained in conical or dome-shaped zones of brecciation which have been caused by the intersection of steep fractures with flat zones of faulting.

There are few deposits in the major thrust-fault fractures. An important feature of the zinc-bearing deposits is the presence of large concentrations of zinc carbonate and silicate ore below or in the lower part of the areas of primary mineralization. This condition has resulted from downward movement of solutions containing zinc derived from the primary minerals above and precipitation of this zinc at greater depths—a phenomenon characteristic of many zinc deposits throughout the world. This feature has obvious importance to exploration in the district and means that search for zinc ore should be directed downward from any discovery of primary ores in underground openings, or that discovery of ore containing chiefly lead (which does not migrate to any extent compared with zinc) should warrant deeper work in the hope of opening up secondary zinc bodies. The mines are very dry, and support of underground openings does not constitute a serious problem.

EDWARDS DISTRICT, NEW YORK

The Edwards district [35] is in St. Lawrence County, northern New York, about 15 miles east and southeast of Gouverneur. The region is characterized by low relief and gently rolling topography.

The zinc ore bodies of the district occupy a belt of pre-Cambrian crystalline dolomites, quartzites, schists, and gneisses of Grenville age, 1½ miles or less in width, extending from Edwards on the northeast to Sylvia Lake, 8 miles southwest. These rocks were intruded by granitic magmas and pegmatites and were consequently

---

[35] Cushing, H. P., and Newland, D. H., Geology of the Gouverneur Quadrangle: New York State Museum Bull. 259, 1925.
Knaebel, J. B., Mining Practice at the Edwards Mine of the St. Joseph Lead Co., St. Lawrence County, N.Y.: Inf. Circ. 6586, Bureau of Mines, 1932, pp. 2–5.

metamorphosed to a considerable degree; the schists and quartzites assumed a gneissic character, while the dolomitic rocks were recrystallized, and tremolite and diopside were developed therein. Dynamic action, which in part at least preceded the period of igneous intrusion, greatly deformed these old rocks and resulted in a rather complex folded structure. The Grenville sediments exhibit a general dip of about 45° northwest, but intense local folding has contorted these crystalline dolomites and associated rocks into a hook-shaped structure plunging steeply to the northwest. Smaller secondary folds, roughly parallel to the main overturn, opened up the limestone and rendered it permeable to the movement of solutions up the dip. The ore-bearing fluids altered much of the tremolite or diopside to serpentine and some talc, and sulphide ore was deposited by high-temperature replacement of the silicated dolomite.

As ore deposition has been controlled by the plunging minor folds, the ore bodies are lenticular in horizontal cross-section. They range from 5 to 25 feet in thickness and 100 to 200 feet in length along the strike and usually are persistent down the dip for 2,500 feet.

The ore is a mixture of marmatite (iron-bearing blende) and pyrite in proportions of about 2 to 1, respectively; pyrrhotite sometimes takes the place of pyrite. Other sulphides, such as galena, molybdenite, and chalcopyrite, are rare, except at Balmat, near Sylvia Lake, where the ore carries 1½ percent of lead. The gangue is composed essentially of dolomite, serpentine, and some tremolite, diopside, talc, and other silicates. Hematite and other iron oxides are found in the upper portions of some of the ore bodies. The ore is commonly rather massive, but inclusions and bands of waste are frequent, and disseminated ore occurs between masses and at the sides and ends of some of the deposits. Figure 14, *A*, depicts a typical breast of ore. Ore in place averages about 17 percent zinc.

Walls are not usually sharply defined; the ore generally grades off rather rapidly into the enclosing rocks, although slips or joints sometimes form walls for short distances. Parallel streaks and bunches of ore may branch off into the walls and may also occur blindly, in which event they must be located by test holes.

The ore bodies lie wholly within the dolomite, usually near the overlying gneiss. They frequently do not outcrop; at the Edwards mine 5 of the 7 known ore bodies apex beneath the surface. They all dip with the formations in which they occur (at 40° to 45°) with local variations ranging from virtually flat to vertical. They likewise vary locally in thickness. (See fig. 14, *B*.)

Post-ore faulting has not seriously displaced the ore; but joints and slips have been more or less developed, especially in the lower levels. Some of these structures roughly parallel the ore, and they induce sloughing off of slabs from the hanging wall and consequent dilution in mining. In a few places these slips are so strongly developed that stoping must be modified to combat the hazard they present.

Both ore and wall rocks are firm and moderately hard, and where the hanging wall has not been badly fractured and jointed the ground will stand unsupported indefinitely, except for more or less sloughing of slabs, as just noted.

Mines have been developed near Edwards and more recently at Balmat, near Sylvia Lake. Indications of mineralization are known at various points along the belt of Grenville dolomites, and some

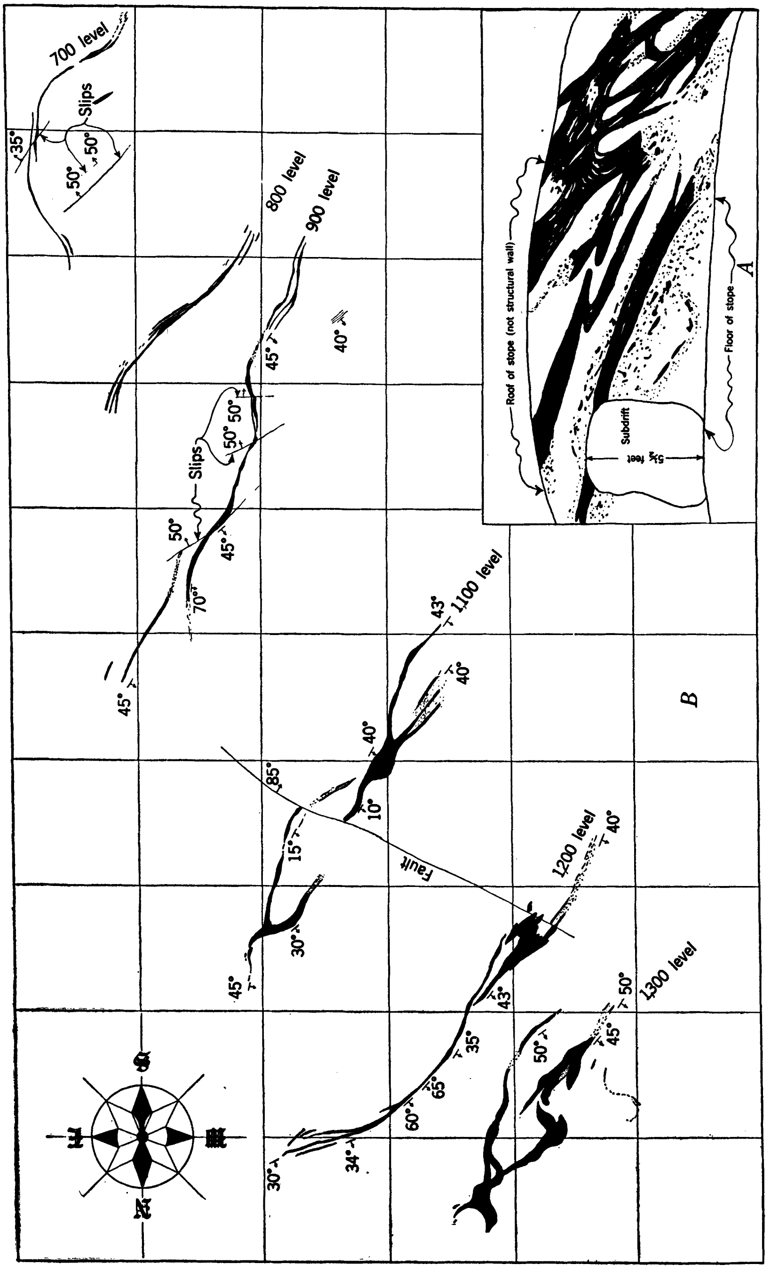

FIGURE 14.—*A*, Typical breast of ore, D-11 stope above 1,300-foot level, Edwards mine, Edwards district, New York; *B*, plan and sections of no. 3 ore body, Edwards mine, 700- to 1,300-foot levels. (Scale: Coordinates at 100-foot intervals.)

prospecting and diamond drilling have been done on the showings, but up to the time of writing the only minable ore bodies that have been opened lie at or near the extreme ends of the belt. Talc has been mined for years near Balmat and between the zinc mines and Gouverneur. Considerable amounts of magnetite and pyrite have also been produced from St. Lawrence County. Thus, the general region has been rather well mineralized, and in view of the fact that most of the zinc ore bodies so far discovered do not outcrop, other deposits may be found in the future.

FRANKLIN FURNACE DISTRICT, NEW JERSEY

The Franklin district, Sussex County, northern New Jersey, contains two large zinc-ore deposits. One, the Mine Hill deposit, is at the town of Franklin; the other, the Sterling Hill deposit, is 2 or 3 miles south-southwest at Ogdensburg.

The deposits occur in a crystalline limestone of pre-Cambrian age which forms a belt up to half a mile wide. Igneous rocks, now metamorphosed to gneisses, intruded the limestone along the western side of the belt and probably provided the source of the metallic mineralization. Cambrian limestones and quartzite overlie the gneiss and earlier limestone on both sides of the belt.

The ore deposits, which many geologists believe to be of contact-metamorphic or closely related origin, are hook-shaped in plan and in section and present the form of plunging troughs which pitch downward from the nose of the hooklike outcrop at angles which near the surface are about 25° or 30° and gradually flatten with depth to 10° or less. The sides or limbs of the troughs dip rather steeply, converging abruptly at the bottom of the deposits, as may be seen from the cross-sections. (See fig. 15.) Figure 15 also illustrates the general geology and position of the ore bodies with respect to their host formations.

At Sterling Hill the limestone within the trough is somewhat mineralized, containing some low-grade ore. The ore bodies and surrounding rocks are cut by numerous pegmatite dikes, which doubtless introduced mineralizers that were more or less responsible for the development of the great variety of oxide, silicate, and other mineral species for which the district is famous. Other veins are numerous; some contain the normal ore minerals, while others are of later origin and carry sulphides, carbonates, and silicates. The workable ore bodies are valuable for their zinc, manganese, and iron. These metals occur chiefly as oxides and silicates; and typical ore according to Lindgren,[36] consists of an aggregate of 50 percent franklinite ($(Fe, Mn, Zn)O(Fe, Mn)_2O_3$), 20 to 30 percent willemite (Zn_2SiO_4), 2 to 6 percent zincite (ZnO), and 3 to 11 percent calcite. Many rare and complex minerals containing zinc and manganese are noted in the deposits.

The ore bodies range from 10 to over 100 feet in thickness, the thicker portions being near the bottoms of the pitching troughs. The ore exhibits a more or less banded character and grades into the country rock without sharply defined structural walls. Depths of more than 1,100 feet have been attained in the workings.

---

[36] Lindgren, Waldemar, Mineral Deposits: McGraw-Hill Book Co., New York, 3d ed., 1928, pp. 830–833.

Lindgren [37] stated in 1928 that the mines yielded annually 600,000 tons of ore containing about 90,000 tons of zinc; an average tenor of about 15 percent zinc is thus indicated.

The Franklin district is one of very few in North America in which important zinc deposits have been formed by processes related to contact metamorphism, although geologists are not all agreed that

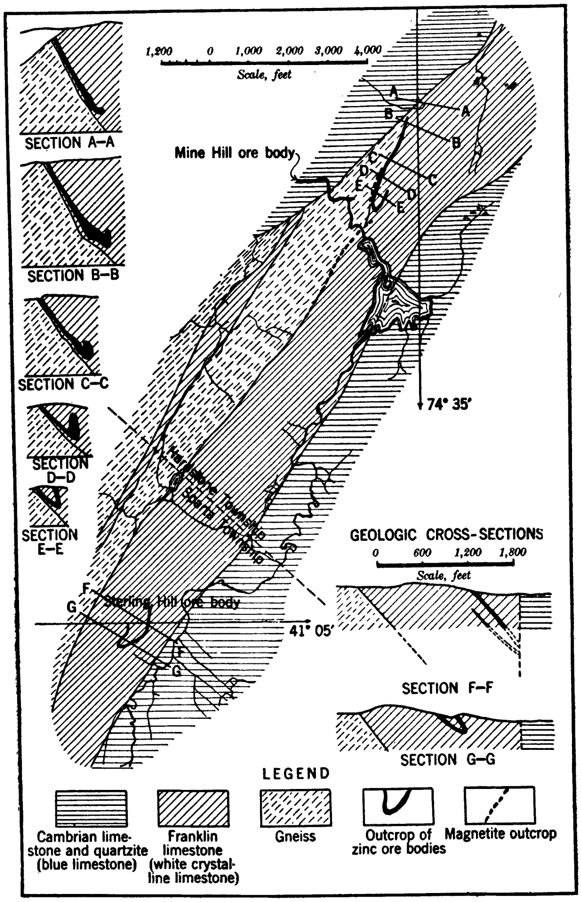

FIGURE 15.—Plan and sections of ore bodies, Franklin district, New Jersey (after Spencer).

it should be so regarded. Kemp [38] believed that the ores were deposited by igneous solutions; Wolff [39] regarded them as contemporaneous with the limestones; while Ries and Bowen [40] believed that the ores were replacement bedded deposits in the limestone, which were later metamorphosed to their present oxide silicate character by the

[37] Lindgren, W., Mineral Deposits: McGraw-Hill Book Co., New York, 3d ed., 1928, p. 831.
[38] Kemp, J. F., The Ore Deposits of Franklin Furnace and Ogdensburg, N.J.: Trans. New York Acad. Sci., vol. 13, 1894, p. 76.
[39] Wolff, J. E., Zinc and Manganese Deposits of Franklin Furnace, N.J.: U.S. Geol. Survey Bull. 213, 1903, pp. 214–217.
[40] Ries, H., and Bowen, Origin of the Zinc Ores of Sussex County, N.J.: Econ. Geol., vol. 17, 1922, p. 517.

influence of the intrusive syenite and later pegmatites. Spencer[41] classified them as of contact metamorphic origin, and this view is held by many others. Lindgren[42] ascribes their origin to pyrometasomatism not closely related to contacts. The history of the ores is complex; however, in their present form they owe much to the influence of nearby magmatic intrusions.

Another deposit of contact metamorphic type in the United States, at Magdalena, N. Mex.,[43] has yielded silver, lead, and zinc from the oxidized zone. Blende, with some galena and chalcopyrite, is found at depth, associated with high-temperature minerals such as magnetite, specularite, tremolite, epidote, and pyroxene. Still other deposits are due in part to contact metamorphism but are important chiefly for the ore contained in associated bodies formed by replacement or other processes.

---

[41] Spencer, A. C., and others, Franklin Furnace, N.J.: U.S. Geol. Survey, Folio 161. 1908, 27 pp.

[42] Lindgren, W., Mineral Deposits: McGraw-Hill Book Co., New York, 3d ed., 1928. pp. 827–833.

[43] Lindgren, W., Graton, L. C., and Gordon, C. H., The Ore Deposits of New Mexico: U.S. Geol. Survey Prof. Paper 68, 1910, pp. 213–285.

Part 2.—EXPLORATION, DEVELOPMENT, AND MINING

EXPLORATION

In this paper the term "exploration" is applied to the search for new deposits and extensions of known ore bodies. During the past 3 years economic conditions and the demand for lead and zinc have not encouraged the search for their ores in hitherto unproductive districts in the United States. Likewise, intensive search for new ore bodies in some producing districts has probably been retarded by the business recession. On the other hand, some operating companies well-fortified with cash have probably employed this period to increase their ore reserves by exploring for extensions of known deposits and for adjacent ore bodies.

Exploration may be conducted from the surface, from underground workings, or from both, depending upon the type, size, shape, and dip of the deposits, their depth below the surface, their position relative to the surface topography and to accessible underground workings, and the relative costs of surface and underground exploration.

In new areas preliminary exploration may be by surface trenches, test pits, or adit levels, if the deposits outcrop at the surface; the deposits are explored further at depth if preliminary results are encouraging.

Often, however, valuable deposits of ore do not outcrop, and there may be no surface indications thereof. In such instances the clue to the presence of deposits may be looked for under similar geological conditions to those of known deposits in the same region. Thus, in the Kansas-Oklahoma section of the Tri-State lead and zinc district there are no surface indications of ore where most of the largest productive mines are located. The mines were discovered by widening the field of exploration about the early discoveries of surface ores made where they were disclosed by erosion of overlying rocks in southwestern Missouri. The first ores mined were lead ores, whereas zinc ore now constitutes the bulk of the production of the district.

In the Southeastern Missouri lead district the first discoveries were made in crevices and pockets connecting with the surface. Most of these deposits were small individually, but they led to discovery of the vast disseminated deposits at greater depths, which have furnished the bulk of the ore from this district.

In both the Tri-State and the Southeastern Missouri district exploration has been principally by drilling from the surface. Churn drills have been employed for the most part in the former and diamond drills in the latter. In both districts the deposits are flat-lying, the horizontal dimensions usually being much greater than the vertical dimension. In the Tri-State district the brecciated, seamy nature of the ground and the presence of broken cherty material make diamond drilling difficult and expensive, carbon loss high, and core recovery poor; hence, the churn drill has been the principal exploration tool in the district. In southeastern Missouri, however, the ground is much more uniform in texture and structure, usually gives good core recovery, and is well suited to the diamond drill.

Tennessee zinc deposits furnish another example of bedded deposits at comparatively flat angles where both diamond and churn drilling have been used extensively for exploration.

The existence of other types of deposits, such as inclined tabular deposits of the fissure-vein and fault-zone types and the contact-metamorphic type, has generally been indicated by mineralized surface outcrops. Many such ore bodies have not outcropped, however, but have been discovered as a result of exploration from workings in nearby deposits which did outcrop. Thus, several ore bodies, some of which do not outcrop, may occur as a series of disconnected ore shoots along the strike of a fissure, fracture zone, or contact. Parallel ore shoots or blind lodes may occur in the walls of known ore bodies, being separated therefrom by barren rock. In other instances ore shoots branch off from the main ore shoots.

Under these conditions exploration may be conducted by sinking, tunneling, or drilling from the surface in geologically favorable areas, or by underground exploration. Underground exploration may be conducted by ordinary crosscutting, drifting, sinking, and raising operations, or by use of drilling equipment. In underground prospect-drilling, diamond drills, ordinary hammer drills, and special long-hole hammer drills are widely employed, and in some districts churn drills are used.

In all exploration it is important to understand the geology of the deposits; the favorable formations; the relations of the ore bodies to structural features such as faults, fracture zones, folds, or contacts; and the general habits of ore occurrence. In an undeveloped region little is known about these factors at the outset, and it is important to note carefully and record all geological information as work progresses for use in directing subsequent exploration to better advantage. For this purpose the assistance of an engineer or geologist having wide experience and knowledge of the habit of ore deposits is valuable.

In a productive region further exploration can usually be directed to best advantage after careful study of the geology of the developed deposits, and a geologist having detailed knowledge thereof obviously is best equipped by experience to direct the work.

Exploration methods employed in the United States are considered in greater detail in the following pages where examples are cited of practices employed at a number of individual mines.

EXAMPLES OF EXPLORATION METHODS

SOUTHEASTERN MISSOURI DISTRICT

As described briefly under "Geology", the ore deposits in the Southeastern Missouri lead district are flat-lying bedded deposits about 750 feet or less below the surface. The lead occurs in the form of galena disseminated in dolomitic limestone and shaly beds, in horizontal sheets along bedding planes, in vugs and cavities, as filling in the walls of joints and crevices, and as aggregates of cubes in channels and joints.

Exploration is conducted mostly by diamond drilling from the surface and underground by diamond drilling or test-hole drilling with light hammer drills or heavy-duty Leyner drills. Little underground exploration is performed by drifting and crosscutting.

Diamond drilling.—At no. 8 mine of the St. Louis Smelting & Refining Co.[1] 80 diamond-drill holes totaling 48,000 feet were drilled from the surface to explore a 40-acre tract before development was planned, shafts were sunk, and the property equipped for mining. The area was first tested with a few holes to determine whether any lead ore existed. These holes indicated ore at depth, and the balance of the 80 holes were spaced at irregular intervals to give as complete information as possible for delimiting the ore bodies.

The first few holes were cored all the way from the surface to the Lamotte sandstone which lies directly under the principal ore-bearing formation in the district, but after the approximate ore horizon had been established the upper few hundred feet of subsequent holes were drilled with a center bit without taking any core; however, the sections of the holes through the ore-bearing horizon were carefully cored. When drilling through the ore horizon, detailed records were kept in the field by the core inspector who checked the depths of each core section, marked the cores, sacked the sludge samples, and looked after core and sludge samples until delivered to the assay office. All cores containing lead were assayed; the results were compared with assays of the corresponding sludge samples, which were used merely to check the core assays. The latter were employed later in calculating tonnage and grade of ore reserves. The drilling was done with equipment owned and operated by the company, and the cost was $0.95 to $1.40 per foot of hole.

The practice described differs from that at some other properties in the district where the cores are employed for visual inspection only and are filed for reference without being assayed, the assays being made upon the sludge samples.

In developed areas diamond drills are employed on the surface chiefly to explore areas outside the boundaries of the old stopes, and underground to test floors of old stopes for ore, which may have been left in early years, and to drill test holes too deep for light hammer drills. Underground drills average 40 to 50 feet of hole per shift. For surface-drilling " B " size rods, core barrels, and bits making 1¼-inch cores are employed, while for underground work " E " rods with "A" bits making 1⅛-inch cores are used.

Test-hole drilling.—As mining progresses test holes are drilled into the ribs, backs, and floors of the stopes to delimit the irregular ore boundaries. Large areas in the Southeastern Missouri district were worked in the early days in which much ore was left in the backs and floors of old stopes. Recently these backs and floors

---

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

have been explored systematically by test-drilling, and large tonnages of ore have been found in some of them. As a rule, light hammer drills are used for this work, but heavy deep-hole hammer drills have also been employed. The use of light drills is confined to holes 22 feet or less in length.

In active stopes the drills are operated from the bottom of headings to test the back before the bluff or bench is stoped up. For testing old backs in high stopes the drills are operated from scaffolds suspended from the roof, a unique feature of practice in the district. These scaffolds have been described previously by Jackson.[2] At some mines the light hammer drills are mounted on a pneumatic-feed leg.

The drills are usually operated dry, and the cuttings are caught in a pan which fits around the front end of the drill. The steel is made up in 2-foot changes starting with a 6-pointed rose bit having a 1¾-inch gage and followed by bits with gage changes of ⅟₁₆ inch up to 100 feet in length. For greater depths chisel bits are used with gages calipered to fit the hole. The cuttings from each 2 feet of hole constitute separate samples which are boxed, tagged, and sent to the assay office, where they are assayed for lead.

Poston [3] has described the use of heavy hammer drills for prospecting in the Southeastern Missouri district. These drills have been employed for holes less than 100 feet deep, using sectional steel threaded and connected by outside couplings. Of 75 holes drilled, totaling 2,750 feet, an average of 35 feet in depth, the deepest was 78 feet; most holes were inclined at angles of 30° above or 35° below the horizontal. The machine was operated by one man who averaged 35 feet of hole per shift with an occasional 50-foot shift. The labor cost was approximately 16 cents per foot of hole. The cuttings are collected as with the light hammer drills.

TRI-STATE DISTRICT

The ores of this district occur in zones of fracturing and brecciation in flat-lying beds of cherty dolomitic limestone, usually at depths of less than 350 feet. Exploration is carried on almost exclusively by churn-drilling from the surface, although sometimes churn drills are employed underground; and hammer drills are used for testing the margins, backs, and bottoms of the workings.

Reigart [4] has discussed the methods and costs of exploring a tract of ground in this district, has pointed out the capital risks assumed in connection with an exploration campaign, and has given cost figures covering actual drilling in the district. The following notes on churn-drill exploration are abstracted largely from Reigart's paper.

Churn-drill exploration.—In spite of certain favorable factors—especially the comparatively shallow depths at which the ore bodies

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

[3] Poston, Roy H., Leyner Drill in Underground Prospecting: Eng. and Min. Jour., vol. 118, November 1924, pp. 856–857.

[4] Reigart, John R., The Cost of Developing to the Operating Stage and Equipping a Small or Medium Sized Mine in the Tri-State Lead and Zinc District: Inf. Cir. 6591, Bureau of Mines, 1932, 18 pp.

lie—this district does not differ much from most other mining districts in that financial casualties have been numerous. In the past the number of failures was multiplied because preliminary exploration was insufficient and it was undertaken and mining operations begun by inexperienced people.

As attention was transferred from the shallower parts of the district around Joplin, Mo., and Galena, Kans., to the Oklahoma-Kansas field, exploration hazards were increased. In this field there are no surface indications of ore, and it can be discovered only by drilling through overlying surface material and shale. Several ore deposits were brought in by drilling only a few holes, helping to keep alive the idea that the district is a " poor man's camp."

Much drilling only proved that large areas were barren of ore, and $50,000 to $275,000 apiece was spent by various companies in prospecting in the heart of the Picher field without finding commercial ore deposits. In numerous instances 40-acre tracts were drilled unsuccessfully by several prospectors in succession, 100 to 150 holes being drilled at a cost of $30,000 to $50,000, without finding ore, where valuable ore was found later. There are also on record numerous instances where several good holes on a tract indicated continuous ore bodies, and mines were opened up and milling plants erected only to find that the ore was not continuous and the enterprise was a total loss. Actual records show that $1,500,000 was spent during a 12-year period for drilling 6,690 prospect holes in Cherokee County, Kans., on properties that were abandoned without any ore being found. An additional $3,000,000 was spent for 9,700 holes on properties which later were brought into production but which have not yet paid back the investment and may never do so. (However, many mines have been opened in consequence of the results of churn drilling and have returned handsome profits.)

The churn drills employed in this district are usually operated by gasoline engines and move under their own power, so that moves from one hole to another are made very quickly. The holes are generally 6¼ inches in diameter and are cased through the overburden and shale down to the more or less solid underlying formations. Where it is anticipated that the underlying formations will be badly broken or will contain many boulders the holes are started with an 8-inch or larger bit to allow for casing through the bad sections of the formation.

Five-foot runs are made except when the hole is in ore; it is then customary to make 2½-foot runs. At the end of each run the cuttings are bailed out and kept in a separate pile. The cuttings are examined carefully, since from the characteristics exhibited information is obtained as to what formations are being penetrated and whether or not the conditions favor ore occurrence. The cuttings are panned at the end of each run; and if enough sphalerite or galena appears in the pan all the cuttings are carefully mixed, and a sample is taken for assay.

When the cuttings are very fine all material bailed from a run is often saved in tubs and allowed to settle and the water decanted, to get the most accurate sample possible.

Many reliable drilling contractors in the district will drill a tract at the prevailing rate of $1 per foot. Where the drilling is to be in ground known to be broken or bouldery, thus requiring extra care and casing, the rate per foot is increased or the contractor paid $25 to $30 per day regardless of footage.

If the tract to be drilled adjoins one or more operating mines or properties upon which ore bodies have been found, a guide is thereby furnished for locating the first holes. If not, 5 or 6 holes usually are first drilled across the tract, spaced about 200 feet apart (on a 40-acre tract), in an effort to pick up ore. These holes may give valuable information for locating subsequent holes. Holes in limestone areas indicate an unfavorable condition, but silicified areas found at horizons known to carry ore are favorable. An effort is then made to obtain by further drilling the outline of the silicified area and to determine whether it is a " run ", a pocket, or an annular occurrence. If one or more holes contain ore and if the shape and size of the silicified areas are determined, the rest of the drilling is confined primarily to tracing ore concentrations and to developing tonnage.

It is impossible to forecast the number of holes required to determine whether there is or is not enough ore to warrant opening a mine. Whereas some properties have been underdrilled before a mine is opened, complete drilling of a tract (which sometimes requires as many as 250 holes) involves heavy capital expenditure and high interest charges thereon over the life of the mine. Roughly, the rock-tons of ore proved before the investment required for opening a mine is made should be 300,000 to 350,000 tons or, expressed in terms of concentrates, 15,000 to 20,000 concentrate-tons. From 60 to 100 holes may be required to prove the necessary tonnage, which would mean 15,000 to 25,000 feet of drilling. Thus, to explore a 40-acre tract one should expect to spend $35,000 to $40,000 for drilling, sampling, assaying, supervision, and consultation.

Following are some examples of the actual amount and cost of exploratory drilling at various mines in the district before expenditures were made for mine development and equipment.

Property	Acres	Holes drilled	Holes in ore	Average depth of holes, feet	Cost of drilling
1	80	129	35	250	$35,000
2	104	129	52	279	53,980
3	320	162	49	326	66,015
4	105	120	43	305	37,482
5	160	219	84	330	79,497
6	60	90	----------	300	27,256
7	80	138	55	235	40,839
8	40	78	47	300	26,450

Some companies make a practice of " shale drilling "; that is, the depth of the bottom of the Cherokee shale bed (lying above the ore-bearing formation) is determined, and when it is found to be in the normal position drilling of the hole is stopped at that point. When deep shale is encountered, indicating the presence of a " slump " or syncline, the drilling is continued to depths of 280 to 350 feet.[5]

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

In recent years an exhaustive study of the structural relationships of the known ore deposits has been made, and the various beds of the Boone formation have been differentiated.[6] Maps of much of the Picher-Miami field have been made showing the flexures and shear zones which provide favorable conditions for ore deposition. The knowledge so gained has proved valuable as a guide to exploration.

Churn drills are sometimes used underground in this district to explore for suspected ore horizons below beds being mined.

Test-hole drilling.—Test-hole drilling is employed underground to test for bottom ore, which often occurs in depressions in the ore body or in narrow, irregular "runs" or channels below the stopes, and to explore the walls for ore bodies, which may be separated from the workings by ribs of barren rock. There are many such barren ribs 20 to 100 feet thick in the brecciated deposits.

Both light hammer drills and heavy-duty machines have been employed for test-hole drilling; the light drills are suitable for drilling short test holes and the heavy drills for the longer holes. Netzeband[7] has discussed the use of the heavy-duty drills for deep-hole prospecting.

The drills are heavy drifter machines with independent rotation. In the Tri-State district it is often necessary to "set up" on the bottom of high stopes, which precludes use of the column mounting usually employed for these machines; they are therefore mounted on heavy tripods. Only two changes of steel are used. The hole is started with a 3¼-inch cross bit, and this gage is used to 50 feet in depth, when a 3⅛-inch bit is substituted to finish the hole. This practice slows the drilling speed in some types of ground because the gage wears off very rapidly; however, in such ground dropping the gage would not help much unless the changes were frequent, and frequent changes would entail the use of a very large starter bit. When hard ground is encountered and the loss of gage is excessive, reaming with the same size bit is first attempted; if not successful the hole is lightly squibbed. Squibbing is not employed unless absolutely necessary, as it may pocket the hole; pockets are undesireable, since the heavier minerals tend to settle in them and later may salt the samples. The holes are drilled with water. For deep drilling they are usually drilled at an angle of 10° above the horizontal for the flat holes and at angles of plus 20° to 30° for prospecting upper horizons. At angles above 30° too much weight is thrown back on the machine by the drill steel.

In sampling the holes a powder box is placed below the collar of the hole, and all the cuttings from a 3-foot run are caught therein. The water is drained; the cuttings are thoroughly mixed; and a 5-pound sample is taken, sacked, and sent to the surface for examination and for assay if necessary. Netzeband further states:

> From checks on churn-drill sampling it has been found that ore reserves valued from churn-drilling have milled out 10 percent higher than the drilling

---

[6] Fowler, George M., and Lyden, J. P., The Ore Deposits of the Tri-State District (Missouri-Kansas-Oklahoma): Am. Inst. Min. and Met. Eng., Tech. Pub. 446–I, vol. 39, January 1932, 46 pp.

[7] Netzeband, W. F., Underground Deep-Hole Prospecting at the Eagle-Picher Mines: Trans. Am. Inst. Min. and Met. Eng., vol. 75, 1927, pp. 35–41.

showed, and it is thought that sampling with deep-hole machines will compare favorably with churn-drill sampling.

He gives the following costs for deep-hole drilling:

	Cost per foot
Labor (operating)	$0.82
Maintenance labor	.06
Repairs	.99
Explosives	.01
Powder, oil, and depreciation	.24
Total	2.12

Banks [8] gives quite a different cost for the Waco section, his figure being $0.644 per foot. Even if allowance is made for possible differences in the character of ground and depth of holes drilled, this cost probably covers only the operation. Banks states further that underground churn drilling costs $1.923 per foot. In the Waco area the cuttings from deep-hole drilling were found to be useful indications of the presence of mineral but proved unreliable when assayed for grade determinations.

Piston drills were also employed at Waco for prospecting floors to depths of about 17 feet at a cost of $0.573 per foot. When the collar of the hole was carefully cleaned before drilling was begun, the samples were found to be fairly reliable.

EASTERN TENNESSEE DISTRICT

In eastern Tennessee both churn and diamond drills are used for exploration from the surface in advance of mining. Holes are not located geometrically. Figure 16, *A*, shows the no. 2 ore body at Mascot, as outlined by the surface drilling when no. 2 shaft [9] was sunk. The drill holes are shown on the plan by circles and on the section by dotted lines. After the mines are opened additional holes are drilled as needed to insure proper mine development. Churn-drill cuttings are mounted on cards for study, and the final logs of the holes are built up from them. All bailings from each 3-foot advance after ore has been reached are sampled and assayed. Specimen cores from diamond-drill holes are filed for formational records. In ore-bearing ground all core is included in samples taken for assay.

Drifts, crosscuts, and raises are seldom driven solely to obtain information on ore. When such developments are in mineralized ground grab samples are taken after each round is blasted. An underground diamond drill is in constant use for exploration. Frequently drifts or crosscuts are driven to provide drilling stations. The drift-diamond-drill mode of exploration is employed principally in the deeper parts of the mine. Test holes are often drilled in the backs of stopes with ordinary drilling machines.

EDWARDS DISTRICT, NEW YORK [10]

The Edwards mine, Edwards district, was opened on the outcrop of an ore body that was accidentally discovered in making a road

[8] Banks, Leon M., work cited.
[9] Coy, Harley A., Mining Methods and Costs, American Zinc Co. of Tennessee, Mascot, Tenn.: Inf. Circ. 6239, 1930, 11 pp.
[10] Knaebel, John B., Mining Practice at the Edwards Mine of the St. Joseph Lead Co., St. Lawrence County, N.Y.: Inf. Circ. 6586, Bureau of Mines, 1932, 25 pp.

cut; however, most ore bodies do not apex at the surface. They are sulphide replacements of folded dolomite beds, occur as lenticular masses 5 to 25 feet thick and 100 to 200 feet long on the strike, and persist down the dip to vertical depths of 900 to over 1,700 feet. The

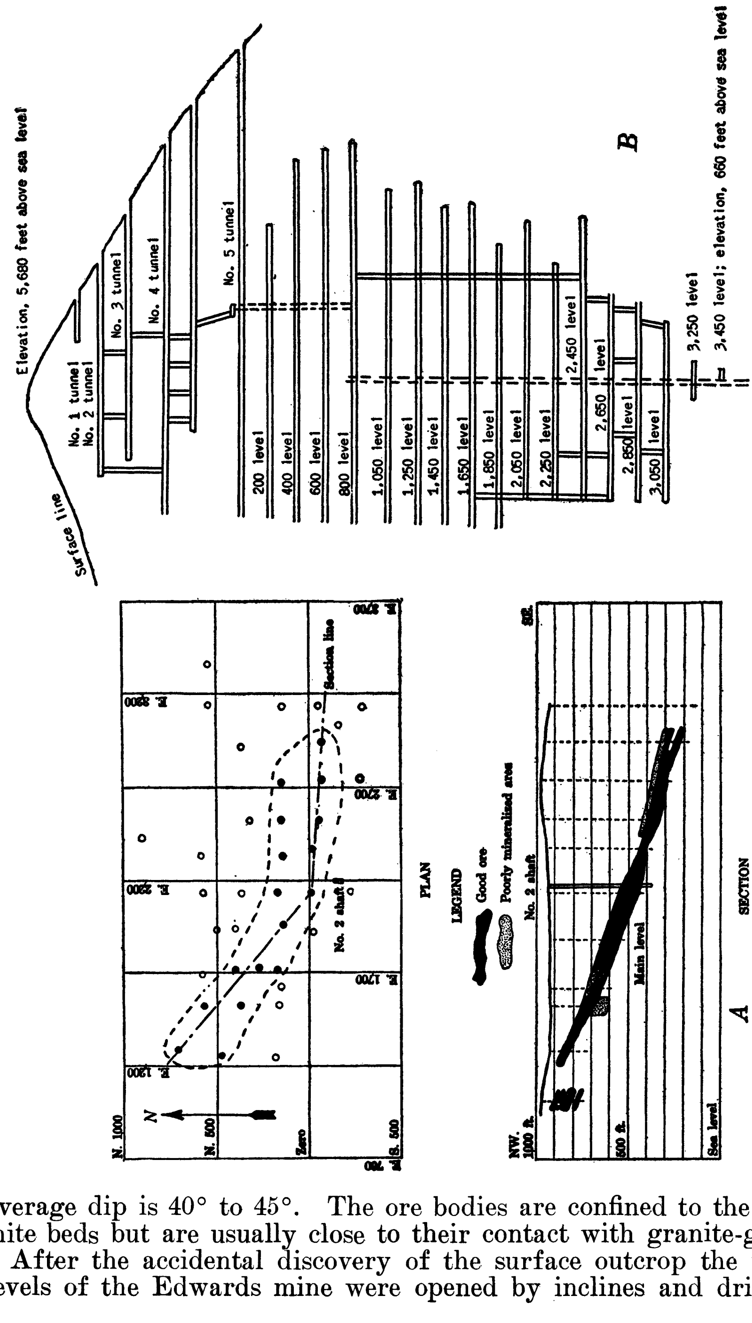

FIGURE 16.—*A*, No. 2 ore body as outlined by surface drilling at time of sinking no. 2 shaft, Mascot, Tenn.; *B*, generalized vertical longitudinal projection, part of Morning mine, Mullan, Idaho.

average dip is 40° to 45°. The ore bodies are confined to the dolomite beds but are usually close to their contact with granite-gneiss.

After the accidental discovery of the surface outcrop the upper levels of the Edwards mine were opened by inclines and drifts in the ore; and lateral exploration was originally by means of drifts at favorable horizons under the gneiss hanging wall and by crosscuts driven into the walls.

The present method involves drifting in the ore, exploring the walls for parallel ore shoots by means of drill holes and occasional crosscuts, and drilling below the levels to establish the extension of the ore before the shaft is sunk deeper. In drilling for downward extensions advantage is taken of the arrangement of the ore bodies around the nose of a tilted fold.

Lateral drilling is conducted to locate suspected ore bodies and to prospect the walls for possible parallel ore shoots. For this purpose both diamond drills and deep-hole drills have been employed. In searching for suspected extensions of an ore body, the hole is directed toward its projected location. If the hole cuts ore, a crosscut is driven without further drilling; if the hole fails to cut ore, other holes are drilled until the ore is located or proved to be nonexistent in that area.

In drilling for downward extensions of an ore body the probable extension is projected, and a hole is drilled from a point in the hanging wall toward the projected location at the horizon to be explored. If the ore is not cut in this hole other holes are drilled until the ore body is located, or until the management is convinced that it does not continue down to the lower horizon. These holes are generally drilled about 45° below the horizontal to cut the ore normal to its dip.

No attempt is made to drill for the downward extensions of hanging-wall deposits, since this would require driving long crosscuts to secure suitable locations from which to drill. Drilling from footwall workings would result in cutting the hanging-wall deposits at flat angles, which would give misleading results.

Diamond drilling.—In diamond drilling the holes are usually deflected considerably when they are deep. Core recovery is good, and visual inspection of the cores is sufficient to enable the observer to classify the material as ore or waste. The sludge is not saved; the core is sometimes split and assayed when in ore. All core in ore is saved and filed in suitable boxes. Representative samples of country-rock core are also saved. During 1930, 3,386 feet of underground diamond drilling were done at an average cost of $2.05 per foot; the drilling was distributed over 10 holes averaging 338 feet in depth.

Long-hole drilling with hammer drills.—Prospect drilling with heavy hammer drills is a regular feature of routine exploration at this mine. The equipment is a heavy drifter-type machine having independent rotation and using sectional steel; it is employed for drilling flat holes into the walls in search of blind parallel ore and splits from the main lode. The holes are drilled at angles of not over 20° above or below the horizontal and are usually not deeper than 50 feet. The cuttings from the holes are caught and assayed. The ore mineral, sphalerite, is more brittle and friable than the dolomite and other gangue minerals. This results in an excess proportion of sphalerite in the cuttings, and in computing the grade of the ore as determined by the assays the assay values are reduced 25 percent.

COEUR D'ALENE DISTRICT, IDAHO

Underground drifting, crosscutting, sinking, and raising constitute the principal methods of exploration in this district, although some test-hole drilling is done in the course of mining operations to define the margins of the ore bodies and to locate deposits in the walls of the workings and separated therefrom by waste rock. The topography of the country is steeply mountainous, the surface largely being covered by mantle rock, soil, and vegetation. Ore occurrences at different mines vary somewhat, but in general they are in zones of shearing or as simple fissures dipping at angles from 40° to vertical. These conditions do not favor exploration by churn or diamond drilling from the surface. Some deposits were first discovered in surface outcrops, while others were discovered by underground exploration at depth.

Bunker Hill & Sullivan mine.—The original discovery of galena ore was made in 1885 on the hillside above the present town of Wardner, where it outcropped in Milo Gulch about 1,500 feet vertically, or 2,250 feet measured on the dip of the lode, above the level of Kellogg tunnel which was driven into the base of the hill from the valley and completed in 1904.

The ore bodies are in quartzite in severely faulted areas, the irregular area in which they occur being composed of a brecciated mass of quartzite about 1,000 feet wide by 6,000 feet long, bounded by three major faults. The dip of the ore bodies is about 40° to 50°.

Since their discovery the deposits have been explored and developed by tunnels driven at successively lower levels to intersect the brecciated zone and by drifting, crosscutting, sinking, and raising. The lowest levels are developed by inclined shafts sunk from the Kellogg tunnel from which exploration has been conducted in a similar manner. (See fig. 7, *B*, p. 24.) After a station is cut at a new level crosscuts are driven to the ore zones, which are then drifted on to the extremities of the ore.

Page mine.[11]—There are two fissure veins on this property, dipping 40° to 60° to the south and terminating on the west against a thrust fault. The veins are thought to lie in an extension of the broad Bunker Hill ore zone and are in quartzite. There are no well-defined walls, and the enclosing quartzites are broken and shattered for 20 or 30 feet into the walls of the vein proper. The outcrops of the veins are low grade, and little minable ore was found above the 300-foot level. Exploration is by the usual underground methods of sinking, crosscutting, drifting, and raising. Test raises in the hanging wall and crosscuts in the foot wall are run at frequent intervals to prospect the full width of the ore zone. Diamond-drilling and long-hole drilling with hammer drills have not been very satisfactory.

Morning mine.[12]—The Morning vein is a fissure vein in quartzite in a zone of close sheeting which follows or cuts the slaty cleavage of the quartzite at small angles and dips at an angle of 80° to ver-

[11] Berg, J. E., Mining Methods at the Page Mine of the Federal Mining & Smelting Co., Page, Idaho: Inf. Circ. 6372, Bureau of Mines, 1930, 8 pp.

[12] Wethered, C. E., and Coady, Leo J., Mining Methods at the Morning Mine of the Federal Mining & Smelting Co., Mullan, Idaho: Inf. Circ. 6238, Bureau of Mines, 1930, 13 pp.

tical. This vein outcropped at the surface, and early exploration was by opencuts, shallow shafts, and tunnels. (See fig. 16, *B*, p. 55.) Two other veins on the property were worked near the surface.

The ore zone in the Morning vein has been virtually continuous from the surface to the 3,450-foot level, approximately 5,000 feet below the outcrop, so that exploratory work has been comparatively simple. To open up a new level the shaft is sunk to the required depth, a crosscut is driven to the vein, and drifts are then run on the vein.

In an endeavor to locate other ore bodies some diamond-drilling, of which there is no available record, was done in the past.

The walls of the vein are prospected by a 225-pound long-hole machine. One and one-quarter inch, round, lugged, shank steel is used, starting with a 3¼-inch bit and reducing the bit gage one sixteenth inch at each change. Most holes are planned to be only 25 or 30 feet deep. The average depth of 62 holes was 46 feet, the deepest being 129 feet.

Hecla and Star mines.[13]—There are several veins in quartzite country rock on these properties. The Hecla and Intermediate ore bodies occur along shear zones and are nearly vertical; however, they converge at the 1,400-foot level. Other ore bodies are of distinctly fissure-vein type, and one is on the extension of the Morning vein. Only one vein outcrops on surface.

Exploration is by crosscutting and drifting. As all fissure veins found have a strike of 45° to 70° southeast, long crosscuts are driven S. 33° W. Because ore-bearing fissures may occur in the south wall of the Hecla ore body this wall is crosscut at fairly frequent intervals. Short crosscuts are driven from the stopes at right angles to the fissures to meet possible branching stringers of ore. In drifting and stoping, holes are drilled into the walls at frequent intervals to prospect for parallel stringers of ore.

Any fissure or shear zone encountered by crosscuts and having the proper strike and dip must be carefully watched, as the ore-bearing fissures have been known to " close up " very short distances from commercial ore.

PARK CITY DISTRICT, UTAH

The principal ore outcrops in this district were merely small knobs of high-grade lead-silver ores which for years eluded search by prospectors.[14] In 1872 the Ontario mine was discovered, then other ore-bearing veins were soon found. Rich float found near Woodside Gulch led to the discovery in 1888 of the outcrop of the vein system now comprising the ore bodies of the Silver King Coalition Mines Co.

Silver King Coalition mines.[15]—The ore deposits are of two types, lode deposits and bedded replacement deposits; mineralization is associated with fissures carrying lode deposits in the quartzite or in the limestone.

---

[13] Foreman, Charles H., Mining Methods and Costs at the Hecla and Star Mines, Burke, Idaho: Inf. Circ. 6232, Bureau of Mines, 1930, 21 pp.

[14] Dailey, M. J., Mining Methods and Costs of the Silver King Coalition Mines Co., Park City, Utah: Inf. Circ. 6371, Bureau of Mines, 1930, 12 pp.

[15] Dailey, M. J., work cited.

The bedded or replacement deposits in limestone are lenticular, the margins being irregularly lobed; some lobes or arms extend hundreds of feet from the main deposits. Certain deposits have been productive for over 10,000 feet along their strike. In the Silver King mines 95 percent of the output is from bedded deposits.

A small amount of diamond drilling has been done, largely to obtain geologic information. The principal method of prospecting consists of drifting in the quartzite beneath the Park City formation and, wherever practicable, upon mineralized fissures parallel to the strike of the ore bodies. By this procedure small stringers of sulphide ore are often found. By raising on such stringers into the ore-bearing members of the formation ore bodies may be opened and developed. Advantages of this method of exploration are: (1) Ore bodies are more economically developed and stoped by means of drifts underneath them; (2) the cost of drift maintenance in quartzite is less than in limestone or ore; and (3) commercial ore deposits are found in quartzite which otherwise might be overlooked.

Another method is to follow the ore downward by means of small winzes or laterally by drifts along the contact or in favorable limestone beds.

Park Utah mine.[16]—The siliceous silver ores and the lead-zinc-silver ores of this mine occur mainly in fissures 3 to 80 feet wide in limestone and quartzite and dip at angles of 40° to 55°. The ore shoots vary in length up to a maximum of 900 feet and are lenticular. The district has been developed entirely by the usual underground methods (shafts, crosscuts, drifts, and raises). In the past some diamond drilling was done at various mines, but no ore discoveries resulted. At one mine good core recovery was obtained from three 300-foot holes, which gave valuable information regarding the formations penetrated.

Some bedded ores have been found in the district. A common method of exploring for bedded ores is to drive drifts and crosscuts beneath favorable horizons. Small stringers of ore, on which raises are driven to their intersection with the ore horizon, are found in driving these headings. If ore is not found by a raise the favorable bed is further explored by drifting.

TINTIC DISTRICT, UTAH

Tintic Standard mine.—This mine is in the eastern part of the Tintic district where no well-defined outcrops indicative of mineral were recognized and where none of the commercial ore bodies outcropped. The earliest discoveries were in other sections of the district, where there was high-grade ore at or near the surface.

The principal ore bodies in this mine are replacements in limestone at or near its contact with quartzite and are all 500 or more feet beneath the surface. (See fig. 10, *B*, p. 31.) These ore bodies when adjacent to quartzite fault faces dip from 45° to vertical and when distant from the faces occur as tabular replacements of flat-lying limestone beds 6 to 200 feet high.

[16] Hewitt, E. A., Mining Methods and Costs at the Park Utah Mine, Park City, Utah: Inf. Circ. 6290, Bureau of Mines, 1930, 18 pp.

According to Wade,[17] all prospecting and exploration are based on a thorough study of the geology. Each successive step in the progress of development is a result of those preceding. Preliminary knowledge was gained from careful mapping of the geology of the old workings; at present, all new workings are closely observed, and the geology is recorded by the geological department.

The beds are explored through shafts and drifts driven into the ore measures where these measures are intersected by the fissure system.

Structural geological features determine the ore bodies in the district, and shafts, raises, winzes, drifts, and crosscuts are used almost exclusively for prospecting. A very small amount of diamond-drilling has been done, and at present churn-drilling is being used successfully to determine the depth of the rhyolite flows (see fig. 10, *B*), the thickness of underlying limestone, and the top of the quartzite in outlying sections of the property.

Ore bodies are found by drifting in favorable horizons on the quartzite-limestone or shale-limestone contacts. Outlines of the ore bodies are clearly defined by structural features and are developed only in the process of mining. The hanging wall is drilled at short intervals during mining, and the foot wall is prospected by the extraction drifts.

Chief Consolidated mines.—Deep-hole prospecting at the Chief Consolidated mines has been described by Dobbel.[18] The following is abstracted from Dobbel's paper.

Ore occurrences throughout the district are typically in lenses on bedding planes or breaks and in connecting pipes, without accurate geometrical trend, so that the ratio of the total length of development in drifts and crosscuts to the number of square feet of ore exposed is very high. The irregular trend of the ore requires excessive cutting of the whole area for offshoots and connecting links which are always present. No regular system of prospecting and development can be planned without overlooking some ore bodies or segments of ore bodies. Diamond-drilling was tried, but costs were high on account of high diamond wear and diamond loss. Churn drills were tried underground but were soon found unsuitable to the conditions. At the same time the ordinary hammer drill was used to drive prospect holes with jointed steel. Much of Dobbel's article is devoted to the mechanical features of the hammer drills and jointed steel and other equipment developed and the methods of operation which came into extensive use in the Chief Consolidated mines for prospecting by deep-hole drilling. Total costs of drilling were given as $0.97 per foot of hole, including operating and miscellaneous labor, steel and steel sharpening, depreciation of equipment, supervision, and blasting out for set-up room.

The deep-hole hammer drill is used principally for finding new ore or locating indications that might open up to ore when explored. A study of the drill cuttings is important in locating the "good" and "bad" limestones and mapping the limestone areas. The loca-

---

[17] Wade, James W., Mining Methods and Costs at Tintic Standard Mine, Tintic District, Utah: Inf. Circ. 6360, Bureau of Mines, 1930, 21 pp.

[18] Dobbel, Chas. A., Deep-Hole Prospecting at the Chief Consolidated Mines: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 677–689.

tion of faults or breaks that may be mineral bearers can also be determined by study of the cuttings. In taking samples the cuttings from each 3 feet of hole are caught in a powder box at the mouth of the hole. All cuttings are sacked and sent to the examination room for quartering and inspection, then are discarded or sent to the assayer. The cuttings are treated much the same as diamond-drill core and have proved as valuable in the Tintic district.

BUTTE DISTRICT, MONTANA

Black Rock mine.—The methods of prospecting and exploration at this mine have been described briefly by McGilvra and Healy.[19] The Black Rock mine of the Butte & Superior Mining Co. is at Butte on a vein a few feet to 120 feet in width and with an average dip of 80°. Other veins occur, some being of commercial importance, though narrow. The veins are in granite, and both walls are marked by talc seams. McGilvra and Healy state further:

> At present all prospecting is done by means of drifts and crosscuts. In the past diamond-drilling and deep-hole prospect-drilling with heavy Leyner machines were tried. The chief objection to long-hole drilling in the Black Rock mine is that the information gained is entirely negative. A 1-inch stringer of soft ore would salt the samples from the hole for the next 10 or even 20 feet of drilling. The diamond drill has been used mainly in prospecting for new ore bodies. The chief objection to this form of prospecting is the rather high cost for the meager information gained. It is believed that most of the veins in this district are worthy of considerable lateral development which can not be done with the diamond drill as cheaply as by crosscutting and drifting. The major problem in exploratory work is faulting. The search for faulted segments of ore bodies is simplified to a great extent by studying the detailed geological maps of the underground workings.

BARKER DISTRICT, MONTANA

Block P mine.[20]—This mine is located upon the principal fissure vein of the Barker mining district. The ore body occurs in a large syenite chimney or stock cut by rhyolite dikes. In early days the vein was prospected by test pits. As the workings were extended the vein was explored by driving adits along the vein at depth and by sinking vertical shafts with crosscut connections from the shafts to the vein. Diamond drills or other deep-hole drilling equipment have never been used in the district.

PECOS DISTRICT, NEW MEXICO

Pecos mine.[21]—The ore bodies are irregular, disconnected, often overlapping lenses of sulphides in shear zones replacing schist in a general area of altered pre-Cambrian granite which has been intruded by fine-grained diorite. The lenses of ore often pinch out abruptly both horizontally and vertically. Cross-sections of the mine indicate that the ore outcropped in places.

---

[19] McGilvra, D. B., and Healy, A. J., Methods of Mining at the Black Rock Mine, Butte & Superior Mining Co., Butte District, Mont.: Inf. Circ. 6370, Bureau of Mines, 1930, 16 pp.

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

[21] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Inf. Circ. 6368, Bureau of Mines, 1930, 21 pp.

Prospecting by drifting consists of following the principal ore shoots along their strike. If the ore body pinches out an attempt is made to follow the most promising of possibly several shear zones. This is difficult, as the ore occurrence is erratic, and no sure guide to ore is known. The shear zone is crosscut or drilled at frequent intervals. The walls and downward extension are explored by diamond drilling from the underground workings, and extensions along the strike are explored by diamond drilling from the surface. The walls are also tested for parallel ore shoots by deep-hole drilling with Leyner machines and section steel. The holes are usually 50 to 100 feet long and are drilled with 1¼-inch steel using only two bit gages.

SANTA RITA-HANOVER REGION, NEW MEXICO

Ground Hog mine.[22]—The outcrop on the Ground Hog claim was mostly covered by overburden, and the exposed portions were barren quartz. The early shipments were lead-copper ores encountered about 100 feet below the surface. The lead-zinc mines of the district are mostly in limestone. The Ground Hog ore body differs from others found in the district and is formed in a fault fissure with foot and hanging walls consisting in many places of fault gouge several feet thick. (See fig. 12, p. 38.)

Prospecting on the lower levels has been confined to drifting and sinking on the vein. On the 300- and 400-foot levels there has been considerable crosscutting to the foot and hanging walls. No prospect drilling has been done.

BATTLE MOUNTAIN DISTRICT, COLORADO[23]

The Eagle mine of the Empire Zinc Co. is at Gilman in the Battle Mountain district, Colorado. The first discoveries were on outcrops consisting of small areas of gossan and iron-stained limestone which were exposed in the gulches and on the hillside along a porphyry-limestone contact. Gold-silver ores were discovered later in a quartzite formation exposed in Eagle Canyon below the porphyry-limestone contact. The oxidized ores in the limestones and quartzite were worked out many years ago. The lower mine workings in the limestone horizon exposed attractive faces of low-grade complex sulphides (pyrite, blende, and galena), but in the quartzite horizon the sulphide ore shoots were small, unprofitable pyrite lenses.

The sedimentary formations in which the ore bodies are found dip at an angle of about 15°. The most important deposits are in the Zebra dolomitic limestone, about 125 feet thick. Just above the limestone bed but separated from it by a thin bed of shale and quartzite is a porphyry sill averaging 60 feet in thickness; and above this are shales, quartzites, and limestones.

The deposits occur in the form of typical blanket (manta) ore bodies, as long cylindrical deposits elliptical in cross-section, and as chimneys.

---

[22] Richard, F. W., Mining Methods and Costs at the Ground Hog Unit Asarco Mining Co., Vanadium, N.Mex.: Inf. Circ. 6377, Bureau of Mines, 1930, 13 pp.

[23] Borcherdt, W. O., The Empire Zinc Co.'s Operation at Gilman, Colo.: Eng. and Min. Jour., vol. 132, Aug. 10, 1931, pp. 99–105.

Lary, Howard N., Exploration Methods and Costs at a Colorado Zinc Mine: Eng. and Min. Jour., vol. 132, Oct. 26, 1931, pp. 353–355.

New ore bodies or downward extensions of known blanket ores are sought largely in the Zebra limestone and in the feeding vents or chimneys below, through which it is thought the mineralizing solutions ascended. Exploration headings are driven parallel to the strike along the Zebra formation, about midway between the top and bottom of the Zebra formation or at the base thereof.

Drill holes are employed to supplement drifts in exploration work and are spaced at 50- to 100-foot intervals along the drifts. They are drilled at angles of 40° to 50° above the horizontal. Drilling is done with deep-hole hammer drills, and individual holes vary in depth up to 225 feet. An interesting feature is the use of stellited bits which hold their gage and cut so well that many holes are bottomed with the original bit. Some bits have drilled as much as 125 feet before losing gage. Another feature, described at some length by Lary, is the surveying of deep drill holes. Diamond-drilling is employed only to a limited extent for drilling tests below the levels and for prospecting deep chimney ores.

In developing blanket ore " up-dip " from the level several methods are employed. One commonly used is to raise to the back through ore that has been developed by drilling over a strike drift, then to crosscut through the blanket from hanging wall to foot wall, repeating these two operations step fashion, to the next level above. Another method is to drive an incline up through the ore.

Chimney ore bodies which range in height from 80 to 125 feet are developed by drifting through the ore on the strike of the formation, then crosscutting northeast and southwest at 50-foot intervals to the waste walls.

Diamond-drilling in ore has been unsatisfactory as a whole because the loose, open character of much of the sulphide ore required cementing of the holes after each shift and resulted in slow speed and poor core recovery. Core recovery ran from 0 to 40 percent in soft ore and from 50 to 75 percent in hard ore. In country rock, however, diamond drilling is much more satisfactory and is employed in drilling for structure in limestone, quartzite, and shale.

SUMMARY OF EXPLORATION PRACTICE

The foregoing examples illustrate the methods of exploration employed in different types of lead and zinc deposits in the United States.

Exploration from the surface by churn or diamond drilling is the accepted method in flat-lying bedded deposits, whether or not they outcrop at the surface; and in some important districts drilling is the only method employed before development, erection of mining and milling plants, and productive mine operation.

The choice between churn and diamond drills depends largely upon the nature of the ground, particularly whether or not it is homogeneous, badly fissured, or brecciated. These factors influence not only the cost of the drilling but also the amount of core recovery that may be obtained with the diamond drill. The chief advantage of the diamond drill is that the core gives an actual section of the formations cut for visual inspection and assay. This section is complete only to the extent that core recovery is complete. If the formation is badly broken much core will be ground up, and the ground portion will give no more information than churn-drill cuttings. In such ground there is also danger of breaking or losing the carbons, with attendant high cost.

In drill exploration it is desirable to cut the ore body at right angles to its dip or nearly so, and in steeply dipping tabular deposits this cannot be done with the churn drill. The choice of preliminary exploratory methods therefore lies between diamond-drilling, hammer-drilling, and the ordinary underground methods of sinking, crosscutting, drifting, and raising. The surface topography, dip, size, shape, and regularity of the ore body and the nature of the mineralization and structure of the ore and wall rocks largely influence the choice of method.

If the deposit dips into the hill and the hillside is steep, long holes started some distance from the outcrop will be required to test the deposit at depth if they are to cut the deposit normal to the dip. These long holes would probably cost more than underground exploration for obtaining the same or a smaller amount of information. Furthermore, long holes are apt to deviate considerably from their intended course and may give misleading information as to the true thickness of the ore and other formations, also as to the location of the ore and structural features of the deposit, unless they are carefully surveyed. If the ore bodies are erratic and irregular in outline the holes may miss them entirely.

If the deposit dips with the slope of the hill the holes will be shorter, and exploration by drilling may be considered.

The nature of the wall rocks and the ore may be such that core recovery will be poor, loss of carbons through breakage or loss of bits may be high, or differences in friability of ore and gangue minerals may result in salting of the samples. Under such conditions underground exploration methods would be preferable, although drilling might supply valuable structural information.

If the ore bodies are very irregular in outline drill holes may pass through a wide section of ore that is only a local bulge, or they may cut very thin sections of the ore body or a barren zone close to good thicknesses of ore on one or both sides; in either case drilling results will be misleading. In irregular ore bodies a combination of sinking, drifting, and crosscutting with drilling is often the best method. Indeed, the authors believe that there are few ore bodies where diamond-drilling or test-hole drilling with hammer drills cannot be used to advantage, to some extent at least, to supplement drifting and crosscutting either during exploration, development, or actual mining.

Occasionally in tabular ore bodies, which are very regular in shape, strike, and dip, lie between well-defined walls, and are continuous on the dip, all that seems necessary is to sink to the next level, crosscut to the ore, and drift thereon. Even in such instances, however, there are often parallel ore shoots and blind lodes, particularly in shear zones, which could be located or proved absent cheaply by drilling but which otherwise would be missed or found only at the expense of considerable blind crosscutting.

Even where assays from drill cores or cuttings are misleading, diamond or test-hole drilling can often be used to advantage to determine geologic structure, the location of favorable beds, contacts between formations, and the presence or absence of mineralization at different horizons and different points at the same horizon. Such information has great value where the geology is complex but where the favorable, formational, and structural conditions have been determined.

For short test holes less than 20 feet in depth, ordinary hammer drills can be used. For deeper holes (from about 50 feet to, in some instances, 200 or more feet) the heavy drifter type of machine with independent rotation and sectional drill steel has often been used to advantage. For still deeper holes, or for short holes where recovery of a core for visual inspection is essential and good core recovery can be obtained, the diamond drill is widely used.

No special comment on exploration by sinking, drifting, and crosscutting is deemed necessary; however, the geological characteristics of the deposits must be considered in directing exploration. Each step should be based upon previous results and upon knowledge already gained of the favorable formations and structures. As the work progresses careful notes should be taken of all geological features, which should be recorded on maps and cross-sections. The foregoing applies equally to drill exploration.

The cost per foot for drifting and crosscutting may be 3 to 10 or more times as much as that for drilling, and this point is important. Thus, any work that fails to reveal the presence of ore by drifting and crosscutting will result in much higher fruitless expenditure than if done by drilling. Speed of exploration is another consideration. Hence, while crosscutting and drifting may give more complete information in some instances, results can often be obtained much sooner by drilling, and if drilling at a given point fails to reveal mineralization much less time is lost. If drilling reveals mineralization but does not prove whether commercial material is present crosscutting can then be done, and little expense will have been incurred and little time lost in preliminary drilling.

In some examples of exploration practice that have been cited, the operators have found drill exploration of little value; and the reasons for the sole use of sinking, crosscutting, drifting, and raising are obvious. The foregoing discussion is not intended as a brief for drill exploration, but this method has received special emphasis since its field of usefulness as an adjunct to underground exploration is perhaps not as widely appreciated as it should be. The subject of drill exploration has recently been discussed in considerable detail by Jackson and Knaebel in Bureau of Mines Bulletin 356.

SAMPLING AND ESTIMATION OF ORE DEPOSITS

In the preceding section the methods commonly employed for exploring lead and zinc ore deposits were described and discussed. Reference was made to obtaining samples during exploration; however, the details of sampling practice and the handling and treatment of the samples were not discussed.

Since the object of exploration is to find ore it is unnecessary to discuss the need of obtaining samples of the ore disclosed to determine what is of commercial grade. The grade of material which can profitably be mined and treated fluctuates with market prices, which in turn may vary widely depending upon general economic conditions and the relation between consumptive demand and productive capacity for the particular metal or metals considered. Before a mine is opened for productive operation it is important to determine as far as possible the tonnage of commercial material and the grade of ore to be expected from the property as a whole and the tonnages and grades of ore in different parts of the deposit. Similar information is also important at a producing mine to determine operating policy and to control the grade of ore mined. These factors also form a basis for determining the price that should be paid for a property or the value of the stock. It is therefore pertinent to discuss briefly methods of sampling and estimating ore deposits containing zinc and lead.

Samples of ore in place are taken to determine the average grade of the ore in the deposit or in different blocks from these small portions; they are taken at points distributed throughout the deposit at intervals designed to represent the average grade of the whole.

Individual samples of ore are not representative of the deposit as a whole. An ore deposit is not a homogeneous mass but comprises mixtures of particles of ore minerals that vary in size and in degree of purity, and particles of gangue composed of different proportions, kinds, and sizes of various minerals. Some ore deposits are much more homogeneous than others, and the ore particles are distributed more uniformly throughout the mass; however, abrupt changes in the percentage of ore minerals within a few feet or even inches are common. A 5-pound sample cut from even 1 cubic foot of ore may accurately represent only the 5 pounds of material constituting the sample and may vary considerably in grade from the grade of the block.

Sampling of an ore deposit is only reliable, therefore, when the samples are so numerous and so spaced throughout the deposit that their individual errors in accurately representing the corresponding sections sampled are compensating; in this way their average grade approaches the actual grade of the deposit.

Large, regular, low-grade deposits in which the valuable mineral is distributed throughout with relative uniformity are obviously the easiest to sample within the required degree of accuracy. In such deposits the sections sampled may be spaced at wider intervals. Erratic deposits wherein bands of rich ore alternate with bands of barren or low-grade material or rich patches of ore occur irregularly distributed are the most difficult to sample accurately, and a relatively large number of samples will be required.

The method of sampling must be suited to the characteristics of the ore body; otherwise, the results will be misleading, regardless of how many samples are taken. If the samples are taken incorrectly errors in the individual samples will be cumulative rather than compensating.

SAMPLING METHODS

The methods commonly employed for sampling zinc and lead deposits may be classified as follows:

1. Drill-sampling: (*a*) With churn drills; (*b*) with core drills, principally diamond drills, and (*c*) with hammer drills.
2. Face-sampling: (*a*) Pick or chip samples, and (*b*) regular channel or groove samples.
3. Grab-sampling.

In some operating mines visual inspection of the face may be all that is required to class the material as ore or as waste, and sampling and assaying may be regarded as unnecessary and expensive refinements; this applies particularly to some high-grade sulphide ores. Long experience with a low-grade ore body may enable the operator to gage very closely the grade of any given face by eye, and the grade of output is controlled by visual inspection aided by daily returns from assays of samples of crushed ore at the concentrator. This practice might be termed "visual sampling."

Before a newly discovered ore body, particularly in a new district, is exploited it is obviously important to determine the grade of the ore as accurately as possible, and one or more sampling methods in the above classification is usually employed.

Methods of exploration by drilling have been described in a previous section, and some mention was made of the spacing of drill holes and the manner of taking samples therefrom. In the pages that follow are given some examples of drill-sampling practice at a number of lead and zinc mines, with examples of other sampling methods.

Face samples as used in this report are those taken from exposed faces of ore or of ore and waste. Pick or chip samples are those taken more or less at random, but at points well-distributed over the face, by chipping off small particles of material at each point. Approximately equal weights of material should be taken at each point, and the quantities taken from bands or patches of different classes of material should be in proportion to the exposed area of each.

A more generally approved method is to cut one or more grooves or channels of uniform width and depth across the face at right angles to the bedding, schistosity, or banding. If the bands are distinct and of decidedly different hardness it is best to sample each separately, giving the assay value of each a weight in proportion to its width when the assays are combined to determine the average grade of that face. Such samples are termed "channel samples." They may be cut with a pick in soft ground or with moil and hammer in hard ground.

Grab samples usually consist of handfuls of broken ore taken more or less at random from points over the top of a pile of broken ore or from the tops of cars of ore. Such samples are often, although not always, very inaccurate. Grab samples can be taken more accurately by employing a small scoop and taking a scoopful of broken ore from each of several points spaced equally over the pile, the points being spotted by measuring with a tape or other means.

Once the method of sampling has been determined it should be followed as mechanically as possible to eliminate the human tendency to select too large a proportion of the better material for inclusion in the sample.

The subject of sampling has been treated at some length in a recent Bureau of Mines publication.[24]

EXAMPLES OF SAMPLING AND ESTIMATING PRACTICE

The following examples describe the methods of combining assays and computing the average grades as well as the sampling practice.

SOUTHEASTERN MISSOURI DISTRICT

Poston [25] has described the methods of sampling and estimation at No. 8 mine, and the following is quoted from his description:

In common with many lands adjacent to known minable areas, the No. 8 shaft area was first tested with the surface diamond drill to determine whether or not lead ore existed. The first few holes showed that lead ore did exist at depth; and eventually some 80 holes were put down on this property at irregular intervals, and sufficient data were accumulated upon which to predicate extensive underground development. These 80 holes comprise some 48,000 feet of drilling, of which records have been very carefully kept.

The first few holes were cored from the surface to the sand, but after the approximate ore horizon had been definitely established the first few hundred feet of each subsequent hole were "plugged"; that is, drilled with a center bit, thereby eliminating the core through this section. This procedure saved both time and money without loss of pertinent geological data.

When drilling through the ore horizon detailed records were kept in the field by a core inspector. This inspector checked the depths of each core run when drilled, suitably marking the cores in the core containers; he looked after sludge samples, checked hole numbers, and stored the cores until they were transferred to the assay room at the main plant.

All cores containing lead were assayed to determine the exact lead content, these results being compared with the assays of the corresponding sludge samples. Sludge samples were used solely as a check upon core sample assays, the latter values being used for all ore-reserve calculations.

The cost of this drilling varied from $0.95 to $1.40 per foot, the variation being due largely to physical conditions beyond the company's control. The drilling was done with equipment owned and operated by the company.

Methods of sampling and estimation.—No mine or stope sampling is done at the No. 8 mine, but the ore is sampled and assayed separately from that from the other mines at the concentrator. All mining plans are based upon data obtained from the surface drilling.

No exploration drifts are driven into virgin ground because the irregularity of the occurrence of the ore and its uniformly flat dip would make this form of exploration much more expensive and less effective than the present system of surface diamond-drilling.

Ore at mine No. 8 that carries less than 14 feet-percent lead content is not mined; in other words, a mine stope must be at least 7 feet in height and must carry at least 2 percent lead throughout the stope face. When more than 7 feet is involved, a calculation is first made to determine whether the entire thickness is minable. If it is not, only that portion averaging 2 percent lead and more than 7 feet in thickness is included as ore. Ore in place is calculated at 12½ cubic feet per ton, and broken ore weighs approximately 2,000 pounds per 20 cubic feet. For tonnage estimates ore is assumed to exist in a circle 150 feet in diameter using the diamond drill hole as its center, and the value is taken as that of the drill-core sample.

Where drill holes are irregularly spaced and are fairly close together, say, a maximum of 200 feet, a system of polygonal areas is used for the calculation of ore-reserve tonnages. The attached sketch (fig. 17) shows in detail how these polygons are formed. For example, it is desired to outline the polygon

---

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

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

around drill hole no. 66. The first step is to draw lines from the selected hole (no. 66) to all nearby holes (in this case holes nos. 6, 76, 28, 14, 67, and 51). Dotted line no. 65 to no. 66 is incorrect, since the holes included for this purpose are determined by measurements that establish the shortest diagonals of the trapezoid which is formed by the four holes under consideration.

The two holes used in each instance are those at the ends of the shortest diagonal of the trapezoid. If this rule is followed throughout, the polygons can be reproduced around each hole by anyone with the same results, thus eliminating the personal equation. Then connect each of the nearby holes with each other. This gives a number of triangles with hole no. 66 at the

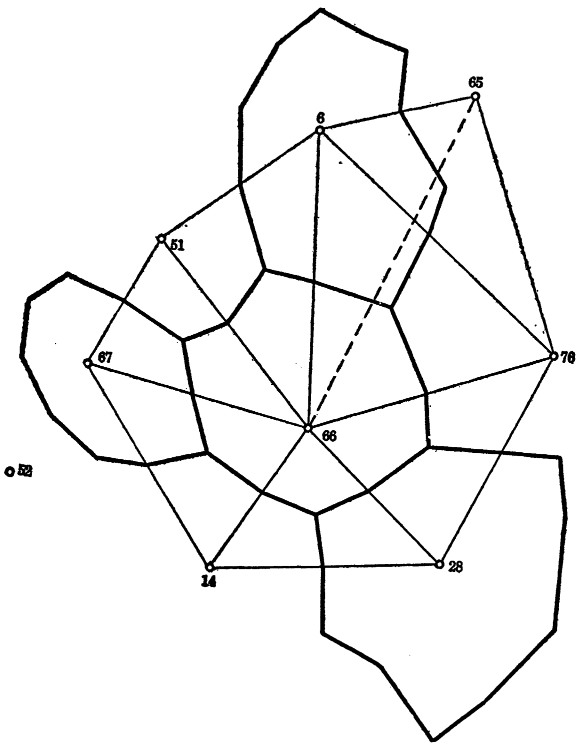

FIGURE 17.—Method of forming polygonal areas around surface diamond-drill holes, No. 8 mine, Southeastern Missouri district.

apex. The center of each of these triangles is now obtained by bisecting each side and drawing a light line from the point of bisection to the opposite apex. This is done from each angle and the center obtained. From this center a heavy line is drawn to the center of each of the three sides. These are permanent lines and, when all construction lines have been erased, result in each hole being the center of a polygon, the area of which is in proportion to the spacing of the surrounding holes.

Polygonal areas are measured by means of a planimeter, and the final equation used to obtain the tonnage figure is as follows:

$$\frac{\text{Surface area (in feet)} \times \text{estimated stope height (in feet)}}{12\tfrac{1}{2}\text{ cubic feet}} = \text{estimated tons of ore.}$$

This tonnage is assigned the lead value shown by the core assays from the drill hole around which the polygon is formed. All core samples containing lead visible to the naked eye are assayed to determine the true lead content. The core record then takes the form illustrated by the following example:

590 feet–595 feet (7.2 percent by assay).
596 feet–597 feet (1.2 percent by assay).

Then

$$\begin{array}{ll} 590\text{–}595 = 5 \text{ feet} \times 7.2 \text{ percent} = & 36.0 \text{ feet-percent} \\ 596\text{–}597 = 1 \text{ foot} \times 1.2 \text{ percent} = & 1.2 \text{ feet-percent} \\ 595\text{–}596 = 1 \text{ foot at } 1.0 \text{ percent} = & 1.0 \text{ feet-percent} \\ \hline \quad 7 \text{ feet} & 38.2 \text{ feet-percent} \end{array}$$

and

$$\frac{38.2 \text{ feet-percent}}{7 \text{ feet}} = 5.46 \text{ percent.}$$

This shows that the above section contains the equivalent of 7 feet of 5.46 percent ore. It will be noted that the 12 inches in this run between 595 and 596 has been arbitrarily assigned a value of 1 percent. This is done to partially compensate for the known loss of lead in the core sample while drilling and to make a complete calculation.

The lead is much softer than the country rock and usually shows evidence of wear where the core is broken in the lead-ore run. Experience has demonstrated that the total ore mined is always somewhat in excess of the estimate made prior to mining.

Heavy-duty air drills using jointed steel are employed to cut long prospect holes in mined-out stopes prior to their abandonment. For details concerning this practice, see article entitled "Leyner Drill in Underground Prospecting," by Roy H. Poston, on pages 856 and 857 of no. 22 of volume 118 of the Engineering and Mining Journal-Press for November 29, 1924.

Jackson [26] has described briefly sampling and estimation practice at another mine in this region. The company owns and operates its own diamond-drilling equipment. All cores are carefully examined by a core inspector who, through many years of experience in the district, can by inspection estimate very accurately the lead content of the rock. Notes are made regarding the physical character of the core, its color, porosity, and estimated lead content; these are carefully recorded. The cores are marked and filed for future reference; they are not assayed. The sludge from the drill holes is dried and assayed, and the assay results are recorded with the other information.

Light hammer drills are employed for testing backs of old stopes, floors, and marginal areas where the depth of hole required does not exceed 22 feet. It is customary to test stope backs with holes drilled at the corners of 25-foot squares over the area to be tested. The drills are usually operated dry, and the cuttings are caught in a specially shaped pan which fits around the front end of the drill. The cuttings from each 2 feet of hole constitute a separate sample. These samples are assayed for lead.

In determining the minability of an area explored by drilling, the physical characteristics of the core or cuttings, such as color and texture, are considered in conjunction with past experience as to grade at the same vertical horizon of the formation in adjacent areas, also the lead content indicated by the assays of the drill samples. Thus, a single hole showing good values at a given horizon, although surrounded by holes showing low values or only traces of lead at this elevation, is usually justification for development of

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

an area, provided adjacent areas at the same horizon have been productive.

In estimating ore proved as a result of drill exploration, a minimum mining thickness of 7 feet and 15 feet-percent lead are taken as a basis for the calculations; that is, ground that will not average 15 feet-percent over a 7-foot thickness is excluded from the ore reserves. Under this rule 1 foot of 15 percent lead would be included and would mine out better than the minimum grade, since the 1 foot of high-grade ore has a much higher specific gravity than the other 6 feet of barren ground which must be included under the rule requiring a minimum stope height of 7 feet. Furthermore, experience has demonstrated that all rock within the boundaries of an ore body contains some lead. When more than 7 feet are involved, a calculation is first made to determine whether the entire thickness is minable, that is, has an average grade of 2 percent lead or better. If not, the minable thickness and grade are determined by a "cut-and-try" method.

TRI-STATE DISTRICT

Netzeband[27] has cited an example of prospecting and valuing a deposit in this district. The following notes are abstracted from his paper.

Exploration is by means of churn-drill holes from the surface. A complete log of every hole is kept, regardless of whether or not mineralization of commercial grade is found. Five-foot screws are usually run, a sample of each screw is piled on the ground, and the depth of the sample is indicated on a tag. When ore is found the cuttings are allowed to settle in a tub, and the clear water is decanted. They are thoroughly mixed, and a 10- to 20-pound sample is taken. Care must be taken to mix the sample well, for the sphalerite is friable and tends to sludge, throwing a considerable portion of the sphalerite into the fines and slimes, whereas the galena tends to settle to the bottom of the sample. About 5 pounds are taken from the original sample for assay.

The formations are not logged in detail but are usually marked as lime or flint, the color being recorded in each instance. Ore grades are variously described. To those less than 2 percent zinc sulphide and 1 percent lead sulphide the term "shines" is applied. Most companies have assays made for higher grades. Several companies collect samples of cuttings from every hole and paste them on a card for future reference.

The use of heavy Leyner-type hammer drills for drilling test holes from underground workings has been mentioned in a previous section.

Method of mine valuation.—To illustrate the method of mine valuation used in this district Netzeband takes as an example a typical property which has been mined out and compares estimated tonnages of raw ore and concentrates with actual recoveries. The property

[27] Netzeband, William F., An Example of Prospecting and Valuing a Lead-Zinc Deposit: Eng. and Min. Jour., vol. 127, June 8, 1929, pp. 913–916.

embraces a 40-acre tract, the outline of the ore body as estimated and as actually mined and the locations of the drill holes being as shown in figure 18.

The following table shows the method of combining and averaging the assays of the drill-hole samples.

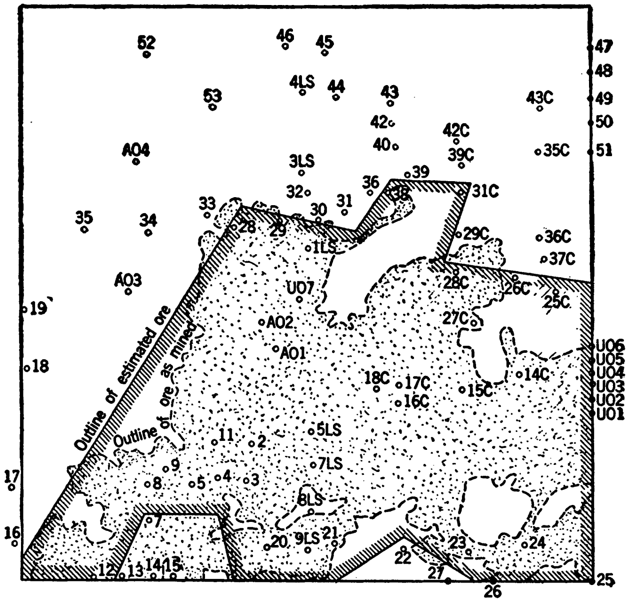

FIGURE 18.—Ore body in Tri-State district, showing drill holes, outline of estimated ore, and outline of ore as mined.

Method of combining and averaging assays of drill-hole samples

Hole no.	Depth, feet	Face, feet	ZnS, percent	PbS, percent	Total, percent	ZnS, feet-percent	PbS, feet-percent	Total, feet-percent
1	240-245	5	2.00	----------	2.00	10.00	----------	10.00
2	245-260	15	3.43	----------	3.43	51.45	----------	51.45
3	240-255	15	4.00	----------	4.00	60.00	----------	60.00
4	250-260	10	1.00	----------	1.00	10.00	----------	10.00
5	222-238	16	9.35	0.12	9.47	149.60	1.92	151.52
8	222-237	15	21.33	.04	21.37	319.95	.60	321.55
9	230-240	10	6.18	.85	7.03	61.80	8.50	70.30
11	225-240	15	15.97	2.52	18.49	239.55	37.80	277.35
12	225-235	10	7.30	.12	7.42	73.00	1.20	74.20
20	222-242	20	6.57	.56	7.13	131.40	11.20	142.60
21	235-240	5	9.97	.21	10.18	49.85	1.05	50.90
23	235-254	9	2.38	----------	2.38	21.42	----------	21.42
24	225-235	10	.72	12.85	13.57	7.20	128.50	135.70
25	240-245	5	----------	2.50	2.50	----------	12.50	12.50
26	231-245	14	5.36	1.78	7.14	75.04	24.92	99.96
28	230-240	10	3.22	.08	3.30	32.20	.80	33.00
29	235-252	17	4.83	.38	5.21	82.11	6.46	88.57
38	230-255	25	4.02	----------	4.02	100.50	----------	100.50
14C	237-248	11	4.60	.90	5.50	50.60	9.90	60.50
15C	231-248	17	5.46	1.73	8.19	92.82	29.41	122.23
16C	238-247	9	4.00	3.00	7.00	36.00	27.00	63.00
17C	237-241	4	6.00	----------	6.00	24.00	----------	24.00
18C	220-250	30	3.00	----------	3.00	90.00	----------	90.00
25C	205-230	25	3.50	----------	3.50	87.50	----------	87.50
26C	245-255	10	4.00	----------	4.00	40.00	----------	40.00
27C	240-255	15	1.90	.11	2.01	28.50	1.65	30.15
28C	245-250	5	3.50	2.50	6.00	17.50	12.50	30.00
31C	240-255	15	4.50	----------	4.50	67.50	----------	67.50
AO1	225-235	10	2.29	1.95	4.24	22.90	19.50	42.40
AO2	230-245	15	1.35	5.56	6.91	20.25	83.40	103.65

Method of combining and averaging assays of drill-hole samples—Continued

Hole no.	Depth, feet	Face, feet	ZnS, percent	PbS, percent	Total, percent	ZnS, feet-percent	PbS, feet-percent	Total, feet-percent
5LS	250–265	15	7.10	0.07	7.17	106.50	1.05	107.55
7LS	245–262	17	4.10	2.03	6.13	69.70	34.51	104.21
8LS	240–245	5		1.50	1.50		7.50	7.50
9LS	240–255	15	1.50	.20	1.70	22.50	3.00	25.50
UO1	237–242	5		1.50	1.50		7.50	7.50
UO2	230–235	5		2.82	2.82		14.10	14.10
UO4	235–245	10	7.05		7.05	70.50		70.50
UO5	230–245	15	15.40		15.40	231.20		231.20
UO6	235–242	7	7.72		7.72	54.04		54.04
Total		486				2,607.08	486.47	3,093.55
Average		11.85				[1] 5.36	[2] 1.00	[3] 6.36

[1] Percent ZnS. [2] Percent PbS. [3] Total percent ZnS and PbS.

The area of the block as obtained by planimeter is 827,520 square feet. As shown in the table the average height of the ore is 11.85 feet, the average grade of the ore being 5.36 percent zinc sulphide and 1 percent lead sulphide, a total of 6.36 percent. The unminable area within the block, which is left for pillars or because it is too low grade to mine, is taken as 12½ percent of the block. Mill recovery before the use of flotation (when most of the ore was mined) was 80 percent of the total sulphide contained.

The average prices of lead and zinc concentrates from 1920 to 1927, inclusive, were respectively $90 and $41 per ton, the prices being based upon 60 percent metallic zinc in the zinc concentrates and 80 percent metallic lead in the lead concentrates. Rock-ton costs for the district averaged $2 per ton during the years considered. Costs of drilling, sinking, and opening the mine for production and of a mill were estimated at $1 per ton of raw ore for 1 year—$90,000 for the total cost of bringing the mine to the production stage.

The following calculations, based on the foregoing figures, were made to determine the probable net profit to be expected from operating the property:

Area	827,520 square feet.
Height of face	11.85 feet.
827,520×11.85	9,806,112 cubic feet.
$\frac{9,806,112}{12.5}$	784,489 tons of rock.
784,489×0.875 (12½ percent being allowed for pillars and unminable material)	686,428 tons.
686,428×0.0536	36,792 gross tons zinc sulphide.
36,792×0.80	29,434 net recoverable tons zinc concentrates.
686,428×0.01	6,864 gross tons lead sulphide.
6,864×0.80	5,491 net recoverable tons lead concentrates.
29,434×$41	$1,206,974 gross value zinc concentrates.
5,491×$90	$494,190 gross value lead concentrates.
Total gross value of concentrates	$1,700,984.
Cost of mill and development	$90,000.
Cost of mining and milling 686,428 tons at $2	$1,372,856.
Royalty at 12½ percent of gross sales, $1,700,984×0.125	$212,623.

SUMMARY

Total gross value of concentrates	$1,700,984
Royalty	212,623
Net value of concentrates	1,588,361
Cost of mining and milling	1,372,856
Cost of mill and development	90,000
Total cost of producing concentrates	1,462,856
Net profit	125,505

From these figures the estimated net profit would not seem to justify the capital expenditure required; however, it has been found by experience that actual production is usually greater than estimated production. The property illustrated milled 721,904 tons of rock, recovering 37,322 tons of zinc concentrates and 13,933 tons of lead concentrates compared with an estimated net recovery of 29,434 tons of zinc concentrates and 5,491 tons of lead concentrates. The actual percentage recovery was 5.17 percent zinc sulphide and 1.93 percent lead sulphide.

EDWARDS DISTRICT, NEW YORK [28]

Very little assaying of cores or drill cuttings is done at Edwards, as visual inspection is usually enough to determine whether the material is ore or waste. If the core or cuttings are judged to be of doubtful grade or if they are obtained from points where ore was not expected, they are assayed. Sludge assays are reduced 25 percent in value before being entered in calculations for grade of ore reserves. The sludge from short test holes drilled into the walls of stopes is examined but not assayed.

Routine channel sampling of ore faces is not practiced at present, although some was done in earlier years. Channel samples have generally proved inaccurate, owing to brittleness of the ore minerals and resultant salting of the samples. The sampling errors are fairly constant, however, and estimates based on channel sampling check closely the results of actual mining, provided the channel-sample assays are reduced 25 percent in value. At the Edwards mine visual inspection of the faces by experienced observers is all that is required ordinarily to judge the grade of ore within the required degree of accuracy, and on account of the expense of sampling channel samples are not taken regularly.

Considerable channel sampling was done at the Balmat property, which was recently opened up a few miles from Edwards.

Due to local swells and pinches in the veins and the unreliability of sampling results, ultrarefined methods of estimating tonnages and grades are not employed. Estimates of ore volumes are made by multiplying the average area on adjacent levels by the distance between them. Tonnages are computed from volumes by applying a factor of 10 cubic feet per ton of ore in place. However, good ore actually occupies only 8.5 to 9 cubic feet per ton. The value of a block is sometimes figured as the tonnage times the average assay

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

value (averaged by the feet-percent method), the sample assays having been reduced 25 percent before being entered in the calculations. More often the grade of ore is estimated on the basis of that of similar ore in neighboring stopes, the grade of which is known from actual milling. Blocks have been estimated by both methods, with remarkably close agreement in the results.

COEUR D'ALENE DISTRICT, IDAHO

Page mine.—Where ore is found in development drifts and raises cut samples are taken at 10-foot intervals, and assays are made for lead and zinc with occasional assays of composite samples for silver. The widths and assays of samples are posted on sample maps. In estimating tonnages a factor of 9 cubic feet per ton is used.

Hecla and Star mines.[29]—Only in recent years has a systematic method of sampling new development work been employed. Now all development work is watched, and when valuable mineral is found samples are taken at intervals depending upon the grade and continuity of the ore. The position and width of the sample are recorded. Enough samples are taken in a face to show the values of the different classes or grades of material exposed. After assay returns are received they may be combined to give the values of parts or all of the face. The grades of all faces are calculated to diluted mining widths, as it is assumed that a certain portion of the wall rock will be mined with the ore and that a minimum width of stope will be carried. The amount of dilution and the minimum width depend upon the condition of the vein and country rock. The tonnage and assay values of the various blocks of ore are calculated by the prismoidal formula. The factor to be used for cubic feet per ton depends upon the grade of ore in each particular block.

Morning mine.—At the Morning mine development drifts spaced 200 feet apart vertically are driven on the vein. When these drifts are run, development samples are taken every 10 feet across the vein. Assays are made for lead, zinc, and silver and, together with the widths of samples, are posted on assay maps. The mining widths of partly mined blocks of ore are shown on the stope maps and are posted monthly. Ore reserves are figured yearly for the entire mine and are based upon the average stope widths and the properly weighted assays from development work done during the year. A factor of 9 cubic feet to the ton is used for estimating tonnages.

PARK CITY DISTRICT, UTAH

Silver King Coalition mines.—Lewis [30] makes the following statement regarding the sampling at these mines:

> Most samples are grab samples taken from cars underground, though a few grab samples are taken from the cars when they reach the surface. They are taken according to the judgment of the sampler. The assay returns from these samples are from 5 to 10 percent higher than those from the sampling mill.

---

[29] Foreman, Charles H., Mining Methods and Costs at the Hecla and Star Mines, Burke, Idaho: Inf. Circ. 6232, Bureau of Mines, 1930, 21 pp.

[30] Lewis, Robert S., Mining Methods of the Silver King Coalition: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 485–497.

Newly opened stopes are given particular attention for a few days. Groove samples taken with a pick are cut daily. It is seldom necessary to sample first-class ore, so most of the samples are of second-class or milling ore.

Park Utah mine.—At the Park Utah mine the grade of the ore in any particular fissure is very uniform, so that sampling is a simple problem. Grab samples consisting of several handfuls of ore from each car are taken at the chutes and combined into one general sample for each chute on each shift. Very little sampling is done in the stopes because the physical characteristics of the ore are such that with a little practice it is easy to distinguish between ore and waste by inspection.

The estimation of ore reserves is based upon the width, length, and height of each ore shoot as determined by mine-development work. The assay value of unstoped ore is calculated from samples taken in adjacent stoped areas and in drifts and raises.

TINTIC DISTRICT, UTAH

Tintic Standard mine.—The methods of sampling and the estimation of tonnage and grade of ore have been described by Wade.[31] The following information is contained in Wade's article.

All material hoisted, both ore and waste, is sampled three different times; waste used for filling is sampled twice.

The first samples taken are face samples. These are taken by specially trained men under direct supervision of the engineering department. Part of the mine is assigned to a sampler; it is his duty to sample daily and to record the assays of every operating face in that portion of the mine. Samples are taken in drifts and raises and every square-set face in the stopes opposite both the cap and girt. These samples are grooves made with a hand pick at right angles to the dip of the vein and weigh approximately 5 pounds. Where a face comprises two or more streaks of different hardness or mineral content, each is measured and sampled separately. These samples are sent to the assay office by 10 a.m., and the determination is completed by noon. Each shift boss is furnished a complete assay sheet showing all the samples taken, and he uses these sample sheets in determining the breaking and classifying of ore in his section of the mine. In addition, the assays are copied on small cardboard tags and nailed on the cap over the face from which the samples were taken, giving the miners full knowledge of the ores in which they are working, which is most important when sorting is required. Assay maps of each floor in the stope, raise, or winze and continuations of drifts and crosscuts are brought up to date twice a week by the samplers. These assay maps contain the dates of mining and details of lacing and filling for each set, and the face assays. On this sheet colors indicate the month in which the ore was mined. Average values so obtained are checked against actual smelter returns. These records are used in estimating ore reserves and by the management in controlling metal shipments.

The second sampling comprises hand-grab samples taken by the trammers as they draw the chutes. These are termed box samples;

---

[31] Wade, James W., Mining Methods and Costs at Tintic Standard Mine, Tintic District, Utah: Inf. Circ. 6360, Bureau of Mines, 1930, 21 pp.

one is taken from each section of the stope and one from each drift. Box samples are used to check face samples; in this way each round of waste broken is resampled before a gob is entered.

The third sampling includes "composite" grab samples of each grade of ore and of the waste hoisted, taken from the cars on the surface by the top carmen as a final check before the ore enters the freight cars.

The grab samples taken by the shift bosses and top carmen are used to determine the value of the ore produced that day. Reports by the shift bosses of the number of cars from each section, which are checked by the top carmen's report of cars of each grade of ore, are used in estimating the tonnage for the day. These figures are employed in calculating daily gross earnings.

Ore reserves are clearly defined by structural limits and are computed by weighted assays from the assay maps.

BARKER DISTRICT, MONTANA

Block P mine.—Vanderburg[32] writes as follows:

The ore minerals are easily recognized, so that visual inspection of the working faces, supplemented by regular assay samples, is relied upon for determining the grade of the ore. For geological reference purposes the drifts and raises are sampled each week by taking cuts across the faces with a hand pick. In taking these samples structural irregularities of the vein are disregarded. Due to the irregular occurrence of the ore in lenses and the variation in width of the mineralization no estimates of ore reserves are made.

BUTTE DISTRICT, MONTANA

Black Rock mine.—McGilvra and Healy[33] state:

All development headings are sampled after each round is blasted. Stope breasts are sampled at each alternate square-set. Samples are taken with a hand pick. Structural irregularities of each breast sampled are considered in taking the samples. Each band of ore is sampled separately, and the widths of the bands of ore and waste are measured. The average grade of the ore when broken is calculated from the measurements and the values of the samples. Ore reserves are estimated by multiplying the length, height, and average width for individual blocks as outlined by development work. The tonnage factor is 10 cubic feet for each ton of ore in place. In estimating positive and probable ore the characteristics of each section of the vein, as determined by past mining operations, are taken into consideration.

PECOS DISTRICT, NEW MEXICO

Pecos mine.—Matson and Hoag[34] state:

All underground openings showing mineralization are sampled by cutting channels. This is usually done with a hammer and moil, although in hard ore a compressed-air hitch cutter may be used. Channels are cut normal to the schistosity where possible; and the length of channel included in individual samples is made to correspond with bands of high-, medium-, or low-grade ore, or bands of waste. The grade of ore is generally easily estimated by inspection, and sampling is unnecessary for stoping purposes. Sampling is usually done along the boundary lines of each floor of a stope previous to filling. The tonnage produced by each stope is determined by survey and checked by the car tally, which is first corrected to agree with the tons milled. No record

---

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

[33] McGilvra, D. B., and Healy, A. J., Methods of Mining at the Black Rock Mine, Butte & Superior Mining Co., Butte District, Mont.: Inf. Circ. 6370, Bureau of Mines, 1930, 16 pp.

[34] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Inf. Circ. 6368, Bureau of Mines, 1930, 21 pp.

is kept of the grade of ore produced by individual stopes. This could be done only by taking grab samples of cars, which would probably be inaccurate with ore of this character; it is believed that all other purposes are better served by the samples taken from ore in place in the stopes.

In diamond drilling a core recovery of about 75 percent is usually obtained. The sludge is always assayed, but experience shows that sludge assays are not reliable. Therefore, when the sludge assay shows mineralization the core is split; one half is assayed, and the other half is kept for permanent record. This procedure prevents overlooking of low-grade mineralization in the drill cores.

Ore reserves are computed from closely spaced vertical transverse sections. Grade estimates are based on the average of all assays available in a given block. No account is taken of probable or possible ore. Therefore, the ore outline is extended only a short distance into unknown ground.

SANTA RITA-HANOVER REGION, NEW MEXICO

Ground Hog mine.[35]—Samples are cut with hammer and moil in all drifts and crosscuts. Channels 3 inches wide by about 1 inch deep are cut across the vein at 5-foot intervals. In crosscuts the channels are cut about 3 feet above the floor; a sample is taken every 5 feet. Raises are not sampled.

Tonnage and grade are calculated for each block, the blocks being defined by main levels and sublevels about 65 feet apart on the slope of the vein, or 50 feet vertically, and by raises between levels which are 50 feet apart along the strike. The average area of the top and bottom of the block normal to the dip, multiplied by the length of the block along the dip, gives the volume of the block. A factor of 9 cubic feet per ton is used to convert volume to tonnage. The average assay value of the ore in each block is obtained by multiplying the width of the ore where each sample was cut by the average assay of the cut and dividing the sum of all such products by the sum of all the widths.

Experience has shown that this method gives too high assay values and too low tonnages. One reason for the high assays is that the sublevels are driven close to the hanging wall where the best ore is concentrated and are not opened over the full width of the vein as are the main levels. The low estimated tonnages are explained by the fact that in stoping a considerable amount of low-grade ore is mined from the footwall. About 25 percent more ore is mined than is estimated, and the grade is correspondingly lower.

Close sampling is not necessary for operating purposes because experienced men can estimate the grade easily by eye. Samples, therefore, are taken only from doubtful ore as it is found in drifting or stoping. Production samples are obtained daily; they are taken at the collar of the shaft and include a small shovelful from each car hoisted, combined into one sample for each shift.

BATTLE MOUNTAIN DISTRICT, COLORADO

Gilman.—As mentioned previously, deep-hole hammer drills are employed extensively for prospecting at Gilman, Colo. The technique of deep-hole drilling and the equipment used have been discussed in detail by Lary.[36]

---

[35] Richard, F. W., Mining Methods and Costs at the Ground Hog Unit, Asarco Mining Co., Vanadium, N. Mex.: Inf. Circ. 6377, Bureau of Mines, 1930, 13 pp.

[36] Lary, Howard N., Exploration Methods and Costs at a Colorado Zinc Mine: Eng. and Min. Jour., vol. 132, Oct. 26, 1931, pp. 353–355.

According to Lary the sampling of drill cuttings is simple. The cuttings are caught in a powder-box, and a sample is obtained of every 6 feet drilled by taking a section of the contents of the box from top to bottom. The sample is placed in a can to which is attached a wooden tag bearing the hole number and a record of the part of the hole represented by the sample. The ore is of a grade that does not necessitate close sampling and is so friable that the sample is usually contaminated. A composite assay of ore in a drill hole is regarded as accurate enough, however, to be used in ore-reserve calculations. Confusion because of salting by waste is avoided by daily reports from the drillers which show the type of rock penetrated. From general experience the driller can distinguish between the drilling of sulphide and the drilling of waste by the color of the return water and the speed of drilling.

From the laboratory where the samples are assayed, they are sent to the mine office for screening by the mine sampler, and the minus 10-mesh plus 28-mesh product is placed on a sample stick. The geologist then inspects the cuttings and writes a description of each sample, which is kept in a notebook on printed forms. Assay results, position of hole, and directional data are recorded.

The cost of deep-hole drilling over a period of years has averaged $0.828 per foot, comprising labor, $0.662, and supplies $0.166.

SUMMARY OF SAMPLING AND ESTIMATING PRACTICE

The practices at the mines described are summarized in table 4. In studying the notes on practices in different lead and zinc mines and districts one is struck by the number of instances in which visual inspection only is employed for control of stoping operations. This practice is in decided contrast to that at most copper, iron ore, and precious-metal mines described in earlier publications,[37] due largely to the nature of the ore occurrence and the characteristics of the ore and associated minerals. In many copper ores, for example, the presence of copper minerals is often obscured by their close association with much larger amounts of pyrite and its oxidized products. The precious metals often occur intimately associated with sulphide minerals, or so finely divided that they are not visible; hence, close sampling is necessary to determine even the approximate grade of the ores.

[37] Jackson, Chas. F., and Knaebel, John B., The Sampling and Estimation of Ore Deposits: Bull. 356, Bureau of Mines, 1932, 155 pp.
Jackson, Chas. F., and Knaebel, John B., Gold Mining and Milling in the United States and Canada; Current Practices and Costs: Bull. 363, Bureau of Mines, 1932, 143 pp.

TABLE 4.—*Summary of sampling and estimating practice*

Mine or district	Method of sampling					Estimates of ore reserves
	Churn drills	Dia-mond drills	Ham-mer drills	Face sam-ples, chan-nel or groove	Grab sam-ples	
Southeastern Missouri district.	--------	(1)	(2)	--------	--------	From diamond-drill samples.
Tri-State district	(3)	--------	(2)	--------	--------	From churn-drill samples.
Mascot No. 2 Mine, Tennessee.	(1)	(1)	(2)	--------	(4)	From sample assays and estimated grades of working faces.
Edwards district, New York.	--------	(1)	(5)	(6)	--------	From grades of adjacent stopes and from samples.
Page mine, Idaho	--------	--------	--------	(7)	--------	From sampling of development openings.
Hecla and Star mines, Idaho.	--------	--------	(8)	(7)	--------	Do.
Morning mine, Idaho	--------	--------	(8)	(7)	--------	Do.
Silver King Coalition mines, Utah.	--------	--------	--------	(9)	(10)	
Park Utah mine, Utah	--------	--------	--------	--------	(11)	Grade calculated from that of adjoining stopes.
Tintic Standard mine, Utah.	--------	--------	--------	(12)	(11)	From groove-sample assays posted on assay maps.
Block P mine, Montana	--------	--------	--------	(13)	--------	Not made.
Black Rock mine, Montana.	--------	--------	--------	(14)	--------	From sample assays.
Pecos mine, New Mexico	--------	(15)	(8)	(16)	--------	From cross-sections and all available assays.
Ground Hog mine, New Mexico.	--------	--------	--------	(17)	--------	From channel-sample widths and assays.
Battle Mountain district, Colorado.	--------	--------	(18)	--------	--------	Partly from composite samples of drill holes.

1 Core or sludge assayed.
2 For testing backs, floors, and walls.
3 Cuttings assayed.
4 From development headings in ore.
5 Cuttings from deep-hole drilling assayed, results 25 percent high; short test-hole samples not assayed.
6 Not regularly done, assays 25 percent high.
7 Samples assayed for lead and zinc; assays for silver on composite samples.
8 For testing walls for parallel stringers and ore bodies.
9 In milling-grade ore.
10 5 to 10 percent higher in grade than mill samples.
11 Taken from cars at loading chutes.
12 From drifts, raises, and square-set faces in stopes.
13 From drifts and raises.
14 All development headings and stope faces.
15 Sludge assayed; core assayed if mineralized.
16 All underground openings showing mineralization.
17 From all drifts and crosscuts at 5-foot intervals.
18 Sampled in 6-foot lengths.

MINE DEVELOPMENT

The term "development", as used in this report, applies to the operations involved in preparing a mine for ore extraction and includes shaft-sinking, tunneling, crosscutting, drifting, and raising. The term is often used also to cover such exploratory operations as have been discussed in the previous section. Exploration and development may be, and usually are, conducted at the same time; development headings ordinarily serve an exploratory purpose and vice versa. In most mines exploration and development continue long after stoping has begun; therefore, the exploration, development, and mining periods usually are not distinctly defined. During exploration, however, effort is directed principally toward finding ore, while during development the work is aimed principally toward preparing the mine for economical extraction of the ore. It may be possible during exploration to plan the exploratory headings with a view to their later utilization as extraction openings, and this obviously may often save time and expense in development of the property.

Some ore bodies, or parts thereof, are so irregular and erratic that the ore is explored and developed as it is stoped. In such cases, however, diamond or test-hole drilling is often used to advantage to explore the formations in advance of stoping.

The type of the ore deposit, its size, shape, and dip, the distribution of the ore shoots, the strength of ore and wall rocks, the depth of the deposit below surface, and the surface topography are determining factors over which the operator has no control in planning development. Experience and skill are necessary, however, to take utmost advantage of favorable natural features and to overcome unfavorable ones in planning development and operation and in extracting the ore. Such factors as rate of output, size of development openings, level interval, stoping method, and cost of the work, while influenced by natural features, can be controlled to some extent and should receive careful attention. The amount of money available for development, equipment, and operating capital must also be considered.

Apparently, the more complete the information provided by the exploratory work the more intelligently can development be planned. As previously pointed out, large regular bedded deposits are easiest to explore and generally simplest to develop. Such deposits are usually low-grade, however, and to operate at a profit development must be planned for a high production rate and the utmost economy in every branch of the operation.

Advantage should be taken of existing exploratory openings where possible, but it may be desirable to provide new ones to permit efficient operation. Thus, the property may have been explored from a shaft that is too small to hoist the expected production; or it may be desirable to replace an inclined shaft with a vertical one, or to substitute a shaft for a series of adits, or to drive a long tunnel to replace earlier shafts and other openings. The exploratory headings may be too small or crooked for efficient motor haulage, or the level interval may not be adapted to the stoping method to be employed. In general, the decision as to the use of existing exploratory openings should be based upon their adequacy for the expected scale and mode of operation and upon the savings in operation to be effected by new openings compared with the cost of providing them.

SHAFT VERSUS TUNNEL DEVELOPMENT

In a flat country where most of the ore deposit lies below drainage it is apparent that the ore must be mined through shafts. In a precipitous country of high relief, where much of the deposit lies above drainage and it is so located that it can be developed by a moderately long tunnel or adit driven from a point where suitable provision can be made for a surface plant and mill buildings, this type of development has obvious advantages.

From the standpoint of first cost of driving, a tunnel is much cheaper foot for foot than a shaft. The cost per foot for a shaft may be several times that for a tunnel, the exact ratio depending upon a number of factors, such as size, nature of ground, cost of power for hoisting and pumping, and amount of water to be handled. Therefore, the question of first cost largely resolves itself into consideration of the relative cost per foot for a shaft and tunnel and the relative length of each that would be required in any particular instance.

However, other factors than first cost are involved. If it is anticipated that a large amount of water will come from the mine, the cost of pumping it in a shaft mine throughout its life may be very high and will increase in direct proportion to the height of the lift and the volume of water pumped. A tunnel drains the workings above its level at no cost for pumping, and over a period of years this fact may effect substantial saving in operating cost. Even if the workings will later be extended below the tunnel level by underground shafts or winzes the water from these lower workings can be pumped to the tunnel level at less cost than to the collar of a shaft sunk from the surface, due to the reduced pumping head. Frequently, moreover, a large proportion of the water in a mine comes from upper levels, and lower levels are quite dry; therefore, most of the water may be drained off directly through a tunnel. Hoisting ore and waste in shafts is an appreciable cost item in a shaft mine, which, with the first costs of hoists and headframe, which are often large, is eliminated by operation through tunnels. Rehandling from cars to shaft pockets and then to skips, or caging of cars, also increases the direct hoisting costs.

On the other hand, operation through tunnels may involve driving excessively long headings; and, even if the tunnels are short, there may be no suitable points from which to drive them because of lack of room for surface plant, mill site, waste dump, etc.

SHAFTS

Vertical versus inclined shafts.—Shafts may be vertical or inclined. As far as operation of the shaft is concerned vertical shafts are usually preferred to inclined shafts.[38] Inclined shafts must be longer to reach the same depth, hoisting speed is usually slower, wear on ropes is greater, and if the shaft is in or close to the lode it may be disturbed and thrown out of alinement because of subsidence induced by adjacent mining or considerable ore may be tied up in shaft pillars. Sometimes, however, an inclined shaft may be the most economical method of developing a deposit. Thus, if the lode dips at a flat angle the shaft may be in or near the lode, usually in the footwall, and only short crosscuts on each level may be required to connect the shaft to the workings. Under these conditions a vertical shaft will require progressively longer crosscuts to reach the lode on successively lower levels if it is sunk in the footwall. If sunk in the hanging wall the average length of the crosscuts will

[38] Jackson, C. F., Some Notes on Underground Transportation: Inf. Circ. 6326, Bureau of Mines, 1930, pp. 23–24.

be less; but in this location it is apt to be disturbed by subsidence induced by mining operations and may be difficult to maintain, especially where it passes through the lode. Surface topography in relation to the ore deposit may make it impossible to locate a vertical shaft suitably, especially in mountainous country, while a favorable location may be available for an inclined shaft; the reverse may be true in other instances.

An inclined shaft is, generally speaking, better adapted for exploration than operating. It follows the ore on the dip, being in or close to the lode; also, virtually all the exploratory work is in ore, and a minimum amount of dead work is required. A well-known principle in exploration is to "stay with the ore" and follow it down regardless of where it goes, diminishing the likelihood of missing it in tunnels or crosscuts driven toward points where its presence has not been proved. In tabular, low-dipping, deposits this principle can be followed better by using inclines than vertical shafts.

When the outlines of the deposit and its dip, strike, and continuity have already been determined by exploratory work it is often much more desirable to sink a vertical shaft for development which can be located for maximum utility and convenience. A vertical shaft in the footwall of the ore body is generally the best method of shaft development, except for thin tabular deposits lying at flat dip angles, provided a suitable location is available.

Size of shafts.—The size of a shaft will depend mainly upon the production rate; method of hoisting (whether in buckets, skips, or cages); requirements for pipe, cable, and ladderways in the shaft; requirements for and method of handling timbers, steel, men, and supplies; and provisions for auxiliary shafts. In most States the law requires a second outlet to the surface, which may be an auxiliary shaft conveniently divided into a manway and an airway compartment.

In the Tri-State district one-compartment shafts are common, 5 by 7 or 6 by 6 feet in cross-section, cribbed through surface material, shale, and boulder ground and untimbered through solid rock. Hoisting is usually in cans or buckets holding 1,400 pounds of ore. Some later shafts at the larger mines are equipped for hoisting with skips or cages; these are usually 5 by 11 to 6 by 12 feet in cross-section. There is usually one "field" shaft or more in addition to the main hoisting or mill shaft.

In the Southeastern Missouri lead district shafts are commonly about 6 by 15 to 6 by 18 feet in the clear and are equipped for hoisting with skips in balance in two compartments, with a third compartment for pipes and power cables. Generally there are auxiliary shafts, which are used for ventilation and for handling men and supplies.

At Mascot, Tenn., the No. 2 mine is developed by a vertical 4-compartment shaft to the 520-foot level with a skip-loading station at 582 feet. This shaft is 9 by 25 feet in rock section; two compartments are used for hoisting with skips in balance, one for a cageway and the other for pipes, power cables, and ladderway. Levels below the 520 are developed by a flat, incline shaft. At the Page mine, Coeur d'Alene district, Idaho, the main hoisting shaft is an incline 7 by 18 feet in over-all dimensions and has 3 compartments. The main Hecla mine shaft in the same district is 5 feet 2 inches by 17 feet 4 inches in the clear and has 2 skip compartments, 1 cage compartment, and 1 pipe compartment. The main shaft at the Morning mine is 7 feet 3 inches by 23 feet 6 inches outside the timbers and has 4 compartments, 2 for skipways, 1 for a cageway, and 1 for a pipeway.

At the Silver King Coalition mines, Park City district, Utah, there are five vertical auxiliary underground shafts. The main hoisting shaft is 5 by 15 feet in the clear and has 2 cageways and 1 manway compartment. At the Park Utah mine the main hoisting shaft is 5 by 12½ feet in the clear and has 1 skipway, 1 cageway, and a third compartment for manway, pipes, and power cables.

The Tintic Standard mine, Tintic district, Utah, is developed by three shafts. The main hoisting shaft has 2 hoisting compartments, each 4 feet 6 inches by 4 feet in the clear, and a manway, 4 feet 6 inches by 2 feet, or about 4 feet 6 inches by 11 feet 4 inches inside timbers. One auxiliary shaft is an incline used for ventilation. The main hoisting shaft of the Chief Consolidated is 5 feet 9 inches by 15 feet 6 inches rock dimensions and has 2 hoisting compartments and a combined manway and pipeway compartment.

At the Block P mine, Montana, the main shaft is 5 feet 4 inches by 9 feet 4 inches outside the timbers and has 1 hoisting compartment and a manway and pipe compartment.

The Black Rock mine, Butte (Mont.) district has a 3-compartment main hoisting shaft 7 by 17 feet outside timbers, an auxiliary shaft for ventilation, and a third shaft for handling supplies.

At the Pecos mine, New Mexico, the main hoisting shaft is 6 feet 8 inches by 20 feet 4 inches outside the timbers. It has two compartments for hoisting, using cages hung under ore skips for handling men and supplies, one pipeway and ladderway, and a fourth compartment used for sinking when the shaft is being deepened.

At the Ground Hog mine, New Mexico, a new shaft is to replace an old two-compartment shaft. The new shaft will have 3 compartments and will be 6 feet 4 inches by 13 feet 8 inches outside the timbers.

At Edwards, N.Y., the main shaft is 7½ feet by 18 feet in rock section, or 6 feet 3 inches by 16 feet 9 inches outside the shaft timbers. There are two skip compartments and a pipeway and ladderway compartment.

The foregoing shafts are all narrow shafts with 1 to 4 compartments in line. Another type of shaft which is used at some iron and copper mines is arranged with two skip compartments along one end, a long cageway across the shaft, and another compartment subdivided for ladders, cage counterweight, pipes, and power cables. (See fig. 19.) Such an arrangement results in a shaft with the width more nearly approaching the length. The long compartment permits use of a long cage upon which timbers, drill steel, and other supplies can be run on trucks without rehandling, thus saving much time and labor. This type of shaft will doubtless find favor at lead and zinc mines in the future, especially if considerable timber is required for the support of mine workings.

Shaft-sinking costs.—Costs of shaft sinking vary widly with location, size, depth, nature of rock, and character of support required; amount of water encountered; cost of power, labor, and supplies; class of labor; and skill of management.

Some typical shaft-sinking costs at lead and zinc mines are given in table 5, with costs of a few shafts at other mines.

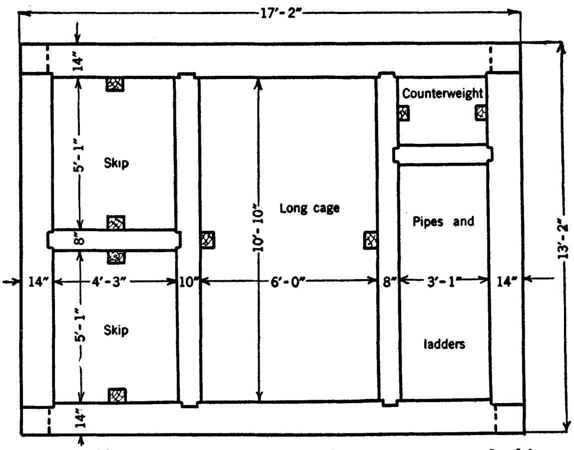

FIGURE 19.—Typical shaft plan for long cage and skips.

TABLE 5.—*Typical shaft-sinking costs*

Mine and location	Shaft section, feet	Kind of rock	Depth sunk, feet	Cost per foot	Remarks
Lead and zinc mines:					
No. 2 mine, Tri-State district.[1]	5 by 7; 1 compartment.	Surface, shale, dolomite, and flint beds.	260	$20.08	Total cost to company.
Hartley-Grantham mine, Tri-State district.[2]	6 by 6	do	200	7–10.50	Contract labor and explosives only; $7 in shale, $10.50 in hard rock.
				18.00	Total cost to company.
Pim shaft, No. 8 mine, Southeastern Missouri district.[3]	6 by 18 in clear; 4 compartments.		730	76.17–92.56	Shaft, concrete; lined to 212 feet. $76.17, sinking; $92.56, sinking and concreting.
Pecos mine, New Mexico.[4]	6.67 by 20.33 outside of timbers.	Metamorphosed igneous rocks.	1,050	119.01	Cost per foot of last 328 feet.
Other mines:					
Ajax mine, Colorado.[5]	6 by 15; 3 compartments.	Granite	(1,481–1,983) 502	57.28	Labor $25.77, materials $16.93, service charges $14.58.
Sylvanite mine, Ontario.[6]	6 by 16; 3 compartments.	Syenite porphyry	963	71.09	Sunk from 1,000 level.
Teck-Hughes, mine, Ontario.[7]	6.33 by 21.33	Porphyry and lamprophrye.	1,484	96.34	
Do	12.5 by 13.0	Porphyry	2,284	120.75	

[1] Netzeband, William F., Method and Cost of Mining Zinc and Lead at No. 2 Mine, Tri-State District Picher, Okla.: Inf. Circ. 6121, Bureau of Mines, 1929, 11 pp.

[2] Keener, Oliver W., Methods and Costs of Mining at Hartley-Grantham Mine, Tri-State Zinc and Lead District: Inf. Circ. 6286, Bureau of Mines, 1930, 8 pp.

[3] Poston, Roy H., Sinking Practice and Costs at the Pim Shaft, St. Louis Smelting & Refining Works of the National Lead Co., St. Francois, Mo.: Inf. Circ. 6588, Bureau of Mines, 1933, 13 pp.

[4] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Inf. Circ. 6368, Bureau of Mines, 1930, 21 pp.

[5] Black, W. S., Cost of Shaft Sinking at Cripple Creek: Eng. and Min. Jour., vol. 120, Aug. 15, 1925, p. 255.

[6] Jackson, Charles F., and Knaebel, John B., Gold Mining and Milling in the United States and Canada, Current Practices and Costs: Bull. 363, Bureau of Mines, 1932, 151 pp.

[7] Henry, R. J., Mining Methods and Costs at the Teck-Hughes Gold Mines (Ltd.), Kirkland Lake, Ontario: Inf. Circ. 6322, Bureau of Mines, 1930, 11 pp.

TABLE 5.—*Typical shaft-sinking costs*—Continued

Mine and location	Shaft section, feet	Kind of rock	Depth sunk, feet	Cost per foot	Remarks
No. 7 shaft, Magma, Ariz.[8]	7.5 by 16.5	Schist	1,465	75.24	Total cost; direct cost, $70.00.
No. 5 shaft, Magma, Ariz.[9]	7 by 21.5; 4 compartments.	do	2,531	116.44	Total cost; direct sinking cost, $76.75.
McPherson shaft, Tennessee.[10]	10 by 19	Schist and graywacke	200	65.34	1,600 to 1,800 levels.
Bisbee Queen mine, Ariz.[8]	7 by 17		823	69.15	

[8] Gardner, E. D., and Johnson, J. Fred, Shaft-Sinking Practices and Costs: Bull. 357, Bureau of Mines, 1932, 104 pp.
[9] Snow, Fred W., Mining Methods and Costs at the Magma Mine: Inf. Circ. 6168, Bureau of Mines, 1929, 32 pp.
[10] Weaver, Lamar, Shaft Sinking in Tennessee: Eng. and Min. Jour., vol. 129, Mar. 24, 1930, p. 302.

A discussion of shaft-sinking practice and costs is outside the scope of this paper; for such information the reader is referred to an earlier Bureau of Mines publication.[39]

The usual three-compartment shaft, 6 by 15 to 6 by 17 feet in cross-section, with 8- by 8-inch timbers, can be sunk for $65 to $75 per foot, if no unusual difficulties such as heavy caving ground or considerable volumes of water are experienced and first cost of equipment is excluded. If the shaft is concreted the added cost will be considerable, varying with the nature of the ground, difficulties encountered, thickness of concrete, etc. The costs of shaft stations and skip pockets are additional items.

LEVEL DEVELOPMENT

Level development comprises tunneling, crosscutting, and drifting. In level development the principal points to be considered are level interval, plan of levels, and size of drifts and crosscuts.

Level interval.—The selection of a level interval is not always given the consideration it deserves. In some very irregular deposits, where a number of disconnected ore bodies occur at different vertical horizons, the level interval may be determined by the elevation of these horizons, and the adoption of a regular interval may not be feasible. However, in other instances the levels may be spaced at regular intervals; in choosing the interval a balance must be struck between a number of factors, including first cost of development, maintenance costs, rate of production desired, regularity of the deposit and level interval best suited to exploration, and stoping method to be employed. These in turn are governed by the type of deposit; the size, shape, dip, and regularity of the ore shoots; and the hardness and strength of the ore and wall rocks.

Usually, the shorter the level interval the greater the first cost of development, since more levels will be required for development to the same depth. On the other hand, if the deposit is very irregular in outline and angle of dip a short interval may be required to

[39] Gardner, E. D., and Johnson, J. Fred, Shaft-Sinking Practices and Costs: Bull. 357, Bureau of Mines, 1932, 104 pp.

explore it thoroughly. Where extensions of the ore body below existing levels can be predicted with some certainty, owing to geological conditions or information obtained by drill exploration, the higher level interval commends itself from the standpoint of first cost and maintenance cost. If long crosscuts from the shaft are required to reach the ore on the various levels the argument for a high level interval is strengthened. Under such conditions it may be desirable, even if a short interval is required for exploration and stope-development purposes, to use a high interval between the main haulage levels, which are connected to the shaft, and to split this interval with sublevels, not connected to the shaft, but driven from raises in the ore body. This scheme can often be employed to reduce the amount of crosscutting required to connect the workings to the shaft. The ore mined on the sublevels and in stopes tributary thereto is then transferred to the haulage levels through ore passes.

Table 6 shows the level intervals adopted at a number of lead and zinc mines in the United States.

TABLE 6—*Level intervals at lead and zinc mines*

Mine and location	Dip of ore bodies	Method of development	Mining method	Level interval, feet	Remarks
No. 1 mine, Tri-State district	Horizontal	Shafts	Open stopes	70	Only 270-foot level connects to shaft; 200 level is a sublevel for mining disconnected upper horizon.
Nos. 2 and 3 mines, Tri-State district.	do	do	do	Only 1 level	
Waco mine, Tri-State district.	do	do	do	Levels at 200 and 300 feet depth.	Levels driven to match 2 separate horizons of ore.
Barr mine, Tri-State district	do	do	do	Single haulage level	Inclines driven to reach higher horizons and lower local "runs" of ore.
No, 8 mine, Southeastern Missouri district.	Flat	do	do	Single main level	Lower portions of ore body reached by flat slope.
Bonne-Terre mine, Southeastern Missouri district.	do	do	do	70 between 2 main haulage levels.	8 other sublevels connected by raises or inclined shafts; levels driven to match separate ore horizons.
Mascot No. 2 mine, Tennessee.	About 20°	Vertical shaft and inclines.	do	50–75	Level interval from incline shafts.
Edwards district, New York		Shafts	Open stopes; some shrinkage.	200 usually; some levels 100	
Bunker Hill mine, Idaho	40°–50°	Inclined shafts from tunnel.	Square-sets	200	Refers to lower levels.
Page mine, Idaho	40°–60°	Inclined shaft	do	300 (incline distance)	
Hecla mine, Idaho	70°–80°	Shaft; adits for upper levels.	Stull-sets and fill	500 and 400 main level interval.	1,400 level between 1,200 and 1,600 not connected to shaft.
Morning mine, Idaho	80° to vertical	Tunnel and underground shafts.	do	200	
Silver King Coalition mines, Utah.	Average dip, 20°	Shafts and 2 drainage tunnels.	Square-sets	100 or 200	
Park Utah mine, Utah	40°–55°	Tunnel and 3 underground shafts.	Square-sets and cut-and-fill.	Upper levels 200; lower levels 300.	
Tintic Standard mine, Utah	Varies from 20° to quite steep.	Shaft	Square-sets	100	
Black Rock mine, Montana	80°	do	do	100 for first 1,900 feet; 150 for balance to 3,000 feet.	
Block P mine, Montana	65°–88°	Adits and shaft	Cut-and-fill	100 feet to 400-foot level; 200 below 400-foot level.	
Pecos mine, New Mexico	Nearly vertical	do	Cut-and-fill and square-sets.	100	
Ground Hog mine, New Mexico.	50°	Shaft and inclined winze.	Square-sets	100	
Battle Mountain district, Colorado.	11° for blanket ore bodies.	Tunnel, shafts, and inclined winzes.	Cut-and-fill and square-sets.	87 vertical; 330 on 15° slope of incline.	
Franklin district, New Jersey.	Pitch, 12° to 26°; dip, 65°.	Inclined shaft	Stopes by shrinkage or cut-and-fill; pillars by top-slicing.	Main levels at 300, 750, 800, 950, 1,050, and 1,150.	Sublevels at 50-foot intervals not connected with shaft.

A short level interval has the following advantages: (1) More thorough exploration of the deposit before ore is extracted, important in some irregular deposits; (2) shorter connecting raises between levels, the per-foot cost of raising increasing rapidly for raises more than about 100 feet high; (3) easier handling of supplies, drills, steel, and timber into the stopes; (4) more points of attack by stoping, permitting a higher production rate (of importance in thin tabular deposits lying at steep angles of dip, particularly where ground conditions require the use of slow stoping methods such as square-setting or cut-and-fill).

Disadvantages of the shorter level interval include: (1) Greater cost of development, due to added footage of drifts and crosscuts with correspondingly more trackage, pipe line, etc., and greater number of shaft stations; (2) greater cost of level maintenance; and (3) more ore tied up in pillars, due to the greater number of drift and floor pillars which have to be left temporarily or permanently.

The stoping method employed may affect selection of the level interval. Thus, if the mining system requires grizzly levels above the haulage levels or if thick floor pillars are required below and arch pillars above the drifts the effective stoping height will be considerably reduced. A high level interval may then be necessary to provide for an economical height of stope.

If the ground is heavy and level maintenance thereby becomes expensive, this may in some cases be an argument for fewer levels and a higher level interval. However, in such ground it may be impossible or very expensive to carry high stopes; and it may be necessary to adopt a short level interval with main extraction drifts in the footwall on each level, connected by short crosscuts to sectional drifts in the vein or by inclined rock raises to the stopes. Haulage drifts in the footwall obviously entail high charges for dead work.

In some broad, thick, flat-dipping deposits, such as those at Mascot, Tenn., which are mined to inclined shafts or slopes closely paralleling the dip, relatively large tonnages can be developed between levels spaced at comparatively short vertical intervals. In the flat ore bodies of the Southeastern Missouri lead district the levels are determined by the horizons at which the ore bodies occur. In the Tri-State district the entire thickness of the ore is usually breasted out from one level, although where ore occurs at 2 or 3 horizons divided by barren horizons separate levels may be driven to match these horizons. Where the upper horizons are mined from sublevels the ore therefrom is transferred through chutes (locally termed "hoppers") to the lower-level workings.

Sizes of drifts and crosscuts.—The size of drifts and crosscuts depends upon such factors as strength of ground, timber requirements for support of the ground, width of lode, type of haulage to be used, size of cars, type of loading chutes or platforms, economical size from the standpoint of cost of driving, stoping method, amount of water on the levels which will affect the size of drainage ditches, and ventilation requirements.

If the ground stands well without timbering the rock dimensions of the headings do not need to be so large to allow clearance for cars and locomotives and room for pipe lines and ditches as where timbering is required. Some ground will stand well if the headings are narrow but not if they are wide. Arching the back often assists in its support.

If the lode dips at a high angle and is not much wider than normal drift width it may be advisable to drive the drift the full width of the lode, thus obviating the necessity of slabbing it later.

If hand tramming is to be used the cars will be small, and usually smaller drifts can be employed than where motor haulage is contemplated. Higher drifts will be required for trolley than for storage-battery locomotives to permit carrying trolley wires at a safe height above the rail. Obviously large cars will require larger headings than small cars. At ore-loading chutes or platforms the drifts will have to be large enough to provide ample room for safe and easy loading. The size of opening required will vary with the type of chute employed; thus, the Treadwell finger chute requires considerable width and headroom, while the curtain or ball-and-chain type requires a minimum of room to accomplish the same results.[40]

If there is much water on the levels a wide drift is usually necessary to provide room for a good ditch on the side of the heading.

The cost of drifting and crosscutting varies with ease of drilling and the breaking characteristics of the ground, the size of opening, the speed of driving, and the methods employed for drilling and mucking. The drilling and breaking qualities of the ground vary with the direction of banding, jointing, or schistosity, as well as with the hardness and toughness of the ground. If the headings have to be timbered the costs will be considerably higher. The larger the heading the greater the mucking cost, particularly if hand-shoveling is employed. Up to a certain size the cost of drilling and breaking often decreases with increase in the size of the heading, as a better and longer break will be obtained. Beyond this critical size the cost per foot will increase, because more holes and more explosive will be required per foot of advance, as well as more mucking. On the other hand, the cost per ton broken will generally be less in the larger heading. Where mechanical mucking is employed the headings must be large enough to permit efficient use of the equipment, and the extra tonnage handled in a large drift adds little to the mucking cost.

Where fast driving is demanded low costs usually must be sacrificed to speed, although up to a certain rate of advance costs may be reduced by faster driving.

The stoping method will influence the general plan of the levels more than will the size of drifts, but it may also affect the latter. Thus, if shrinkage or cut-and-fill stoping is to be done directly on drift timbers, it may be more desirable to drive the original drifts high than to return and take down backs before stoping.

This bulletin will not discuss drifting and crosscutting practices in detail. For such data the reader is referred to Bureau of Mines

[40] Jackson, Chas. F., and Knaebel, John B., Underground Chute Gates in Metal Mines: Inf. Circ. 6495, Bureau of Mines, 1931, 22 pp.

Bulletin 311, Drilling and Blasting in Metal-Mine Drifts and Crosscuts, by E. D. Gardner.

Table 7 shows the size of drifts and crosscuts employed at certain lead and zinc mines in the United States and a few figures on the cost of drifting and crosscutting.

Development prior to ore production will cost considerably more per foot than similar work during productive operation, since all overhead, plant operation, and maintenance will have to be borne by it. The few costs given in table 7 are those at producing mines and are, on the average, low for total costs during a purely development program, except under very favorable conditions.

72670°—35——7

TABLE 7.—*Drifting and crosscutting data at lead and zinc mines*

Mine and location	Type of ore and rock	Size of headings, feet	Mining method	Type of haulage	Cost per foot	Remarks
Mine No. 1, Tri-State district.	Dolomite and chert.	7 by 7 (crosscuts between ore bodies termed "pull drifts").	Open stopes	Hand, mules, and trolley locomotives.	$7.50 labor; $14.13 total	Untimbered.
Mine No. 2, Tri-State district.	do	do	do	Hand and mules	$6.50 labor only	Do.
Waco mine, Tri-State district.	do	8 by 8	do	do		Do.
Barr mine, Tri-State district.	do	7 by 7 to 8 by 10	do	do	$6.00 labor; $10.00–$12.00 total.	Do.
No. 8 mine, Southeastern Missouri district.	Dolomitic limestone.	7 by 8	do	Trolley locomotives	$4.89 labor; $7.87 total	Do.
Bonne Terre mine, Southeastern Missouri district.	do	7½ by 8 to 9 by 10; main haulageways 8 by 12 minimum.	do	Trolley locomotives with mules for gathering.		Do.
Mascot No. 2 mine, Tennessee.	do	8 by 8 (main haulage); 7 by 6 (crosscuts).	do	Trolley locomotives	$8.17 total	Average cost of drifts, crosscuts, raises, and slabbing.
Edwards district, New York.	Dolomite and some gneiss.	6 by 7	Open stopes with some shrinkage.	Storage-battery locomotives.		Usually untimbered.
Bunker Hill mine, Idaho	Quartzite and ore	7 by 8 (drifts); 15 by 8 (main crosscuts).	Square-sets	Storage-battery locomotives on levels; trolley locomotives in tunnel.	$11.35	Average cost of 1,168 feet in 1931.
Page mine, Idaho	do	6 by 8 inside timber; 8 by 9.5 rock dimension.	do	Storage-battery locomotives.		Drifts timbered.
Hecla mine, Idaho	do	8 by 8 clear for main haulage; 5 by 7 for crosscuts; 13 feet high in ore and full width of lode up to 18 feet.	Stull-sets-and-fill	do		
Morning mine, Idaho	do	13 by 13 (drifts); 6 by 8 (crosscuts)	do	do		Drifts all timbered.
Silver King Coalition mines, Utah.	Quartzite, limestone, and ore.	6 by 8 in clear	Square-sets	Trolley locomotives on main levels.	$11.35 average for drifting, crosscutting, sinking, and raising.	Some drifts timbered.
Park Utah mine, Utah	do	8 by 8 (main haulage); 5 by 7 (exploration crosscuts).	Square-sets and cut-and-fill.	Trolley locomotives	$23.84; main tunnel 8 by 8, one third timbered.	Drifts usually timbered.
Tintic Standard mine, Utah.	Limestone, quartzite, shale, and ore.	6 by 8 over all; 4.33 by 7.25 in clear	Square-sets	Hand; mules for longer trams.		Timbered.
Black Rock mine, Montana.	Granite and vein matter.	6 by 8 to 8 by 10 over all; 3.66 by 7.25 to 5 by 7 in clear.	do	Storage-battery locomotives.		Drifts timbered.
Block P mine, Montana	Rhyolite and vein matter.	6 by 8 over all; 3.5 by 6.5 in clear	Cut-and-fill	do		Do.
Pecos mine, New Mexico	Schist and ore	7 by 8 (crosscuts); drifts 5.33 at top and 5.8 at bottom by 7 high in clear.	Cut-and-fill and square-sets.	do	$8.08 labor; $11.30 total	Do.
Ground Hog mine, New Mexico.	Limestone, shale, sandstone, and ore.	5 by 7	Square-sets	Hand	$3.25 to $4.50 per foot for contract labor only.	Drifts not timbered.

Plan of levels.—The level plan or level lay-out naturally will depend largely upon the size and shape of the deposit. The strength of ore and wall rocks, the mining method employed, the rate of production desired, and the haulage system are other determining factors.

In wide, horizontal, bedded deposits with firm ore and capping, where the levels are breasted out from the shaft the full height and width of the ore in the form of stopes, the level plan will correspond to the outline of the ore. Figure 5, page 21, shows wide, irregular ore bodies connected by "pull drifts" and short, flat inclines. Tracks are laid as stoping advances; main haulageways are carried behind the advancing faces, with branches ahead to the different stope

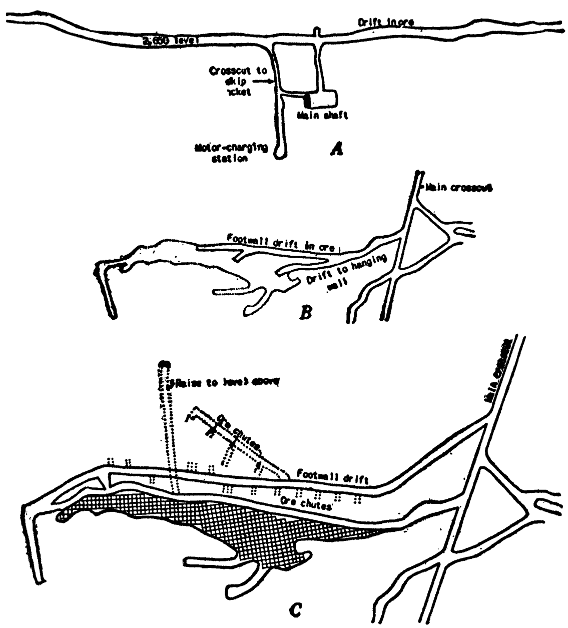

FIGURE 20.—*A*, Plan of level, Morning mine, Idaho; *B*, partly developed level in wide ore body, Bunker Hill & Sullivan mine; *C*, complete development of sill floor, showing ore chutes for higher floors, Bunker Hill & Sullivan mine.

faces. The ore is shoveled into cars from the floor of the stope by hand, by power shovel, or by scraper.

In wide, thick ore bodies where the ore is stoped over the level and drawn off by gravity through chutes into the mine cars, one or more parallel drifts may be driven on the strike of the ore. Where the ore is not too thick and dips at a steep angle a crosscut from the shaft to the vein with a single drift on the vein may be all that is required for development of a level. This is illustrated by figure 20, *A*, which shows the plan of a level at the Morning mine, Idaho.

Where the ore is thicker or dips at a flatter angle two or more parallel drifts may be required to extract the full width of the ore, if only gravity methods for transferring the broken ore from the stopes to the chutes are used. Thus, figure 20, *B*, shows a partly developed level at the Bunker Hill & Sullivan mine, Idaho; one drift is carried in ore along the footwall and another along the hanging-wall side of the ore body. Figure 20, *C*, shows complete development of the level or sill floor shown in figure 20, *B*, a footwall drift having been driven from which ore chutes are put up to cut the stopes at higher elevations.

In some wide ore bodies drifts are driven parallel to the ore in both foot and hanging walls; these are connected by transverse drifts through the ore at regular intervals. This arrangement permits use of a loop haulage system by which the empty trains go in one side to the transverse drifts where they are loaded, and the loaded trains go out the other side and back to the shaft.

In mines where it is difficult or expensive to maintain drifts in the lode or vein, it is often desirable to run the main haulage drift in the footwall with crosscuts therefrom to the vein at regular intervals for extraction of ore.

At the Park Utah mine the drifts are advanced along the footwall; and, where the ground is heavy and the vein is exceptionally wide, drifts are sometimes run both in the foot and hanging walls. From the footwall drift raises are run at 50-foot intervals to the stopes above. Where close filling of the stopes is necessary and the dip of the vein is not steep, filling can be introduced into the stopes below to better advantage from a hanging-wall drift than from a footwall drift, not only because waste can be run in to fill the upper part of the stope by gravity but also because ore from the stopes above and filling to the stopes below can be handled in separate drifts at the same time without one operation interfering with the other. The ore is drawn from chutes in a drift on the vein or in the footwall, while waste is trammed through the hanging-wall drift.

Where there are a number of parallel veins or lenses of ore the level pattern will consist of one or more crosscuts with drifts on each vein. Where there are scattered, irregular ore bodies oriented in various directions the level pattern obviously will be irregular with crosscuts between the ore bodies or from a main haulage crosscut to each ore body.

STOPE DEVELOPMENT

The foregoing brief discussion relates to general development and covers the work required for reaching the ore bodies and developing the levels. Before actual stoping can be begun most mining methods require further preliminary work, which is termed "stope development" or "stope preparation"; it consists principally of raising and installation of chutes and with some mining methods requires the driving of sublevels.

Many mining companies combine stope development with general development on their cost sheets, making no distinction between these two classes of work. Others combine stope-preparation costs with stoping costs, particularly if the former cover merely an occasional raise. Where stope preparation constitutes an important item of cost it would seem logical to differentiate this work on the cost sheets, as some companies do.

In the Tri-State district, where most development consists of actual breasting out of the stopes and where only occasional raises are run, no stope preparation is required and very little development once the shaft is down. Much the same situation exists in the Southeastern Missouri lead district.

At Mascot, Tenn., some stope preparation is required; raises and chutes are started from the extraction drifts to and through the ore body with short raises or connecting drifts for entrance to the stope after stoping around the starting raises has commenced.

At Edwards, N.Y., chute raises are driven from the haulage drifts on 30- to 40-foot centers for 15 to 30 feet up the dip. A subdrift is then driven connecting the tops of these raises; the floor of this drift forms the bottom of the stope. Where underhand stoping is to be employed a raise must be driven from one level to the next, or to the top of the ore, before stoping can be started.

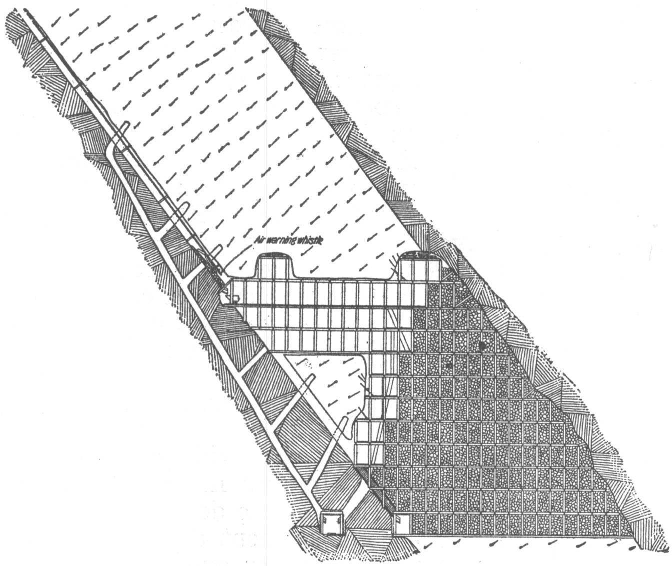

FIGURE 21.—Cross-section showing stope development by footwall raise, Bunker Hill & Sullivan mine.

At the Bunker Hill & Sullivan mine, where square-set stoping is employed, a raise is driven on the footwall (see fig. 21), and later another is driven in the footwall itself with branch raises as shown in the figure. This work, however, can be done concurrently with stoping, which starts on the hanging-wall side.

At the Morning mine ventilation raises are driven in either the hanging or the foot wall (usually the latter) to connect each level with that above.

At the Hecla mine stope preparation involves putting up raises in ore from level to level at 100-foot intervals along the strike of the vein. These raises contain 3 compartments, for chute, manway, and timber slide, respectively; the raises are timbered with stull sets. Sometimes there are 4 compartments, 2 being used for chutes.

At the Park Utah mine raises are usually run from level to level in ore at 100-foot intervals for introduction of filling. In heavy ground, where the extraction drifts are in the footwall, short rock raises are driven from the drift to the vein on 50-foot centers for use as ore passes and manways.

At the Tintic Standard mine at least one two-compartment raise for each stope is first driven in ore to the next level to serve as a starting raise for the stope and for introduction of filling material. Later, it may be necessary to drive other raises for fill or to raise from footwall drifts through waste to the upper part of the stope to provide extraction chutes.

At the Block P mine, Montana, where cut-and-fill stoping is employed, through raises are driven at 690-foot intervals along the vein. Chutes at 46-foot intervals are placed between drift sets, and stoping starts directly above the drift timber without further preparation.

At the Pecos mine, New Mexico, both square-set and cut-and-fill stoping are employed. In preparation for cut-and-fill stoping, two-compartment raises are driven in ore 12 feet over the back of the drift, where a stope is silled out, leaving a solid pillar of ore 12 feet thick between the bottom of the stope and the back of the drift. These raises are 30 feet apart along the drift. Fill raises are run from the stope to the level above at 30- to 60-foot intervals.

At the Ground Hog mine, New Mexico, main levels 100 feet apart vertically, or 135 to 140 feet on the dip, are connected by raises at 50-foot intervals along the vein; in addition, these raises are connected by sublevel drifts halfway between the main levels.

COST OF MINE DEVELOPMENT

Obviously, development costs will vary widely at different mines, depending upon the size, shape, dip, and interval between various ore bodies and other characteristics of the deposits, as well as upon mining methods, efficiency of management and labor, engineering skill, and cost of labor, materials, supplies, and power.

The tonnage of ore developed per foot of development serves as a rough measure of the development cost at a given property. Unfortunately, few mines appear to keep records on this basis. Table 8, however, shows this relationship approximately for a few lead and zinc mines. In some instances the figures represent current development for a short period only and exclude such major development as shafts and some of the main crosscuts and drifts therefrom.

Table 9 gives costs of current mine development at a number of lead and zinc mines per ton of ore mined during specific periods of operation. The figures cannot be considered as comparable, since an abnormal amount of development work may have been done at some mines compared with the tonnage mined, while at others less than a normal amount may have been done during the period covered by the cost figures; however, these figures show how development charges may vary at different mines operating in deposits of different types and under regular operating conditions.

At some mines the figures may include general development only and may exclude stope development or stope preparation, while at others they may include both general development and stope preparation.

TABLE 8.—*Tons of ore produced per foot of development work at lead and zinc mines*

Mine or district	Type of ore body	Level interval, feet	Mining method	Production per foot of development, tons
Tri-State district [1]	Flat, irregular ore bodies	Usually only 1 level.	Open stopes (heading and bench).	250
Do.[2]	do	do	do	600
No. 8 mine, Southeastern Missouri district.[3]	Flat, irregular bedded deposits.	1 level	do	250+
Mascot No. 2 mine, Tennessee.	Large, irregular bedded deposits; dip, 18° to 22°.	50 to 75	Open stopes, underhand mill hole, and heading and bench.	155
Edwards mine, New York.	Irregular bedded deposits; dip, 40° to 45°.	Upper levels, 100; lower levels, 200.	Open stopes with some shrinkage stoping.	27.5 (1930)
Bunker Hill mine, Idaho.	Wide, deep, shear zone; large ore bodies.	200	Square-sets-and-fill	125
Page mine, Idaho	2 fissure veins in shear zone in quartzite; dip, 40° to 60°.	300	do	19
Morning mine, Idaho	Long fissure vein in quartzite; continuous to great depth; dip, 80° to vertical.	200	Stringer-sets-and-fill	66
Silver King Coalition mines, Utah.	Large irregular bedded deposits in limestone; some fissures in quartzite.	200 at lower levels.	Square-sets-and-fill	5.4
Park Utah mine, Utah.	Fissures in quartzite and limestone; ore shoots, 3 to 80 feet wide up to 900 feet long; dip, 40° to 55°.	200	Square-sets and cut-and-fill.	15.7
Pecos mine, New Mexico (1928–30).	Irregular lenses in shear zone in schist.	100	Cut-and-fill and square-sets; some shrinkage.	14.7
Ground Hog mine, New Mexico.	Ore body 3 to 25 feet wide by 600 feet long in fault fissure; dip, 50°.	100	Square-sets-and-fill	13.2

[1] Average for 3 mines; all development included.
[2] Average for 2 large mines; all development included.
[3] Estimated.

TABLE 9.—*Current mine development costs per ton of ore mined*

Mine and period covered	Type or ore body	Level interval, feet	Mining method	Ore mined during period, tons	Cost of development per ton of ore mined
No. 8 mine, Southeastern Missouri district, 1928.	Irregular, flat bedded deposits.	1 level	Open stopes (heading and bench).	168,089	$0.045
Mascot No. 2 mine, Tennessee, 1929.	Large, irregular bedded deposits; dip, 18° to 22°.	50 to 75	Open stopes, underhand mill hole, and heading and bench.	528,626	.053
Page mine, Idaho, 1928.	2 fissure veins in shear zone in quartzite; dip 40° to 60°.	300	do	90,460	1.170
Morning mine, Idaho, 1928.	Large fissure vein in quartzite, continuous to great depth.	200	Stringer-sets-and-fill	276,890	.442
Hecla and Star mines, Idaho, 1928.	Several ore bodies in shear zone dipping 70° to 80°.	300 and 400	do	312,942	.638
Silver King Coalition mines, Utah, 1929.	Large, irregular bedded deposits in limestone; some veins in quartzite.	200 at lower levels.	Square-sets-and-fill	180,208	[1] 2.086
Park Utah mine, Utah, 1928.	Fissures in quartzite and limestone; ore shoots 3 to 80 feet wide up to 900 feet long; dip 40° to 55°.	200	Square-sets and cut-and-fill.	192,250	[1] .828
Tintic Standard mine, Utah, November 1929.	Trough ore bodies replacing limestone beds; extend over wide areas and large vertical height.	Usually 100 (200 feet, level 900 to 1,100; 150 feet, level 1,100 to 1,250).	Square-sets-and-fill	8,758	[2] 1.143
Pecos mine, New Mexico, 1927–29.	Irregular lenses in shear zone in schist.	100	Square-sets-and-fill and cut-and-fill; some shrinkage.	578,658	[3] .859
Ground Hog mine, New Mexico, 3 months, 1930.	Ore body 3 to 25 feet wide by 600 feet long in fault fissure; dip 50°.	100	Square-sets-and-fill	11,146	[4] 1.393

[1] Development in rock and ore.
[2] $0.299 in ore and $0.844 in rock.
[3] All development, including shaft sinking, stations, and pockets.
[4] $0.922 in ore and $0.471 in rock.

STOPING

The beginning of stoping operations marks the opening of the active productive life of a mine. Considerable ore may have been produced, however, during exploration and development.

At most mines development continues long after stoping begins, and lasts almost to the end of the life of the mine. Likewise, stoping may begin very early in the development period. In the Tri-State district, as previously pointed out, the usual development consists merely of sinking a mill shaft and one or more field shafts to the bottom of the ore; stoping from the shaft then begins almost immediately. The deposits usually have been at least partly proved by churn-drilling; and enough tonnage has been shown to warrant going ahead with sinking, installing mine equipment, and building a milling plant. The ore is then developed as it is stoped.

In districts where the ore is of high grade but insufficient tonnage has been proved to warrant erection of a mill high-grade mine-run ore, or sorted ore of high tenor, may be shipped direct to smelters, and stoping may begin early in the life of the property.

In this event the income from the sale of stope ore and ore removed in development may assist materially in defraying the cost of further exploration and development and may even show a profit over the cost of such work.

Where enough capital is available good judgment usually dictates that, during the early life of a mine, development should be carried on at a rate which will block out ore faster than it is removed by stoping, thus providing a substantial ore reserve. This not only acts as a backlog assuring continued and even production but also forms a basis for planning the most efficient operation and the capital expenditures warranted; moreover, it permits taking advantage of favorable market conditions by stepping up the rate of production. On the other hand, development too far ahead of ore extraction by stoping absorbs capital and interest charges thereon. The characteristics of the ore deposit (especially its size and the regularity of ore occurrence), the cost of development and operation, the financial condition of the company, and the market conditions should be considered in determining the proper rate of development. If the ore is stoped as fast as developed the company will always be in a precarious position, and, unless deposits are very regular or large, production will usually fluctuate so as to preclude the economies in operation possible with an even rate.

In large deposits of low-grade ore a high production rate is usually necessary for profitable operation, a milling plant is required, and development must be kept far enough ahead of mining to permit extraction of ore by the most economical methods and at a rate that will supply the mill with a steady feed.

In this connection the method of stoping to be used is important. Thus, where a comparatively slow method of stoping, such as square-setting or cut-and-fill stoping, must be employed, development must provide a greater number of active stopes at all times than where more rapid stoping methods are employed, the size of the individual stopes remaining the same.

To summarize, an appreciable reserve of ore blocked out in advance of stoping operations is a decided advantage for attaining the most economical operation, as this reserve will permit not only advance planning of methods and details of ore extraction, with attendant savings in operating costs, but also proper balancing of month-to-month development with stope production and of rate of output with mill capacity and capital expenditures.

STOPING METHODS

Selection of stoping method.—The stoping method at a property is usually evolved from experience in mining the deposits, at least as to variations of the method employed and details of practice. Low stoping costs obviously should be the aim in selecting the method, although, as will be pointed out later, the method resulting in the lowest stoping cost per ton of ore may not give the lowest over-all cost per pound of lead and zinc.

With some mining methods costs are much lower than with others. Unfortunately, the operator's problem is not the simple one of adopting the cheapest method but is rather the selection of a method that will best fit the natural physical conditions of the deposit.

At some mines there may be little, if any, choice in the general method of stoping to be used, owing to the characteristics of the ore bodies and the wall rocks, and it may be necessary to employ high-cost methods. At other mines conditions at once make it apparent that low-cost systems can be applied. At still other mines there may be a choice between two or more general methods or between variations of one system, and good judgment based upon thorough knowledge of ground support and of mining technique is required to select one that will give the best results and lowest cost.

In the selection of a stoping method the question of support to prevent failure, caving, and subsidence, particularly during the working of the stope, is fundamental, although other factors may be important. Permanent support may be an important secondary consideration and may indeed be the determining factor between selection of a method wherein subsidence and breaking through to the surface will occur and one wherein the surface can be kept intact. In other words, the degree with which the ore and wall rocks will stand in and around excavations is the determining factor. Thus, the length or area of unsupported arch that will stand without failing, the length of time it will remain intact, the nature of support required, the spacing of supports needed, the direction of pressures acting on the mine workings, the materials available for supplying temporary and permanent support and their cost, and the requirements for providing permanent support against surface subsidence are important factors. The grade of the ore may also be important, since the value of the ore left in supporting pillars may be large if the ore is high-grade, whereas the loss of ore in such pillars may be less than the cost of supplying artificial support plus the cost of mining the pillars if the ore is low-grade. With high-grade ore a wasteful method of mining by which considerable ore may be lost would not usually be applied, while with low-grade ore such a method might be justified on account of its low cost.

The following classification of stoping methods was adopted by the Bureau of Mines in 1928 for use in connection with a comprehensive study of mining methods and costs inaugurated in that year. This classification is based upon the method of stope support employed during the period of active stoping operations.

CLASSIFICATION OF STOPING METHODS

Stopes naturally supported:
- 1. Open stopes (removing all the ore).
  - (*a*) Open stopes in small ore bodies.
  - (*b*) Sublevel stoping.
- 2. Open stopes with pillar support.
  - (*a*) Casual pillars.
  - (*b*) Room (or stope) and pillar (regular arrangement).

Stopes artificially supported:
- 3. Shrinkage-stoping.
  - (*a*) With pillars.
  - (*b*) Without pillars.
  - (*c*) With subsequent waste-filling.
- 4. Cut-and-fill stoping.
- 5. Square-set stoping.

Caved stopes:
6. Caving (ore broken by forced caving).
   (a) Block-caving, including caving to main levels and caving to chutes or branched raises.
   (b) Sublevel caving.
7. Top-slicing (mining under a mat which, together with caved capping follows the mining downward in successive stages).

Combined methods:
8. Combinations of supported and caved stopes (such as shrinkage-stoping with pillar-caving, shrinkage-stoping with top-slicing of pillars, etc.).

During 1930 United States lead and zinc mines produced approximately 19,400,000 tons of crude ore, of which about 14,661,000 tons were mined by open-stope mining, 1,896,000 tons by square-set mining, 1,359,000 tons by cut-and-fill mining (including 710,000 tons by stringer sets-and-fill in the Coeur d'Alene district, Idaho), 974,000 tons by shrinkage-stoping, 476,00 tons by top-slicing, and 34,000 tons by block-caving.

OPEN STOPES

In operations using open stopes the ore and wall rocks are self-supporting over and around the excavations. However, natural support afforded by pillars of ore or waste may be employed; and occasional artificial support in the form of stulls, cribs, or packs may be used to hold local blocks of loose or weak ground temporarily. This method is applicable when deposits are of strong, firm ore with strong walls and capping.

The open-stope method is employed exclusively in the Tri-State lead and zinc district, in the Southeastern Missouri lead district, and in the zinc mines of Tennessee and is the principal one in use at Edwards, N.Y.

In the first three districts support in wide stopes is provided by casual or irregular pillars. In the Tri-State and Southeastern Missouri districts the heading-and-bench system is employed. At Mascot, Tenn., most of the ore is recovered by underhand stoping around mill holes, a variation of heading-and-bench mining. At Edwards, N.Y., a more or less regular arrangement of stopes and pillars is employed, especially in the lower levels.

HEADING-AND-BENCH SYSTEM

In the heading-and-bench variation of open-stope mining a low heading is driven under the capping at the top of the ore, and as the heading advances the bench is stoped behind it to the bottom of the ore or to the floor of the level.

Southeastern Missouri and Tri-State districts.—Figures 22 and 23 illustrate methods of stoping by the heading-and-bench system in these districts. The ore is breasted out the full width of the stope as shown, advancing in one direction in narrow ore bodies. In wider ore bodies, branch stopes are driven in the same manner, cutting around the pillars, which are left to support the back. Figure 22, *A*, *B*, and *C*, shows the details of the practice employed in the Southeastern Missouri district, while figure 23, *A* and *B*, illustrates heading-and-bench stoping in the Tri-State district. It will be noted that in southeastern Missouri the bench or stope holes are drilled downward into the bench, whereas in the Tri-State district they are drilled in from the face of the stope or bench at flat angles. Figure 23, *B*, illustrates a method of springing or chambering the stope holes employed occasionally, while figure 23, *A*, shows the more common method of drilling and blasting without springing the holes.

Where ground conditions permit, vertical bench holes drilled by 1-man "plugger" machines are usually cheaper than flat holes. In the Tri-State district, however, the broken, often vuggy, nature of the ground makes drilling of vertical down holes very difficult,

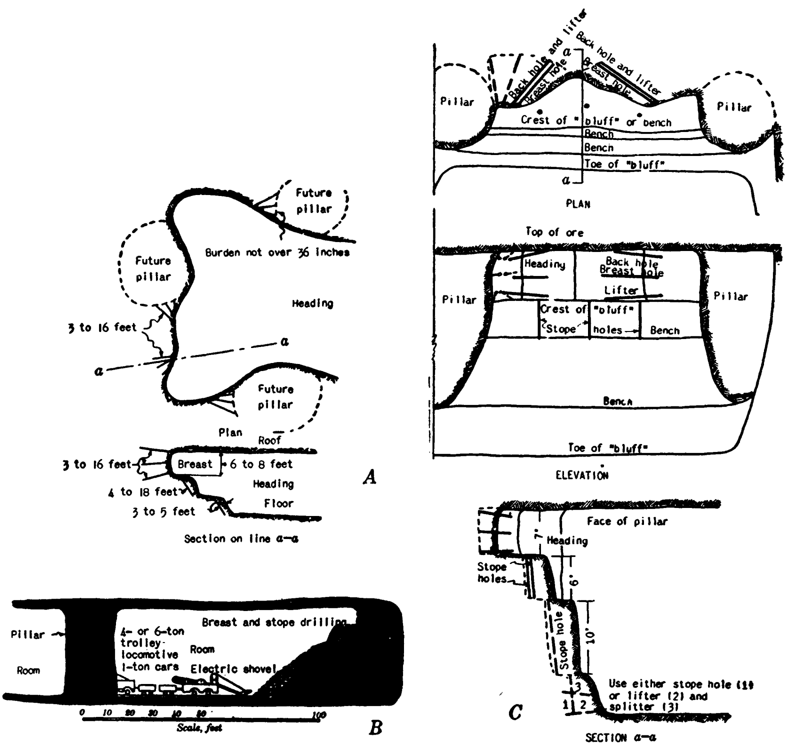

FIGURE 22.—*A*, Method of advancing breast and bench, No. 8 mine, Southeastern Missouri district; *B*, section showing mining with electric shovel in thick ore, No. 8 mine; *C*, method of driving heading and bench stope and turning pillars, Southeastern Missouri district.

and flat holes are much better suited to conditions in this district.

Mascot, Tenn.—Figure 24 illustrates the method of carrying a heading-and-bench stope at Mascot, Tenn.[41] Here, the bench holes are drilled vertically downward. The figures in circles indicate the number of sticks of powder used in each hole, while the other figures indicate the order of firing. Eight-foot headings are cut immediately below the bed that has been chosen for the roof of the stope. In the regular heading-and-bench system the ore is attacked from

[41] Coy, Harley A., Mining Methods and Costs, American Zinc Co. of Tennessee, Mascot, Tenn.: Inf. Circ. 6239, Bureau of Mines, 1930, 11 pp.

one side, the stope being advanced along the ore body. The broken ore is blasted or shoveled off the benches and is hand-shoveled into cars at the toe of the stope.

In the mill-hole system, by which most of the ore is now won, mining spreads out in all directions around a central raise or mill hole. The heading is started under the roof of the ore, and the

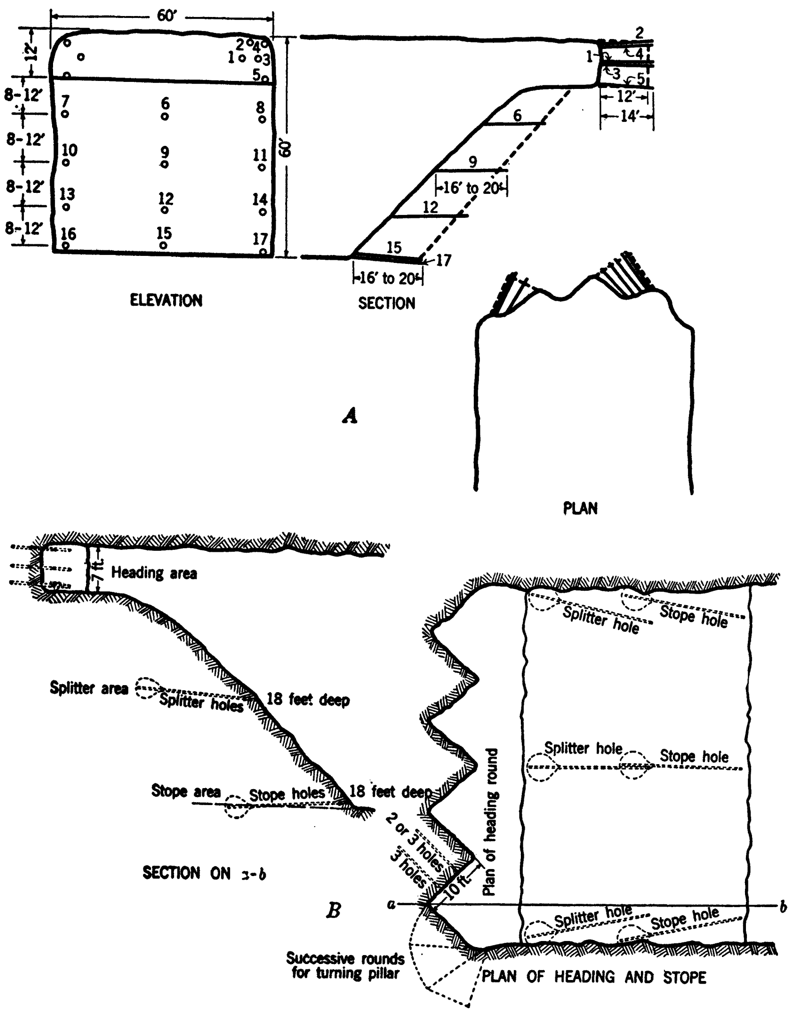

FIGURE 23.—Heading and bench stoping, Tri-State district; *B*, heading and bench stoping, showing use of "sprung" holes.

stope face is then benched down around the mill hole from the heading forming a funnel-shaped stope. (See fig. 25, *A*.) Access to the stope is gained through short drifts or raises driven from nearby stopes or drifts. As long as the sides of the mill hole are steeper than the angle of repose of the broken ore it runs by gravity into the chute at the bottom. When the stope has been widened out so that the sides slope at an angle flatter than the angle of repose, two courses are open for further widening of the stope: (1) Auxiliary raises may be put up into the sides of the stope for drawing off the

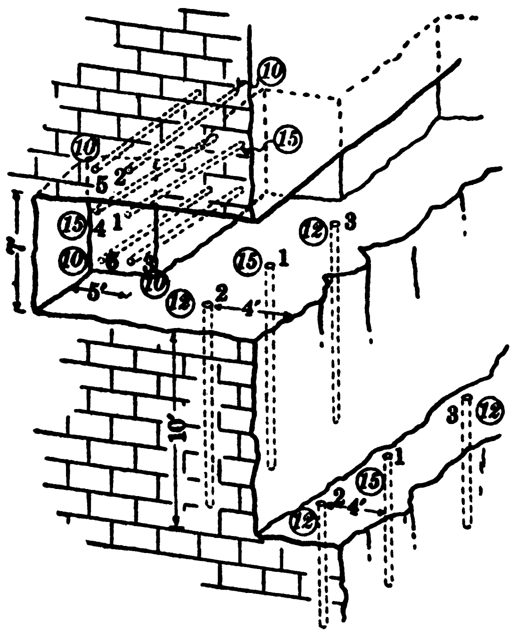

FIGURE 24.—Method of carrying heading and bench stope, Mascot, Tenn.

ore as it is broken, or (2) the ore may be slushed into the original mill hole by power-operated scrapers.

The ore bodies differ in thickness and occupy low angles of dip; as a result the bottom of the ore stands at various heights above the haulage levels. These two factors determine whether the ore shall

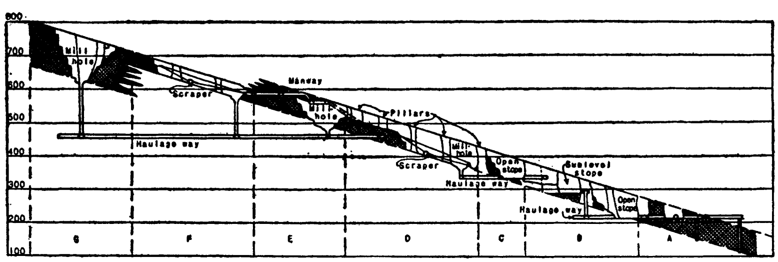

FIGURE 25.—General features of the several variations of open-stope mining at Mascot, Tenn.

be mined by direct mill holing or by slushing. Where the thickness of the ore is sufficient and the development required is not excessive, mill holing is preferred; but where long drifts and high raises are required and the thickness of the ore does not justify so much development, slushing is being used more and more. The first development for slushing is identical with that for mill holing in that a raise is driven to the top of the ore. Stopes are 40 to 60 feet wide with pillars 25 to 30 feet in diameter left at the corners between adjoining stopes.

Figure 25 shows the general features of the several variations of open-stope mining at Mascot. Prospecting at low elevations by crosscutting and diamond drilling is in progress at *A*. *B* illustrates open stopes mined by the regular heading-and-bench system and sublevel stopes for mining out such ore as was left between main-level stopes, a practice followed before the mill-hole and slushing methods

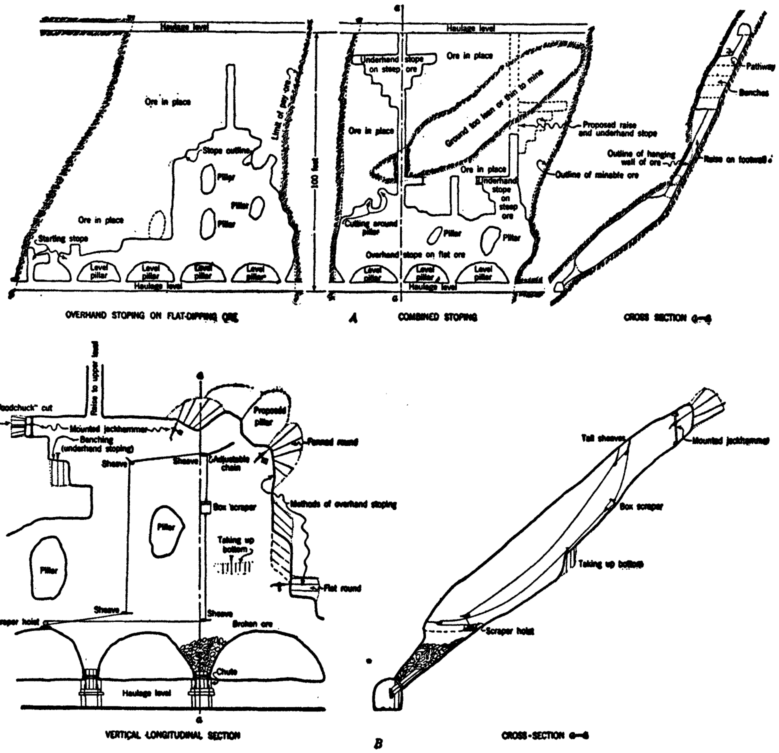

FIGURE 26.—*A*, Open stopes, Edwards mine (generalized); *B*, breaking and handling ore in open stopes, Edwards mine.

were adopted. *C* depicts an open stope from a crosscut through the ore body, mined by heading and bench with hand shoveling. *D* pictures the slushing of such ore as remains along the footwall of the stope after the angle of repose of broken ore has been reached. *E* shows a typical mill hole before slushing equipment is installed, while in *F* slushing is under way in thin ore considerably above the haulage way as a supplement to mill-hole operation. In *G* is shown mill holing in thick ore considerably above the haulageway.

Edwards, N.Y.—Figure 26, *A* and *B*, illustrates the method of open stoping employed at Edwards, N.Y.[42] Here both overhand and

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

underhand stoping are employed, and in somes stopes a combination of the two methods of advance is used, as at the right in figure 26, *A*. The illustrations show the use of pillars over the haulage levels and irregular pillars within the stope itself. Pillars are left where necessary to support the roof, and often it is possible so to arrange them that lean or barren rock may be used for support. In some parts of the mine, especially in the lower levels where the hanging wall is blocky and dangerous, it is planned to drive stopes 15 feet wide up the dip between 40-foot pillars which later will be cut through and slabbed as much as safety will permit.

The method of starting overhand stopes is shown at the extreme left of figure 26, *A*. In overhand stopes the ore usually is first breasted out along the hanging-wall side if it is thick, the footwall ore being benched up later from below. (See fig. 26, *B*, at the right.) Breasts are driven 6 feet high, regardless of the thickness of the ore. The drilling is done chiefly by mounted jack hammers run wet, although stopers may be used (particularly in cutting around pillars and steep veins). A maximum of nine 8-foot holes is drilled per heading or breast. The method of placing holes is shown at the left in figure 26, *B*.

Underhand stopes are employed chiefly for mining the upper portions of blocks, the lower parts of which have been mined by overhand stoping, particularly where the dips are steep and the walls relatively strong. Underhand stoping is often used in combination with overhand stoping, and a stope may be worked at various stages by alternating between the two methods. Underhand stoping is started from a raise or an old stope and continued by simply " milling " the ore into the raise or stope by means of down holes in the benches. Benches are usually about 6 feet wide and at least 7 feet high and are drilled with 3 rows of holes, each carrying a 2-foot burden.

The angle of repose of ore broken in the stopes is 42° to 45°. Where the dip is steeper, gravity moves the ore to the chutes below. Where the dip is flatter mechanically operated scrapers are employed to drag the broken ore to the chutes, as shown in figure 26, *B*. For this purpose hoe-type or semi-box-type scrapers actuated by double-drum hoists are employed. The smaller scrapers are operated by 7½-hp. turbinair hoists and the larger ones by 10-hp. electric hoists. In the future larger-capacity electric hoists will probably be employed. Scrapers are used for hauls up to 300 feet long and are effective in stope service. They have greatly increased output per man and reduced mining costs.

Other mines.—The zinc ores of Wisconsin are mined entirely by open-stope methods. Other lead and zinc mines employing open-stope methods in whole or in part are the following:

Bertha Mineral Co. at Austinville, Va., which produces about half the ore from open stopes and the other half by shrinkage stoping.

Golconda Lead Mines Co., Idaho, and several other small mines in Idaho.

Bristol Silver Mining Co., Pioche, and Pioche No. 1 mine of the Combined Metals Co., in Nevada.

Horn Silver mine of the Tintic Lead Co., Park City Consolidated Mines Co., and some others in Utah.

At Gilman, Colo., the Empire Zinc Co. mine obtains most of its output by cut-and-fill stoping and also employs some square-setting and open-stope mining. Open underhand stopes have been tried as a substitute for horizontal cut-and-fill mining. As the waste-rock back and the top ore are weak, the ore is extracted by breast stoping, a 15- to 20-foot shell of ore being left overhead.[43] The ore in the floor is milled into drawing raises, and the stope is worked down to the bottom waste rock. When completed, the stope is 200 to 250 feet long and about 40 feet high. A stope 20 feet wide was successfully operated by this method. After removal of the broken ore the stope walls were gob-fenced, the stope was filled, and the top shell of ore was taken out by cut-and-fill methods.

SUMMARY OF OPEN-STOPE METHODS

Open-stope mining is applicable to firm ore bodies with strong walls and capping. In the sublevel variation of the open-stope system, used for mining certain iron-ore and copper deposits, the ore and walls need be only moderately secure. As far as the authors are aware, however, this variation is not applied to stoping lead and zinc ores in the United States.

Where open stoping can be applied it has the following advantages and disadvantages:

Advantages:

1. Stoping cost per ton is low considering the fact that the ore is usually hard.
2. Mechanical loading and haulage methods can easily be applied.
3. Irregularities of the ore bodies can be followed and high percentage of extraction obtained.
4. Sorting of waste underground is possible, and lean ore or baren rock within the boundaries of the deposit may be left in place as pillars.
5. Different classes of ore can be mined selectively.
6. The ore can be removed as fast as it is broken so that capital is not tied up in broken ore.
7. Usually most of the development is in ore.

Disadvantages:

1. In some instances considerable ore must be left permanently in the form of pillars for support of the workings.
2. In high stopes it is often difficult to inspect the roof and walls and take down loose slabs of rock.

Open-stope methods and costs have been discussed by Jackson in greater detail in an earlier paper.[44]

SQUARE-SET STOPES

In mines employing square-set stoping the ore is excavated in units consisting of small square or rectangular blocks, and the top and sides of the excavation are immediately supported by framed timbers before an adjoining block of ground is removed. This method ap-

[43] Borcherdt, W. O., The Empire Zinc Co.'s Operation at Gilman, Colo.: Eng. and Min. Jour., vol. 132, Aug. 10, 1931, pp. 99–104.
[44] Jackson, Chas. F., Mining Ore in Open Stopes, Central and Eastern United States: Inf. Circ. 6193, 1929 (revised 1931), Bureau of Mines, 30 pp.

plies where the ore and (or) the wall rocks are so weak that even small openings will not stand unsupported except for a very short period and where cheaper stoping methods cannot be employed because of the high grade of the ore, the necessity of permanently supporting the overlying material against caving and subsidence, the great irregularity of the deposit, or combinations of these conditions.

The individual blocks excavated as separate units are usually 5 to 6 feet square and 6 to 8 feet high. These blocks are often termed " sets " of ground, although the term " sets " applies more properly to the framed timbers supporting the excavation.

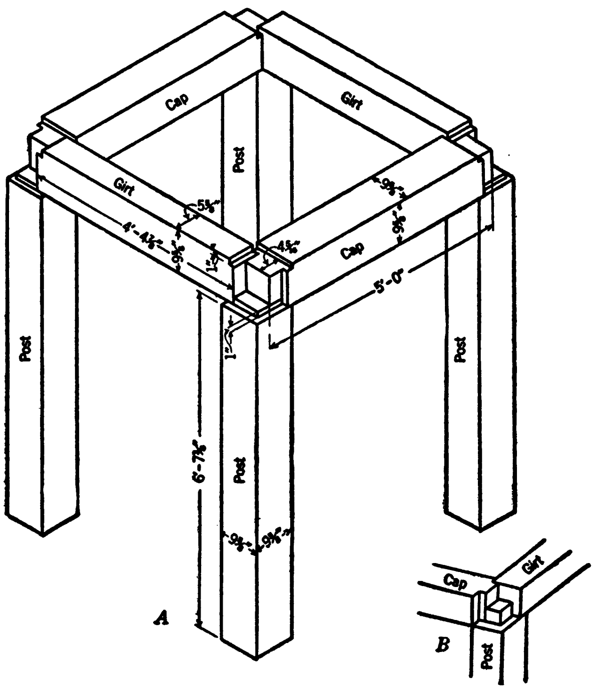

FIGURE 27.—Typical square-set timbering: *A*, Complete square-set, caps abutting; *B*, framing with posts abutting.

In ground requiring the use of square-sets it is usually necessary to run in waste filling for permanent support soon after the ore is excavated and the supporting timber is placed. Today, square-setting is seldom employed except where prompt filling is required.

A complete set of square-set timbers consists of 4 posts, 2 caps, and 2 girts framed into each other. (See fig. 27, *A*.) Adjoining sets are framed into the initial set so that for each additional set in line with the first one, 2 posts, 2 caps, and 1 girt (or 2 posts, 1 cap, and 2 girts) are required, the timbers of the initial set forming one side of the adjoining set. Where a set is stood in a corner, filling out between timbers behind and on one side and with solid ground ahead and on the other side, only 1 additional post, 1 additional cap, and 1 additional girt are required. (See fig. 28, *X*.)

The cap timbers are usually placed in line with the direction of greatest pressure, the girts at right angles thereto tying the sets together.

Framing details vary with the nature and direction of the ground pressures and the preference of the management. There are many variations, such as those in the length of the "horn" (or tenon), depth of dap, method of abutting (whether cap or post tenons butt together), etc. Square-set framing methods have been discussed at length by Gardner and Vanderburg.[45] Figure 27 shows two common methods of framing. In *A* is shown a method of framing with caps abutting and in *B* a method with posts abutting.

Square-set stoping is employed for mining most of the ore at the Bunker Hill & Sullivan mine and the Page mine in Idaho; the Park Utah, Tintic Standard, Chief Consolidated, Lark, Utah-Apex, Silver King Coalition, Eureka Lilly, and North Lily, and other mines in

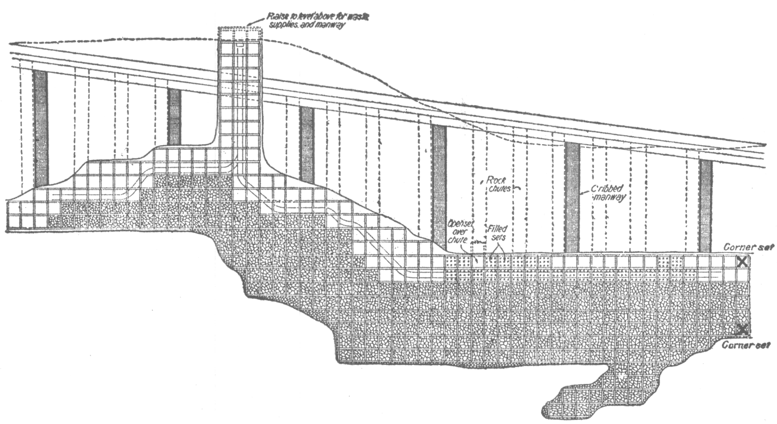

FIGURE 28.—Plan of typical floor, square-set stope, Bunker Hill & Sullivan mine.

Utah; the Comet and Black Rock mines in Montana; the Ground Hog mine in New Mexico; the Ruby mine in Arizona; and others. Numerous other mines employ square-setting for mining part of their ore, including the Pecos mine in New Mexico, where square-setting and cut-and-fill stoping are employed about equally; the Dayrock mine in Idaho; the Empire Zinc Co. mine at Gilman, Colo., where about 35 percent of the output is mined by square-setting; and the Highland Boy mine in Utah, where about 80 percent of the ore is won by square-setting.

At the Hecla and Morning mines in Idaho mining with stringer sets and filling is the chief method employed. Some authorities consider this system a variation of square-set stoping. The authors of the present paper prefer to classify the system as a modification of cut-and-fill stoping, because the timbering is quite different from regular framed square-sets and because the method and sequence of working seem more like cut-and-fill stoping. The method will

---

[45] Gardner, E. D., and Vanderburg, Wm., The Square-Set Method of Stoping: Inf. Circ. 6691, Bureau of Mines, 1933, 73 pp.

therefore be described later in more detail under Cut-and-Fill Stoping.

The following descriptions of square-set stoping methods at a number of lead and zinc mines will illustrate the procedure and principal variations of the method employed.

COEUR D'ALENE DISTRICT, IDAHO

Bunker Hill & Sullivan mine, Idaho.[46]—In mining the large, irregular masses of Bunker Hill type ore (as distinguished from the Jersey veins, which are mined by cut-and-fill) the square-set-and-fill method is employed. The practice is to remove all material showing any galena; therefore, extremely low-grade ore is sometimes mined, but by so doing high-grade ore is often uncovered. Ordinarily, one wall of the ore body is well-defined; sometimes there is no wall, but never are there two. No two cross-sections of the ore bodies are alike, even when close together. The ore bodies range from 300 to 1,000 feet in length and 30 to 125 feet in width.

Although a definite hanging wall is seldom found the rock over the ore is termed "the hanging wall." This is invariably heavy, and square-sets closely filled with waste are the only known safe method of support. As rapidly as the ore is removed square-sets are erected, and as soon as convenient these sets are tightly lagged and filled with waste.

Square-set timbers consist of 12-inch round posts and 8- by 10-inch caps and ties (or girts). The girts are unframed; that is, the ends are cut square. The square-sets, which are covered with 2-inch plank, prevent sloughing of ore from the face, protect the miners, and retain the waste filling. Neither the sets, nor any heavier timbers alone, could support the mass of the hanging wall.

Formerly, the square-sets stopes were carried up with a flat back. Later, a low, flat-arch back was carried, but at present the stopes have a high sharp arch, as shown in figure 29, *B* and *C*. This figure shows three stages in the mining of a stope in vertical cross-section. Figure 20, *C*, page 93, shows the development of the sill floor of a stope in plan and figure 28 the plan of a typical stope floor.

Figure 29, *A*, shows stoping begun on the hanging-wall side of the ore body and a footwall raise started; *B* shows the stope advancing toward the footwall, the raise completed, and the main haulage drift advancing under the footwall (note the step face of the stope); and *C* shows the sill floor completely developed and the top of the stope connected to the waste raise.

The stope is started on the hanging-wall side, the sill floor is covered with slabs, and square-set timbers are placed. These sets are extended, and the back row of sets is filled with waste to support the wall. A new floor is started above the sill floor, the first row of sets being extended into the hanging wall as shown at *A* to obtain filling for the sill-floor sets. As the stope is advanced toward the footwall and upward, a steeply arched back is maintained at all times. From the drift along the footwall and at about the center of the ore shoot a raise is started on the footwall and driven

[46] Brown, U. E., Mining Methods of the Bunker Hill & Sullivan Mining & Concentrating Co., Kellogg, Idaho: Inf. Circ. 6407, Bureau of Mines, 1931, 9 pp.

to the level above. Another drift is driven about 20 or 30 feet back in the footwall, which is extended while stoping is being started and which finally becomes the main haulage way from the stope.

It will be noted from figure 29 that, until the top of the stope and the waste raise are connected on the level above, filling material is obtained by breaking back into the hanging wall. After this connection is made, waste is drawn off from the waste raise into cars and trammed and dumped into the stope. The stope is worked so that the highest part is opposite the raise, sloping down toward

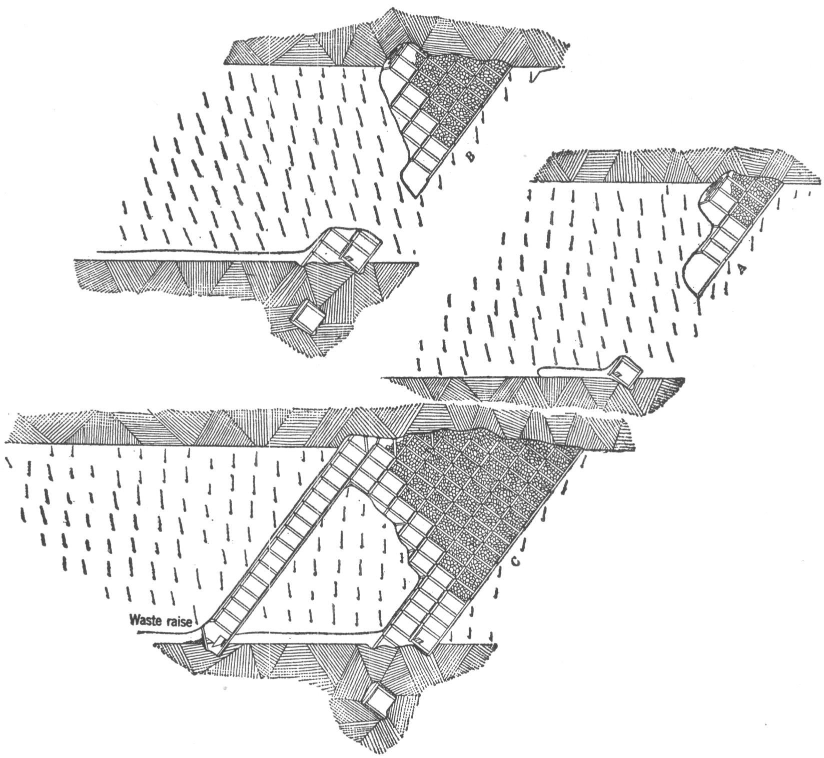

FIGURE 29.—Sections showing stages in square-set stoping, Bunker Hill & Sullivan mine: *A*, Stoping begun and footwall raise started; *B*, stope advancing toward footwall, raise completed, and main haulage drift advancing under footwall (note steep face of stope); *C*, sill floor completely developed and top of stope connected to waste raise.

each end. Thus, only those lower sets which are opposite the raises have to get their filling from the hanging wall.

Figure 21, page 95, shows a still later stage in the mining of a stope at this mine. The sill floor has been entirely worked out and filled, one or more raises have been driven from the haulage way under the footwall to serve as ore passes, and branch raises have been driven as shown to tap the stope at successively higher floors.

If the stope is very long additional raises may be driven on the footwall to the level above as the stope is extended longitudinally. Such raises, not being needed until stoping is in progress, start at one of the higher floors from a row of square-sets driven to the footwall. Figure 21 also shows the temporary slides that are used to get the ore into the chute and the waste into the sets which are being filled. Figure 28 is a typical stope-floor plan and illustrates the method of distributing the waste. It also shows rock chutes spaced 15 to 20 feet apart in the footwall of the stope and cribbed manways 50 feet apart.

The sharply arched back and vertical stope face shown in the figures have certain important advantages under the conditions at this mine. The high sharp arch throws the weight of the ore on the footwall while the weight of the hanging wall is borne by the solidly filled square-sets. The lagged, tightly filled sets are kept close to the face and thus limit the open area, reducing the size of the space over which the ore may be scattered by blasting. The blasted ore falls into chutes or, if sorting is required, on floors laid on the timbers. Stopers are used for drilling steeply inclined uppers; they are drilled more easily in this type of ground than are flat holes.

Waste for filling must be available at all times for introduction when needed in the vertical slice system. The waste entering the stope has about the consistency of wet concrete and settles in the lagged square-sets to a compact mass, forming a solid support to the stope.

Page mine, Idaho.—The following is abstracted from Berg's description of stoping practice at the Page mine.[47]

A crushed, broken ore zone with variable widths and dips of the vein requires the use of square-set stoping. Breast stoping with light drills is preferred to overhead stoping with stoper drills, since it gives better control of breaking and eliminates heavy steel losses and large stoper-machine repairs. (Note the difference between the practice here and at the Bunker Hill & Sullivan mine.)

Stopes are mined in blocks about 250 feet long and are taken in one lift of 300 feet. Ore passes are 35 feet apart and are cribbed. When a sill floor is opened, 12- by 12-inch sheeting caps are put in 12 inches above the drift cap, permitting drift timbers to be repaired without loss of head clearance. The extra caps also support the waste filling. Caps and posts are of round timber, the caps averaging 12 inches in diameter and the posts 9 inches; girts are of 5- by 8-inch sawed material. The ground is so heavy that timber must be placed as soon as room is made for a set. Some sorting is done in the stopes.

It is desirable to keep the stopes filled to within three floors or less from the back of the stope. Crosscuts are driven into the footwall to obtain waste filling. These are started 5 by 7 feet in cross-section and are enlarged to 8 by 12 feet 25 or 30 feet from the stope. The waste from these crosscuts is dragged into the stope by single-drum tugger hoists and drag scrapers.

PARK CITY DISTRICT, UTAH

Silver King Coalition mines, Utah.—The method of stoping described by Dailey[48] is not a true square-set method. Instead of

---

[47] Berg, J. E., Mining Methods at the Page Mine of the Federal Mining & Smelting Co., Page, Idaho: Inf. Circ. 6372, Bureau of Mines, 1930, 8 pp.
[48] Dailey, M. J., Mining Methods and Costs of the Silver King Coalition Mines Co., Park City, Utah: Inf. Circ. 6371, 1930, 12 pp.

regular framed square-sets, stringer sets of long square caps, round posts, and round collar braces are used regularly under average conditions. (See fig. 30, *A*.) Where the ground is heavier and the timbers are subjected to greater pressure, the stringer sets are framed as shown in figure 30, *B*. In the larger stopes portions are filled with waste to support the hanging wall as the stope advances, but in

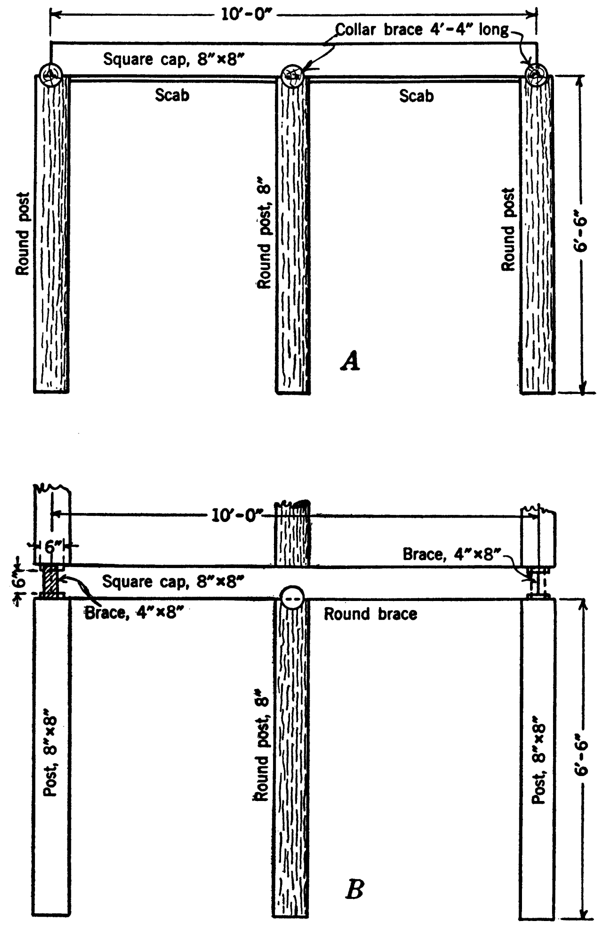

FIGURE 30.—Stringer sets used in stopes at Silver King Coalition mines, Utah.

this event as many of the timbers as possible are removed before filling and reused.

Between 15 and 20 percent of the material broken in the stopes is sorted out underground as waste and returned to the stopes for fill. If the waste thus obtained does not fill a stope, waste drifts and raises are driven in places that have merit as prospects, or for ventilation or haulage openings. After a stope section is mined out the footwall is swept clean before filling by experienced men using wire brooms. At this mine the ore areas are so large that production comes from many places.

Park Utah mine, Utah.—At the Park Utah mine three different methods of stoping are used—square-set-and-fill, cut-and-fill, and open-stull stoping.[49] In 1930 stull stopes and cut-and-fill stopes were seldom employed, since the width of the veins and the character of the wall rock in the active stopes are such that these methods are not applicable.

The square-set system practiced at the Park Utah mine is, according to Hewitt, an adaptation of the rill stope and was planned primarily to permit the waste filling to flow to its destination with a minimum of handling. The first stope section is 100 feet or 20 sets long, 6 sets high, and the full width of the vein, which may be 12 to 80 feet. Successive sections are 50 feet or 10 sets long

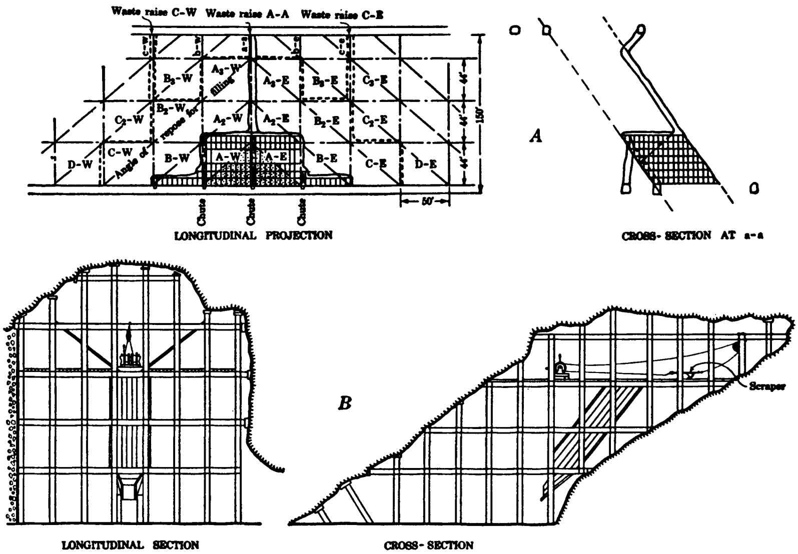

FIGURE 31.—*A*, Square-set method, Park Utah mine; *B*, scraping in flat square-set stopes, Park Utah mine.

and 6 sets high. The sections are mined out across the vein from foot to hanging wall, the sill floor being started from the top of the footwall drift.

When the 100-foot section represented by blocks A–W and A–E, figure 31, *A*, has been carried 6 sets high, stoping is started in blocks B–W, B–E, A_2–W, and A_2–E, simultaneously by horizontal slices 1 set high. The slices in blocks B–W and B–E are started at the ends and in blocks A_2–W and A_2–E from waste raise A. On each slice the square-sets are placed in position at the footwall first. The length and height of the sections are determined by the angle of repose of the waste material employed for filling—about 40°. The broken, inclined lines in figure 31, *A*, represent the angle of repose of the waste at various stages of stoping. Waste is run in from time to time as the sections are carried up, so that mining and filling are continuous. The waste, which is derived largely from develop-

[49] Hewitt, E. A., Mining Methods and Costs at the Park Utah Mine, Park City, Utah: Inf. Circ. 6290, 1930, 18 pp.

ment work, is run into the stope from hanging-wall raises connecting with the hanging-wall drift at the level above. As the stope advances to the position shown in the dotted outline additional waste raises, C–W and C–E, are driven. Waste raises are driven every 100 feet, are 5 by 8 feet in section, and are divided into 2 compartments. Chutes and manways are carried up on 50-foot centers. Horses of waste in the ore are drilled and blasted, and the waste is dropped into the stope fill.

Ore broken in the stopes is run through 8-inch grizzlies of 60-pound rail placed across the square-sets directly over the chutes. Leyner machines are used wherever possible for drilling in the square-set stopes and are mounted on a cross arm attached to a column. When this type of machine is used and horizontal holes are drilled a less ragged back is left after blasting than when stopers are used.

Scrapers are employed wherever possible in mining and for distributing waste filling, and they have been used satisfactorily in all three mining methods. Hand shoveling in stoping operations is used only for cleaning down the footwall preparatory to running in waste filling, for final leveling off of the fill after a section has been carried up to the floor pillar in square-set stoping, and for leveling waste near the hanging wall in cut-and-fill stopes. Double-drum air or electric hoists are employed with 31-inch hoe-type scrapers weighing 350 pounds. Figure 31, *B*, illustrates adaptation of scrapers to square-set stoping. The hoist is mounted on a truck and moved over a 3-inch plank floor to any desired line of sets. The scraper is dragged in any line of sets from the footwall to the hanging wall of the stope. It is stated that stoping costs are 30 percent less with scraping than with hand shoveling. One square-set stope, 4 sets long by 7 sets wide, produced 2,000 tons in 1 month with 4 men on each of 2 shifts.

TINTIC DISTRICT, UTAH

Tintic Standard mine.—Wade[50] has discussed the stoping methods at this mine at some length. The following notes are abstracted from his discussion.

The general plan of stoping has been to drive a vertical square-set raise from the footwall at the level from which the stope is started to the level above. (See fig. 32, *A*, at *a*.) From this raise the stope is started, and through it waste is dumped when filling is required. For every stope (not stope section) started it is necessary to have at least one raise to the level above for introducing waste filling.

The first section of a stope started at the raise is cut two or three sets wide and extends from the raise to the hanging wall. This section is mined floor by floor as high as safety will permit; then it is filled, leaving manways on the outside of the section at convenient places. (See fig. 32, *A*, at *b*.) These manways afford a point of attack when the next section is started; they also give access to any floor above without driving a raise.

[50] Wade, James W., Mining Methods and Costs at Tintic Standard Mine, Tintic District, Utah: Inf. Circ. 6360, Bureau of Mines, 1930, 21 pp.

The original section is completed to the hanging wall or mined 8 floors high and is filled before a new section 2 sets wide is started from the sill on one side. While the second section is being filled a third is started on the other side of the original section. By alternating from one side to the other a regular output can be maintained.

After a number of sections have been mined there is a filled stope with a vertical face of ore on the footwall end. This ore is mined in

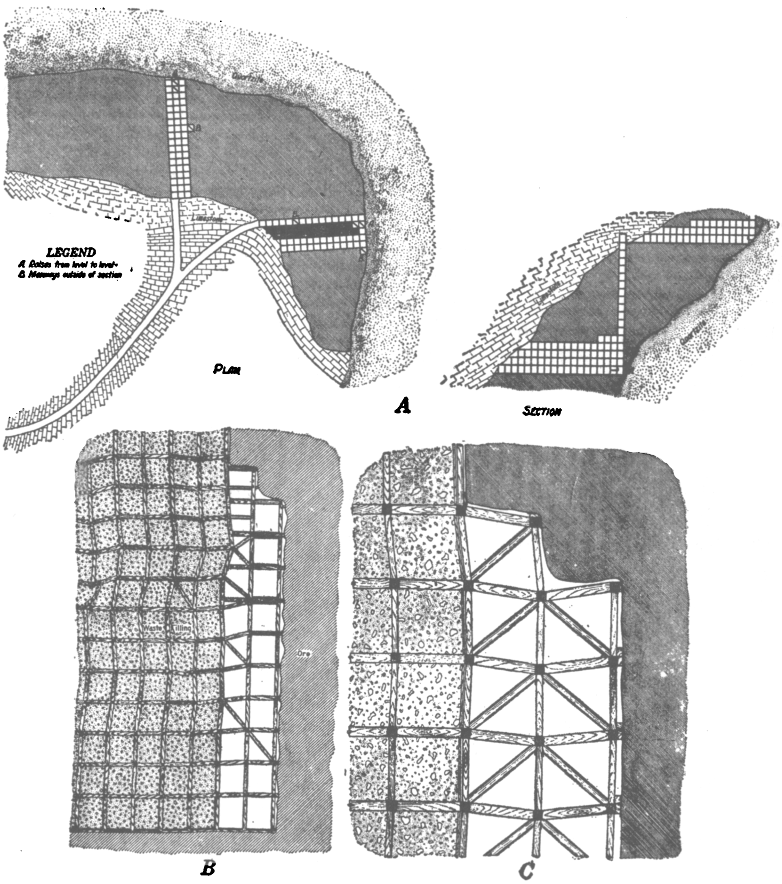

FIGURE 32.—*A*, First step in starting a stope, Tintic Standard mine; *B*, vertical section of stope, showing method of timbering; *C*, horizontal section through stope, showing use of diamond bracing.

the same manner, a two-set section from a raise being started and advanced up the footwall.

Virtually all stopes produce 3 different types of ore besides waste, and the 3 different products are shipped separately. Each type of ore is broken separately when possible; otherwise, sorting must be done. To insure a fixed tonnage of any one product more stopes must be operating than would be needed if only one type of ore were produced.

The ground in this mine ranges from solid ore to material as fine as sand. Few sections can be mined floor by floor and filled. If ground will not stand while one floor is mined some variation of square-set mining is employed, such as working vertical instead of horizontal cuts, working from the top down by floors, or working from the top down by vertical cuts or by combinations of these methods.

Stopes settle during and for a period after filling. Filled stopes will settle as much as 7 feet between levels spaced 120 feet apart, causing great distortion of the timbers and settling of the filled stope from the level above. Timbers in sections mined along the sides of these filled stopes are affected considerably. The best method of combating settling is to keep the filling as close to mining as possible, allowing stopes to settle while they are mined.

The square-sets consist of 9- by 10-inch posts, 10- by 10-inch caps, and 6- by 10-inch girts with 6- by 10-inch sills. Caps are placed parallel to the footwall so that the ends bear against the gob and settling of the fill will not disconnect the timbers at the tenons. Before ground is removed to make room for a set along the side of a filled stope, a stringer is placed against the gob posts and braced to prevent the gob from moving.

Figure 32, *B* and *C*, shows vertical and horizontal sections through a stope and the method of timbering.

BUTTE DISTRICT, MONTANA

Black Rock mine.—The stoping methods at the Black Rock mine, Butte & Superior Mining Co., have been described by McGilvra and Healy,[51] from whose description the following is abstracted.

The walls of the vein and the ore body itself are very heavy and will not stand open even when supported by timber. The vein when undercut needs support as soon as an opening is made and soon develops much side and down pressure. Square-set stoping is employed throughout the mine; it gives maximum support to walls and back and permits the required flexibility in mining. Waste filling is kept as close as possible to the mining breast; often, additional support in the form of bulkheads is required.

The ground in this mine may be divided into three classes:

1. Ore and walls strong enough to be mined by carrying a stope upward with square-sets followed closely by filling.
2. Loose and running ground that would cave if worked upward. This type of ground is mined by underhand stoping, the square-sets being carried downward.
3. Ground intermediate between these classes which is mined by a top-slicing method using square-set timbering.

Standard square-sets are of round timber 10 to 12 inches in diameter. Sets are 5 feet 10 inches square by 8 feet high, measured from center to center of the timbers.

Overhand stopes are silled out the width of the ore body on the first floor above the footwall drift which is timbered with framed sets. Chutes are installed at 30-foot intervals along the drift. When

[51] McGilvra, D. B., and Healy, A. J., Methods of Mining at the Black Rock Mine, Butte & Superior Mining Co., Butte District, Mont.: Inf. Circ. 6370, 1930, 16 pp.

a stope is started "sheeting caps" are placed and blocked about 14 inches above the cap of the sill set. (See fig. 33, *A*.) When the second floor of the stope is completed, sheeting poles are laid across the caps, and scrap timbers are placed over the poles to hold the waste fill.

A stope is carried upward in short blocks 4 to 10 sets long. A chute and manway are maintained at the end of the block and later form a supply raise for the next block. Where the vein is wide the hanging-wall half of the vein is stoped out first. Two-inch lagging is used to retain the waste fill on the end of the stope adjoining ore

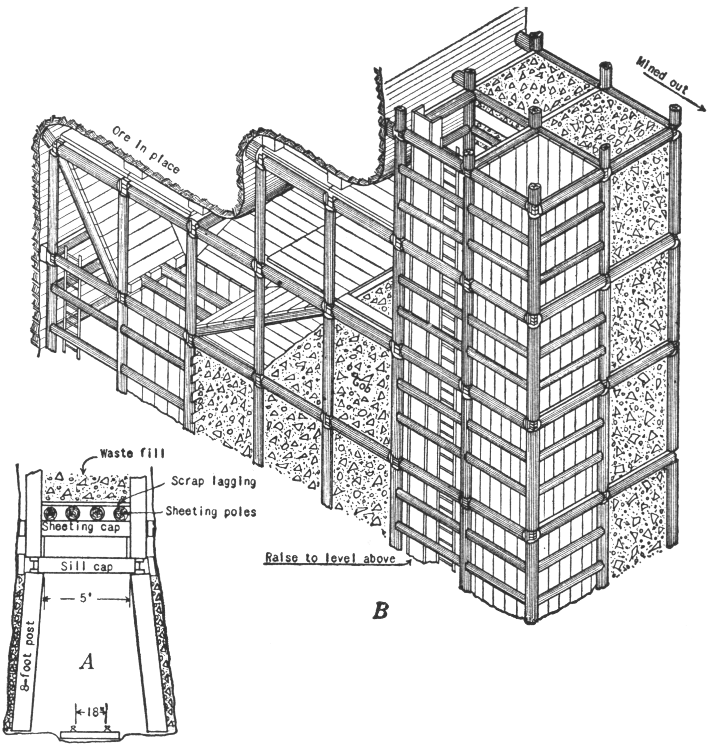

FIGURE 33.—Overhand square-set stoping, Black Rock mine, Butte, Mont.: *A*, Sill set, showing sheeting cap; *B*, general cross-section of overhand stope, footwall view.

in place. Usually, only one set of ground is blasted out at a time. The ore is shoveled into chutes by hand, or if clean enough it is blasted onto a slide. Bands of waste are blasted separately as they are encountered and are dropped into the gob. When waste cannot be blasted separately it is sorted out on the mining floor by hand. Figure 33, *B*, is a section of an overhand stope.

Stopes over two sets wide usually require cribbed bulkheads, as well as square-sets and waste filling. When a block has been mined to within 2 or 3 sets of the level above a line of sets from foot to hanging wall is carried up, the sill timbers of the upper level being caught with stringers.

In underhand stoping with square-sets (see fig. 34) a raise is first driven from one level to the next. Starting from the raise a cut is made in the ore on the next floor below the sill of the upper level.

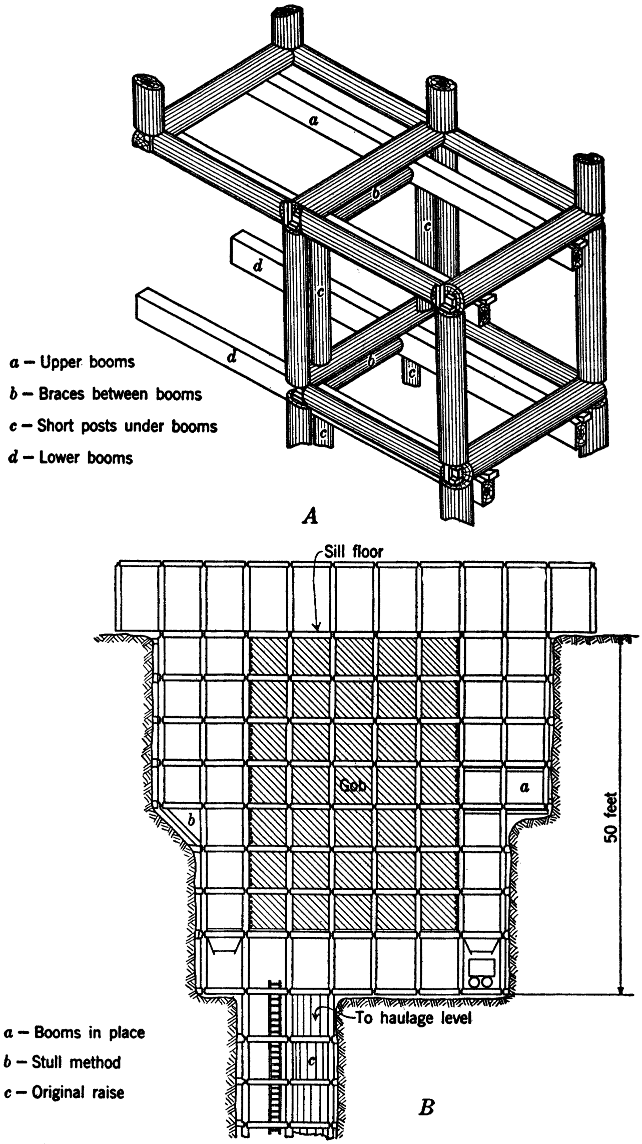

FIGURE 34.—Underhand square-set stoping, Black Rock mine, Butte, Mont.: *A*, Method of using booms; *B*, vertical section of stope.

Booms are then placed to catch the bottom of the sill timbers. (See fig. 34, *A*.) Booms are of 5- by 10-inch timber 13 feet long. If down pressure is excessive booms are used for every set, and two pairs of booms are employed; the upper pair is moved down when the next

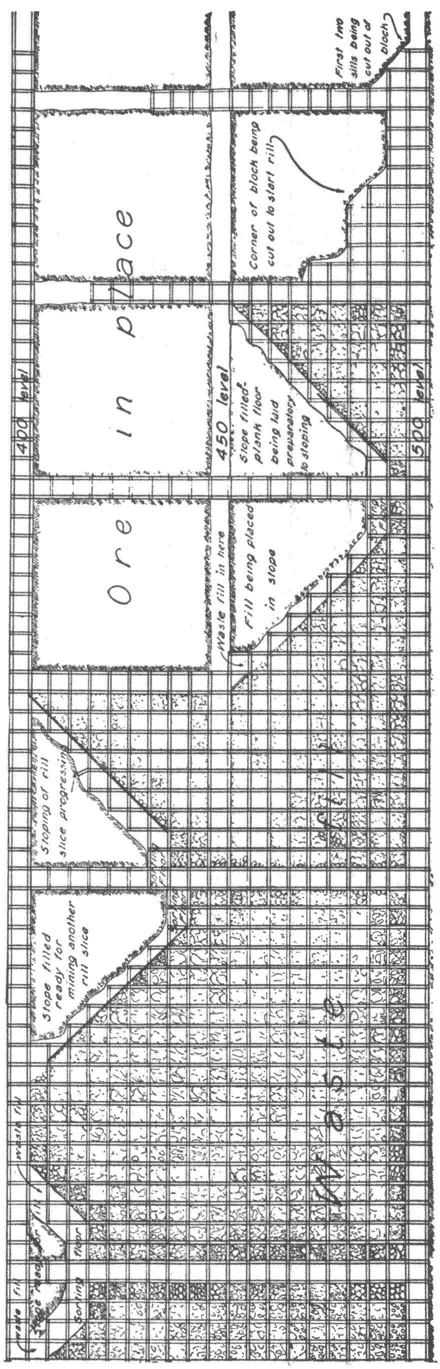

FIGURE 35.—Square-set-and-fill stoping method, Ground Hog mine, Vanadium, N. Mex.

lower floor of the stope is to be mined. If side weight is excessive a diagonal stull will hold the cap in place while the ground is being worked out for posts. Caps are used for posts in underhand stoping.

After the first set is in, the ground for the set immediately below and for each succeeding set is taken out in like manner until the stope is 50 feet below the level. At this point an intermediate level is developed along the footwall to be used for tramming ore back to the chute raise. (See fig. 34, *B*.) Starting at the top again, another line of sets is mined down alongside the first sets on the hanging-wall side. Mining is thus continued until a line of sets from foot to hanging wall has been mined out vertically from the upper sills to the intermediate level. The next line of sets is then mined. As mining progresses waste filling is run in and is held in place by small poles or old lagging. Usually, a complete column of sets is filled at one time.

As soon as a few lines of sets are mined and filled a similar stope may be started on the opposite side of the original raise. The ore below the intermediate level is mined in like manner after the upper half has been worked out.

Top slicing with square-sets is employed in heavy ground that ordinarily would be worked by the underhand method, except for some abnormal local condition that makes it impractical to hold intermediate workings open any length of time. Slices

1 to 3 floors thick are taken out, starting from the raise and working from the top downward. No filling is used, but the workings close in of themselves and fill the stope. Successive slices are driven below each other under the caved ground.

SANTA RITA-HANOVER REGION, NEW MEXICO

Ground Hog mine.—Figure 35 illustrates the square-set stoping system employed at the Ground Hog mine, as described by Richard.[52] He states that where the vein is narrow the ore is mined and the stopes are filled with the use of very little timber. Where the vein is wide, filled square-set stopes are used.

Stoping is started on the main level by mining out the full width of the ore and placing 10- by 10-inch floor sills upon which double 2-inch flooring is laid from wall to wall at 5-foot intervals. In ore over 10 feet wide square-sets are placed on the sills. The square-sets are 5 feet square by 7 feet high; 8- by 8-inch posts and caps and 6- by 8-inch ties or girts are used. Before stoping, stulled raises are driven to the level above on 50-foot centers.

Flat-back stopes were formerly employed; but on account of slips entering from the hanging wall, with resultant caving of ore and wall rock, this method was dangerous.

Rill stoping is now used. A rill stope is started by stoping next to the center raise of a block of ore to as great a height as possible and then running in fill through the raise to its natural angle of repose. Floor is laid on the fill, a 3-foot inclined cut is taken out, and another fill is begun. This process is repeated until the toe of the slope is within about 10 feet of the chute, after which the length of the slide is not increased; a level space is maintained next to the chute during subsequent stoping to serve as a shoveling floor. Most of the ore broken on the rill slides to this floor, where it can be sorted.

Slabs of ore could be held better with this system than in flat stopes since, although a single inclined cut exposes more slips, each is exposed only for a short distance. The flat shoveling space provides facilities for sorting the waste from the ore, and the shoveler works only in one place where he can be protected by timbers if necessary.

BATTLE MOUNTAIN DISTRICT, COLORADO

Gilman.—At Gilman most of the production comes from cut-and-fill stopes.[53] However, a square-set aisle is frequently carried on each side of a cut-and-fill stope, reducing the unsupported span which only rarely is as much as 35 feet.

Square-set stoping is employed in mining blanket ore at the top of the Zebra limestone and directly under the capping, where it is usually irregular in outline, of variable height, "short" and weak, and covered by heavy capping. Much of this ore is developed as it is mined. Mining is started from a raise put up to the top of

---

[52] Richard, F. W., Mining Methods and Costs at the Ground Hog Unit, Asarco Mining Co., Vanadium, N.Mex.: Inf. Circ. 6377, 1930, 13 pp.

[53] Borcherdt, W. O., The Empire Zinc Co.'s Operation at Gilman, Colo.: Eng. and Min. Jour., vol. 132, Aug. 10, 1931, pp. 99–104.

the ore, using square-sets for support. Usually the ore is broken upward from the bottom, temporary timber slides being placed in the square-sets to direct the broken ore to the chutes. Use of filling depends upon the requirements of each stope, and no advance provision is made for handling waste. If the square-set support retards caving of the back so that the working faces are not endangered, no filling is provided. If filling is deemed necessary it is obtained by driving a raise into the capping above the stope, which is branched out to form a Y a short distance above the back of the ore. The faces of the Y's are then drilled with long holes, which are loaded heavily and fired simultaneously.

When the ore is short and sandy it is worked down from the top by underhand square-setting. This method has been applied to some ore that is practically unconsolidated sand, picking only being required to make room for the sets.

Pyritic ore bodies of the chimney type are also worked by square-setting, overhand and underhand stoping being employed as conditions may require in specific instances.

SUMMARY OF SQUARE-SET STOPING METHODS

Square-set stoping is applicable to the mining of high-grade ore which is too weak to stand long without support even over spans of a few feet, where the walls require prompt support by filling, and where (due to the necessity of providing support to prevent caving of the overburden) top-slicing cannot be employed. It may often be employed to advantage as an adjunct to other methods for mining out pillars between filled stopes, or floor pillars under a mined-out level, and for other special purposes. In the most common application of square-set stoping the stopes are mined from the bottom upward by overhand stoping, but it is also employed for mining from the top downward in exceptionally weak, loose, or running ground; and for mining laterally with a vertical stope face as at the Bunker Hill & Sullivan mine. Square-sets are also employed occasionally to provide temporary support in top-slice stoping instead of drift sets, which are commonly used for this purpose.

Although the square-set method should not be used where other methods are applicable on account of the high stoping cost, it has certain advantages. The principal advantages and disadvantages are enumerated below.

Advantages:

1. Irregular ore bodies can be mined, leaving horses of waste or lean ore in place, and tongues or offshoots of ore may readily be followed.
2. Waste may be sorted out in the stopes, and ore dilution can thus be reduced to a minimum.
3. Square-set stoping affords a means for complete extraction of the ore afforded by no other known method in weak, heavy ground where the walls and overburden must be supported permanently to prevent caving and subsidence of the overlying rock.
4. By prompt filling after excavation of the ore and timbering only a small amount of ground need stand open at a time.
5. The grade of ore mined can be controlled readily, since only one set of ground is broken at a time, whose grade can be determined before extraction by careful inspection and sampling.

Disadvantages:

1. The cost of stoping is high due to the cost of timber and of framing, handling, and placing the timbers, the slow rate of extraction, and the limited size of individual blasting rounds.
2. Extraction of the ore is slow, and only a small tonnage is mined per man-shift.
3. Fire hazard is present due to large amount of timber in the stopes.
4. The accident rate is higher than that for any other underground stoping method, undoubtedly due in part, however, to the fact that it is usually employed in bad ground.

Square-set stoping has been discussed recently in detail by Gardner and Vanderburg in Information Circular 6691, Square-Set System of Mining.

CUT-AND-FILL STOPING

In cut-and-fill stoping the ore is mined upward from the level. The stope is started on the level, above the sill-floor timbers, or on top of an arch pillar left over the level, by breaking out a horizontal or inclined slice of ore, removing the broken ore through chute raises, then filling the space left by the excavation with waste. (Filling is mine rock or some other material especially provided for the purpose, such as sand, quarried rock, or mill tailings.) Another cut is then made above the fill, and the cycle of operations is repeated. The method is applicable to the mining of deposits dipping at angles of 50° to 55° or more, or to large bodies of ore of considerable vertical height, with strong or moderately strong ore and weak walls, one weak wall, or walls that become weak after short exposure.

In tabular deposits where the ore will stand safely unsupported when undercut the full width of the lode, the stopes are usually run longitudinally. (See fig. 36.) In wider deposits the stopes are commonly run transversely, either one alongside the other (the first one or preceding ones having been mined out and filled) or, more often, by a series of stopes being driven across the ore body and separated by stope pillars which are extracted after the stopes have been worked out and filled. (See fig. 37, *A*.)

The successive cuts in a stope may be taken out horizontally (referred to as horizontal or " flat-back " cut-and-fill stoping) or at an angle approximating that of the angle of repose of the filling material (termed " rill stoping ").

The method as commonly used requires only a small amount of timber for temporary support. Usually the only timber required in the stope itself is that employed as temporary props or cribs to support local patches of bad ground; thus timbering is not used systematically. In the "stringer-set" variation of the method, employed at some of the mines in the Coeur d'Alene district of Idaho, considerable timber is required.

Cut-and-fill stoping practice varies considerably at different mines and under different conditions.

Among the lead and zinc mines employing regular cut-and-fill stoping as the principal or as an auxiliary mining method are the following: Sunshine Mining & Milling Co. and several smaller producers in Idaho; Empire Zinc Co. at Gilman, Colo.; Block P mine at Hughesville, Mont.; Pecos mine in New Mexico; and Two G mine at Tybo, Nev.

In the Coeur d'Alene district of Idaho, the Hecla and Morning mines employ the "stringer-set" variation of the general method; together they accounted for about half the tonnage of lead and zinc

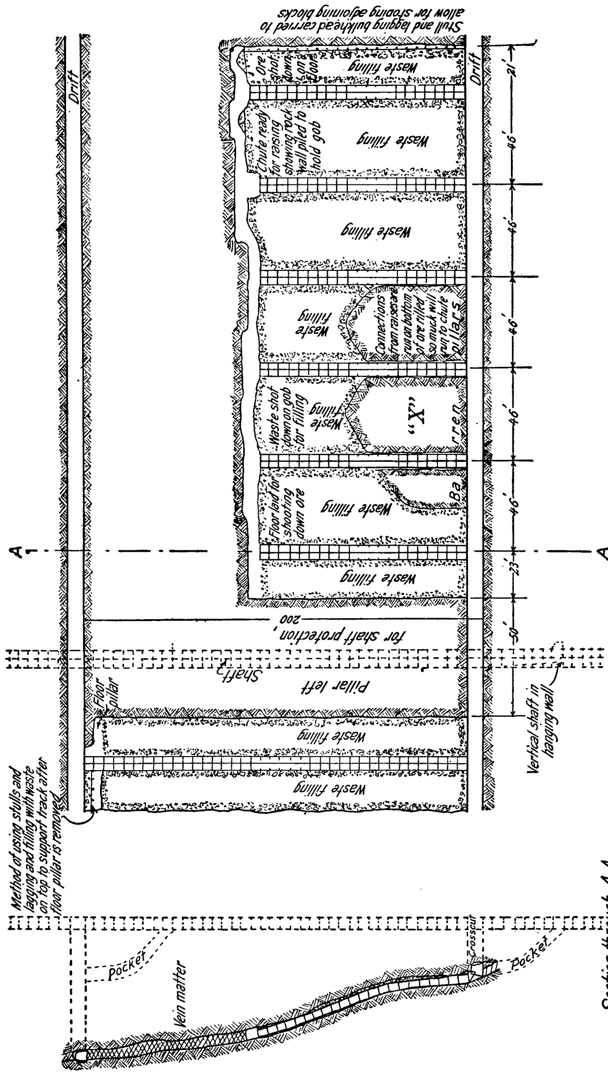

FIGURE 36.—Longitudinal and cross sections of typical cut-and-fill stope, Block P mine, Hughesville, Mont.

ores mined in the United States in 1930 by cut-and-fill stoping. Only one other mine is reported as having produced over 100,000 tons of lead or zinc ore in 1930 by this method; the remaining lead and zinc mines had only a small output in 1930 or produced only a small part of their output by the cut-and-fill method.

BARKER DISTRICT, MONTANA

Block P mine.—This mine, at Hughesville, operates in a narrow vein; the ore is 1 to 4 feet thick but is mined to a width of 5 feet. The vein dips 65° to 88°.

The ore is mined by horizontal cut-and-fill stoping. The vein is so narrow in places that waste must be broken to allow room for working; moreover, the arrangement of the ore in bands with inter-

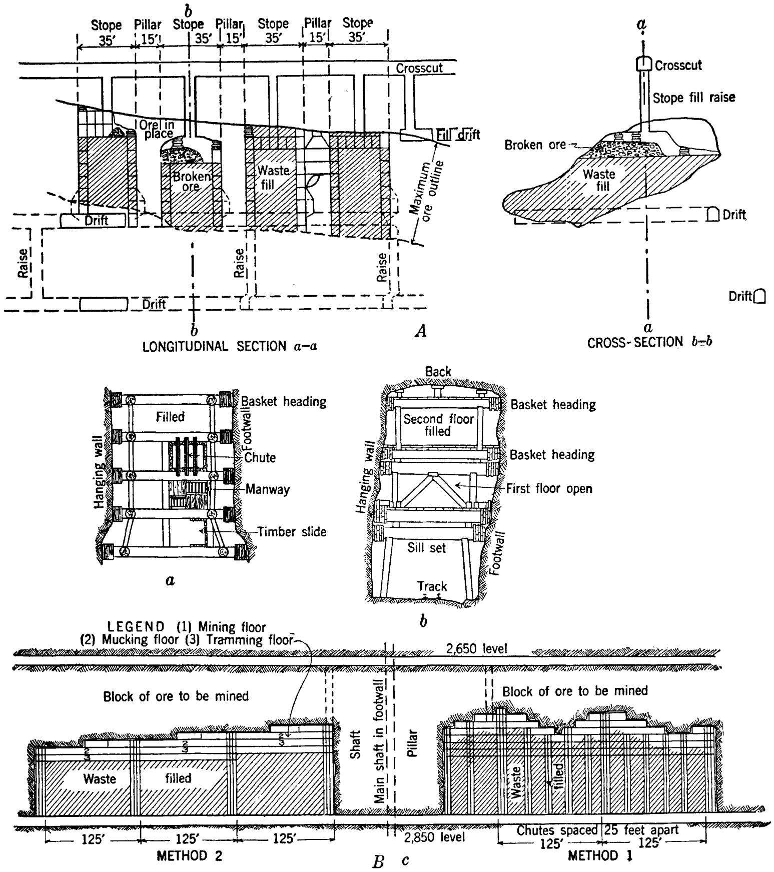

FIGURE 37.—*A*, Cut-and-fill stoping, employing alternate stopes and pillars, Gilman, Colo. (after Borcherdt, Eng. and Min. Jour.); *B*, stoping with stringer sets and fill, Morning mine: *a*, Plan of stope timbering; *b*, elevation of stope timbering; *c*, generalized longitudinal section through stopes.

vening bands of waste necessitates a large amount of hand-sorting and selective mining to maintain the grade of ore sent to the mill.[54] The following description of stoping at this mine is abstracted from Vanderburg's paper.

Stopes average 400 feet in length, 750 feet being the maximum. Figure 36 shows vertical longitudinal and cross sections of a typical

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

stope. The vein is developed on each level by a drift timbered with standard drift sets. A raise to connect with the level above is started 690 feet from the shaft as soon as the drift has advanced far enough, other raises being put through at 690-foot intervals. These raises improve ventilation and explore the vein ahead of stoping. Beginning at the shaft, every ninth drift set is placed on a 6-foot instead of the regular 5-foot interval to accommodate a chute and manway, giving a spacing of 46 feet for stope raises.

When a stope is being started the lagging over the back of the drift sets is removed, and a 5-foot cut is blasted out of the back onto the drift floor for the entire length of the stope section. This ore is shoveled into cars by hand. Chutes are then installed in the sets provided to receive them, raise timbering is begun, and lagging is replaced over the drift sets. The stope is then carried upward by successive horizontal cuts along the full length of the section. Selective blasting and hand sorting are employed to keep ore and waste separate and to provide a high-grade product. The amount of waste produced will more than fill the stopes. Where ore and waste occur in alternate bands across the vein they are blasted together, and the waste is sorted out by hand. Where the ore occurs as a single strong band in the vein the waste is blasted first, then the ore. Where the ore is weak and in a single band the ore is blasted first, then the waste.

Before blasting, a 3-inch plank floor is laid over the fill to prevent mixture of ore with the filling material. Holes are drilled with hand-rotated stopers, and one man usually drills about 15 holes per shift. The holes are inclined slightly so that the ore is thrown toward the raise by the blast. When the stope is mined to within about 5 feet of the level above, stulls are placed across the stope on 5-foot centers and lagged over, and waste is run onto the lagging to support the track on the level. The fill is lagged off from the unbroken ore at the end of a stope section by a vertical row of stulls set on 5-foot centers and covered with 3- to 5-inch-diameter poles placed vertically and nailed to the stulls. Figure 36, *X*, shows the procedure when a horse of barren material is found.

As the stope is carried upward, rock walls are built around the tops of the chute raises on each mining floor, allowing the stope floor to be raised the height of one raise set without timbering. After a wall reaches a height of 4 feet a set of raise timbers is installed inside the rock-wall enclosure. Chute and manway raises are built up periodically by timbering crews comprising a timberman and one helper.

PECOS DISTRICT, NEW MEXICO

Pecos mine.—Matson and Hoag [55] have described stoping methods at this mine, and the following notes have been abstracted from their description.

The ore shoots are irregular and not continuous, pinching out and giving way to others. Sometimes, 2 to 4 parallel veins are separated by only 10 to 20 feet of barren, loose schist or blocky diorite. Occa-

[55] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Inf. Circ. 6368, Bureau of Mines, 1930, 21 pp.

sionally the parallel veins can be mined simultaneously, leaving the waste in place, but usually the waste must be mined with the ore and gobbed back in the stope.

The mining method must provide for sorting of wastes to avoid serious dilution, as at least 20 percent of the ground broken in the stopes is rejected as waste. It must be such that runs from the walls can be checked quickly and that the walls can be prospected at close intervals for parallel ore shoots. The surface must be kept intact to prevent an inflow of water from the river which runs parallel to the outcrop 400 feet away.

Cut-and-fill and square-set-and-fill methods are used about equally. Strong walls and ore favor the use of the cut-and-fill method.

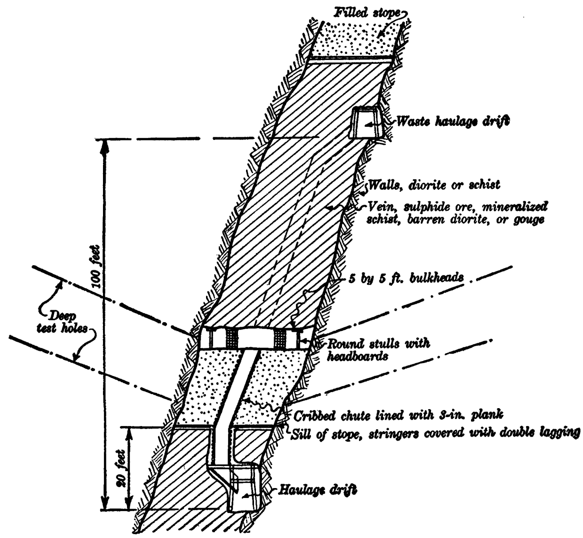

FIGURE 38.—Section across vein through cut-and-fill stope, Pecos mine, New Mexico.

Sometimes a stope will be started by one method, and the other method will be substituted later.

Inclined cut-and-fill stoping has been tried and abandoned. The tendency of the ore and walls to cave and run, the difficulty of erecting stulls, cribs, or bulkheads on an inclined floor, and the lack of storage space for waste with this system offset any advantage to be gained from gravity handling of ore and waste on the inclines. The horizontal cut-and-fill method is therefore employed. Figure 38 is a vertical section across the vein through a cut-and-fill stope. It will be noted that the stope is silled out above the level, leaving an arch pillar of ore over the back of the drift. Cut-and-fill stoping is used wherever possible, although often the ore and walls are so loose that many stulls and bulkheads are required.

The ore bodies are mined in sections, the length of each depending upon the width of the vein and ground conditions. During the first 2 years of operation stope sections in wide veins were 15 to 60 feet long with 15-foot pillars between them. Recent practice in wide veins has been to cut the initial stopes 30 feet long and space them 100 to 150 feet apart, with extraction chutes at both ends of the section. After the first stopes have nearly reached the level above, new sections (15 feet long as a rule) are started on one or both ends, using the former extraction chutes for waste raises.

Stopes are mined upward by successive 7-foot horizontal cuts from hanging to foot wall by filling after each cut. In narrow veins stopes are silled several hundred feet in length with chutes spaced 30 feet apart. Every other chute is provided with a manway. Fill raises are driven 30 or 60 feet apart in ore.

Stopers are used for drilling where possible, but in many places the ground is so loose that flat holes drilled with jack hammers are more satisfactory. Before blasting, a shoveling floor of 2-inch plank is laid on the fill. All shoveling is by hand because of the need for sorting. There is little heavy blasting. Back and floor pillars are left until all stoping is completed on the level below and then removed in short sections by square-setting. Waste filling for the stopes is obtained from development work on the levels, from prospect workings driven from the walls of the stopes, from sorted waste, or from a surface glory hole.

BATTLE MOUNTAIN DISTRICT, COLORADO

Gilman.—Most of the production from the property of the Empire Zinc Co. at Gilman is obtained from horizontal cut-and-fill stopes and square-set stopes.[56] In elliptically shaped, cylindrical ore bodies which are 35 to 60 feet high vertically and 100 to 250 feet long horizontally the ore body is sectioned into transverse stopes and pillars. Stopes are 35 feet and pillars 15 feet wide. The stopes are worked by ordinary cut-and-fill methods from the base to the top of the ore. In some instances the ore is mucked by hand to drawing chutes, but more often it is scraped. Filling from development or waste-caving stations is passed into the stope through raises from an overlying level and is spread by scrapers. Filling may either follow the ore cut closely or be introduced after a complete 8-foot slice is removed, depending on the condition of the ore back.

A square-set aisle is carried on either side of the stope, reducing the span, which rarely exceeds 35 feet. The square-sets also partition the waste from the walls and provide a tie for the 15-foot stringers and trusses that comprise the timbering in pillar stoping. Timber cribs are used for temporary support in the stopes when necessary. Stopes are filled to the top and cribbed tightly against the back. Figure 37, *A*, illustrates the stoping method at this mine.

Pillar removal follows mining of the ore from the stopes. The procedure is to break the ore from one end of the pillar, drawing it through chutes in the square-set aisles and timbering across these aisles with 10- by 10-inch by 15-foot stringers. Mining progresses longitudinally from one end of the pillar to the other.

---

[56] Borcherdt, W. O., The Empire Zinc Co.'s Operation at Gilman, Colo.: Eng. and Min. Jour., vol. 132, Aug. 10, 1931, pp. 99–104.

COEUR D'ALENE DISTRICT, IDAHO

Morning mine.—The stoping method at this mine has been discussed by Wethered and Coady.[57] The swelling nature of the ground and the gradual breaking down of the walls along innumerable shear planes results in heavy ground which requires timbering. It is necessary to support the walls immediately after removal of only a small cut, and a regular system of timbering is used as in square-set stoping; unframed stringers reaching from wall to wall are employed, however, instead of framed caps, posts, and girts. (See fig. 37, *B*, *a* and *b*.) The method thus bears a decided similarity to square-set stoping. Figure 37, *B*, *c*, illustrates the stoping system in longitudinal section. Method 1 is a system employing chutes on 25-foot centers and method 2 a system with chutes on 125-foot centers. The cost of maintaining chutes on 25-foot centers was extremely high, and the other method is now in general use. In method 1 the broken ore could, of course, be shoveled directly into the chutes, while in method 2 it is shoveled into cars and trammed to the chutes.

Regular drifter machines are used in stoping. A stope round consists of 10 to 20 holes, depending upon the width of the ore. Holes are drilled to a depth of about 5 feet. Cuts are taken 9 feet high.

In the present stoping system the broken ore falls to a mucking floor, one set below that from which the drilling is done, and is shoveled by hand through movable steel slides into a 1-ton car on the tramming floor one set below. The cars are trammed to the chute and dumped.

As soon as the face on the mining floor has advanced 125 feet the track, car, and steel slides are moved up one floor, a new mining floor is started, and the cycle is repeated. Waste filling follows the stope upward as shown in figure 37, *B*. Waste for filling is obtained from sorting the ore in the stopes, or from waste raises and crosscuts. Scrapers are employed for dragging the waste from waste crosscuts into the stope. The waste fill is not leveled but allowed to lie at its angle of repose in the stope. The cycle of operations corresponds to that of regular cut-and-fill stoping, consisting of cutting out a slice, removing the broken ore, and filling. This procedure is repeated to about the eighteenth floor of the stope; the stope is then filled. The remaining floors are mined in vertical sections from the level above downward.

Hecla mine.—A similar method is employed at the Hecla mine and has been described by Foreman,[58] who terms it the "stull-sets-and-fill method." Figure 39 is a generalized sketch of the stoping method. The stope to the right of the standard three-compartment raise shows the general advance of the floors and the method of filling, while at the left of the raise is shown vertical slicing to the level above.

The maximum width of drift is 18 feet; where the vein is wider the stope is widened to full width on the first floor. Floors are mined in both directions from the completed three-compartment

[57] Wethered, C. E., and Coady, Leo J., Mining Methods at the Morning Mine of the Federal Mining & Smelting Co., Mullan, Idaho: Inf. Circ. 6238, Bureau of Mines, 1930, 13 pp.

[58] Foreman, Charles H., Mining Methods and Costs at the Hecla and Star Mines, Burke, Idaho: Inf. Circ. 6232, Bureau of Mines, 1930, 21 pp.

raises for 100 feet. The first floor is mined from the raise to the end of the block and is timbered as the face advances. The broken ore is passed through the top lagging of the drift sets into cars on the haulage level. The second and third floors are mined likewise. As the third floor is being mined the timbers for supporting the filling in the stope can be placed. The bottom of the waste corral is on top of the first floor above the drift. After the third floor is advanced far enough the second floor can be filled with waste from the raise, a track being placed on the third floor for tramming it. When the third floor is completed and the fourth floor is begun this track is used in transferring broken ore from the face of the stope on the fourth floor, the ore being dropped from the mining floor into a car on the third floor and thence trammed to a chute. Chutes are provided and maintained through the fill at 100-foot intervals.

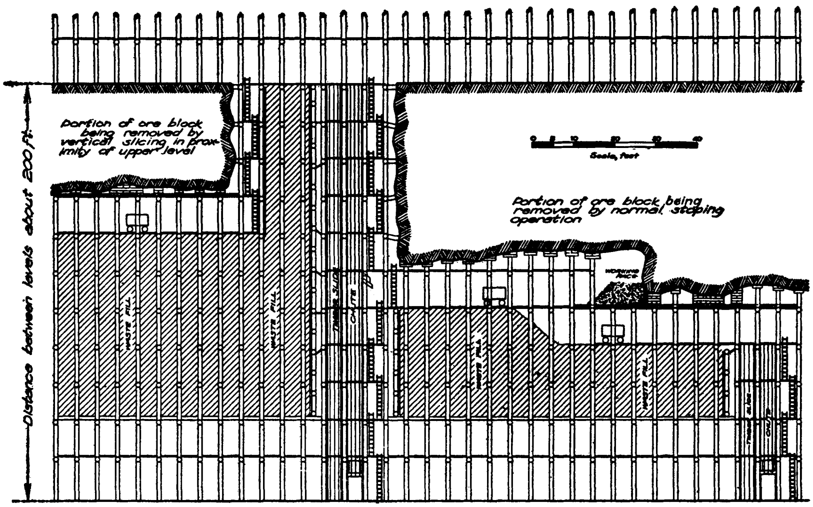

FIGURE 39.—Generalized sketch showing method of stope mining at the Hecla mine.

When the fourth floor has been advanced 25 to 35 feet from the raise, filling is started on the third floor. With the system employed at this mine practically one floor only is open at a time. When wall conditions are good, however, the track for waste fill is placed on every second floor, and two floors are open at a time between the top of the fill and the back of the stope.

The stoping of horizontal floors continues upward until only 3 or 4 floors of ground are left between the back of the stope and the level above. This ground is then mined in short vertical slices. (See fig. 39.)

The method employed allows continuous mining and continuous filling, the space between back and fill being kept at a minimum, and permits a regular schedule of operations.

SUMMARY OF CUT-AND-FILL STOPING

This method is applicable to the mining of medium- and high-grade ore bodies of strong or moderately strong ore, generally dipping 50° to 55° or more and having a weak wall or walls. The ore itself must stand well over moderate spans for a short time. In narrow ore bodies, where the ore will stand well over the full width of the vein, stopes are usually run lengthwise of the vein. In wide ore bodies the stopes are usually run across the vein and separated by pillars of ore which are extracted later.

The regular sequence of operations is: Drilling and breaking, removal of broken ore, filling, then repetition of the cycle. To operate efficiently and steadily an ample supply of waste for filling must be available at all times so that there will be no delay caused by waiting for fill when it is required.

Waste may be obtained from development headings, from sorting in stopes, from waste stopes in the walls of the vein, from surface glory holes or gravel pits, or from mill tailings. Obviously, if waste rock has to be broken for providing fill, it will increase the stoping cost appreciably.

The principal advantages and disadvantages of the cut-and-fill method are given below.

Advantages:

1. Bad walls may be held safely, and subsidence of the workings and surface can usually be prevented. (However, filling cannot be packed tightly enough under a wide back to support it indefinitely.)
2. The walls may be prospected as the stope is carried upward, and any offshoots of ore discovered may be followed and mined. Waste broken during such work can be left in the stope as filling material.
3. Areas of lean or barren material can be left unmined. Waste unavoidably broken with the ore can be sorted out and left in the stope.
4. On account of the support quickly provided by the fill, the ore is diluted less with wall rock than with shrinkage stoping.
5. The ore is removed from the stope as fast as it is broken; hence, large capital sums are not tied up in broken ore left in stopes, as with shrinkage stoping. Rapid removal of broken ore minimizes oxidation of sulphides in the ore, sometimes important if the ore is to be concentrated by flotation.

Disadvantages:

1. The rate of output from individual stopes is limited and is often irregular due to suspension of breaking and drawing operations during filling. It is a comparatively slow method of ore extraction.
2. Filling must be provided promptly when and in the quantities required, or production will be held up. Handling ore and waste on the same level often results in haulage complications.
3. The stoping cost per ton of ore is comparatively high, especially where considerable waste rock must be broken to provide material for filling.

Cut-and-fill stoping has been discussed recently by Johnson and Gardner in Information Circular 6688, Cut-and-Fill Stoping.

SHRINKAGE STOPING

Shrinkage stoping is employed as the principal method at a few lead and zinc mines in the United States and as an auxiliary method at a number of others.

It is adapted to the mining of tabular deposits dipping about 55° or more from the horizontal, where the ore and wall rocks are firm and will stand unsupported over considerable spans and where the ore is fairly regular in outline and does not contain many small waste inclusions.

Shrinkage stoping is the principal method employed at the Empire Zinc Co. mine in Grant County, N. Mex., and by the Sunnyside Mining & Milling Co., Eureka, Colo. It is used for producing 50 percent of the ore at Austinville, Va., and for part of the output from the Tamarack and Custer mine in Idaho, the Treadwell-Yukon mine at Tybo, Nev., and other mines.

In this method the ore is mined upward from the level by horizontal or inclined slices. Instead of drawing off the broken ore and filling the excavation with waste, as in cut-and-fill stoping, the stope

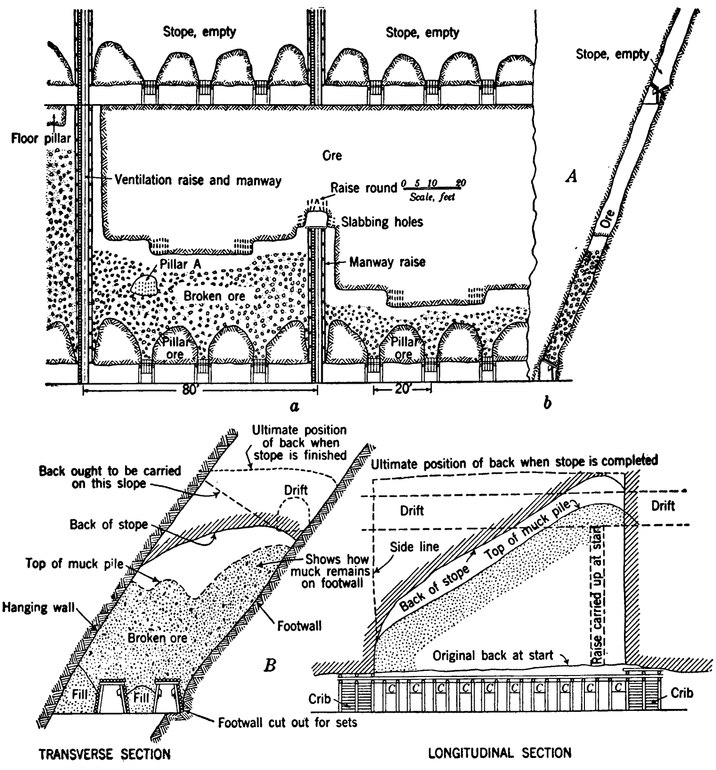

FIGURE 40.—*A*, Shrinkage stoping on drift pillars in narrow vein: *a*, Longitudinal vertical projection; *b*, vertical cross-section; *B*, sections of longitudinal stope, Franklin, N. J. (after Haight and Tillson, Trans. Am. Inst. Min. Eng.).

is kept full of broken ore, only enough ore (usually about 40 percent of that broken) being drawn off after each blast or cut to leave room between the top of the pile of broken ore and the back of the stope for the miners to work in. The miners stand on top of the broken ore to drill and blast the next cut in the back of the stope. The remainder of the broken ore (about 60 percent of the total broken in the stope) is drawn after completion of the stope to the level above or to a floor pillar below it. Figure 40, *A*, illustrates the shrinkage-stoping method.

Shrinkage stoping is capable of a number of variations. Thus, the cuts may be horizontal (horizontal or flat-back stoping) or inclined (rill stoping). Stopes may be started directly on top of the drift timbers or 15 to 20 feet above the back of the drift on arch pillars of ore left over the drift. (See fig. 40, *A*.) The stopes may also be run longitudinally with the vein in relatively narrow ore bodies (see fig. 40, *A* and *B*), or across the vein with pillars of ore intervening between adjacent stopes in wider ore bodies. (See fig. 41.)

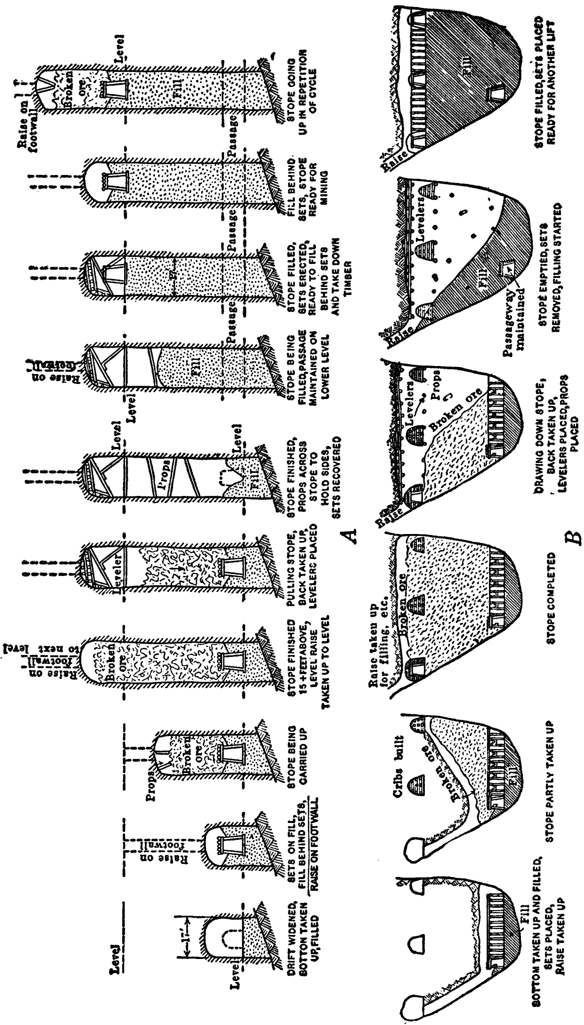

FIGURE 41.—Successive stages in mining a transverse shrinkage stope, Franklin, N. J.: *A*, Transverse sections of transverse stope; *B*, longitudinal sections (after Haight and Tillson, Trans. Am. Inst. Min. Eng.).

In narrow veins having very strong walls, stopes may often be emptied and abandoned; otherwise, it may be necessary to fill the stopes with waste, after they have been worked out and drawn empty, to prevent failure and subsidence of workings. Where transverse stopes are mined out between pillars of ore it is usually necessary to fill the stopes with waste so that the pillars may be extracted, commonly by top-slicing or square-set stoping. Sometimes, only thin pillars are left between stopes; after the stopes have been mined on both sides of these thin pillars they may be undercut, thus inducing caving, and the caved pillar ore can then be drawn off with the stope ore when the stopes are pulled empty.

Ordinarily, little if any timber is employed in the stopes themselves, although occasional stulls may be used to support local patches of bad ground temporarily.

FRANKLIN DISTRICT, NEW JERSEY

Franklin and Sterling Hill.—At Franklin most of the ore was stoped in transverse stopes 18 feet wide, separated by pillars of ore 32 feet wide. Many of these transverse stopes were mined by shrinkage stoping or a combination of cut-and-fill and shrinkage stoping. After the ore was drawn from these stopes they were filled, mill sands being employed for filling material. For some years most of the output from the Franklin mine has been obtained from pillar mining, using the top-slicing method. The top-slicing method employed at this mine will be described later under "Top-slicing." Figures 40, *B*, and 41 illustrate the shrinkage method formerly employed at Franklin.

Figure 40, *B*, shows longitudinal shrinkage stoping in narrow ore. The level was silled out the full width of the ore, drift sets were stood, and waste filling was run in around them; then shrinkage stoping was begun on top of the drift timbers. The back of the stope was arched as shown in the transverse section; the stope face was carried on an incline or rill, as shown in the longitudinal section.

Figure 41 shows the successive stages in mining a transverse stope in wide ore, as follows: Widening and timbering the haulage-level drift and filling around the sets; carrying up the shrinkage stope to the level above; catching up the back on timbers; drawing down the broken ore; placing props to support bad patches of wall; and, finally, filling the stope with sand or waste. These stopes were approximately 18 feet wide and as long as the width of the ore body, which varied up to 150 feet. Figures 40, *B*, and 41 are taken from a description of mining practice by Haight and Tillson.[59] For detailed information on the system employed the reader is referred to their earlier work.

SUMMARY OF SHRINKAGE STOPING

As previously pointed out, shrinkage stoping is applicable to the mining of tabular, steeply dipping deposits or wide, thick ore bodies of strong ore enclosed between strong walls. Some advantages and disadvantages of the method follow:

Advantages:

1. Only a small amount of stope development is required; and after level development is well started, production from stopes can be begun. Thus, the mine can quickly be brought into the productive stage.
2. Broken ore in the stopes assists in temporarily supporting the walls and eliminates the need of timber for this purpose, except for occasional props or stulls to support local patches of loose ground.

---

[59] Haight, C. M., and Tillson, B. F., Zinc Mining at Franklin, N.J.: Trans. Am Inst. Min. Eng., vol. 57, 1918, pp. 720–825.

Advantages—Continued.

3. No shoveling, wheeling, or tramming is required in stopes.
4. No long ore passes are required between levels, since ore is drawn off from the bottom through a number of chutes or short raises.
5. A large reserve of broken ore is maintained which can be drawn upon to give a uniform rate of output. (Note disadvantage 1.)
6. The method is cheap, provided walls are firm and excessive dilution of ore with waste does not occur.

Disadvantages:

1. Although the shrinkage method permits rapid development of a mine to the stoping stage, only about 40 percent of the ore can be drawn during the mining of a stope; the balance is tied up in the stope until it is completed, resulting in continuous tying up of capital for breaking unavailable ore.
2. Although long raises and ore passes are not required or are few, close spacing of chutes or short chute raises is required to pull down the broken ore evenly and prevent "hang-ups."
3. Virtually no opportunity is afforded for sorting waste from ore in stopes or for following narrow offshoots of ore without undue dilution of ore with waste. Consequently, parallel ore bodies of considerable size in one or the other wall may be overlooked.
4. Although block holing of slabs or large blocks of ore is possible on top of the ore pile, many of these may become covered with finer ore and be overlooked, remaining unbroken until they give trouble at the chutes below. In wide stopes it is often dangerous to go under the back to bulldoze chunks.
5. Some highly pyritic ores may take fire if left broken very long in stopes. More often, slight oxidation of the surface of the mineral particles while they are lying in the stope will seriously affect their complete recovery later by flotation in the milling plant.
6. Travel into and through the stopes and handling of drills, drilling gear, and drill steel over the uneven top of the broken ore in the stope are often difficult and consume a considerable percentage of the working time in the stope. Drilling equipment must be moved some distance before blasting at the end of the shift and in again at the beginning of the next shift, often under difficult conditions.
7. Accidents caused by sudden subsidence of the pile of broken ore have occurred in narrow veins, due to temporary "hanging up" of the ore followed by its collapse when men have been working over the hung-up area.

Shrinkage stoping has been discussed in considerable detail by Jackson, in Bureau of Mines Information Circular 6293.

TOP-SLICING

So far as the authors know, top-slicing is not employed for first-mining at any lead and zinc mines in the United States.

At Franklin, N.J., however, practically all the current output is produced by top-slicing the pillars left between the original stopes. The pillars average about 32 feet in width and up to 150 feet in length. The level interval is 100 feet, so that the pillars are blocks of ore 100 feet high, 32 feet wide, varying in length up to 150 feet, and lie between filled stopes on either side.

The pillars are developed for top-slicing from footwall raises connected by longitudinal rock drifts. The raises are opposite the center lines of the pillars; and a drift is driven from them through the center of the pillar on each sublevel, beginning at the top, to the hanging wall of the ore body. Raises are put up from the level at 20-foot intervals along the center line of the pillar to serve as ore passes from the slices. These are connected by a drift driven from foot to hanging wall on each sublevel.

Sublevels are 10 feet apart vertically. When the drift on the highest sublevel has reached the hanging wall, slices are started to the right and left along the hanging wall toward the filled stopes on either side and are timbered with hardwood timber, principally oak. Light blasts are fired to break the ground, using space-loading of the holes, to prevent blasting out the timbers. When the fill is approached the last of the ore in the slice is removed a little at a time, beginning at the top and working in breasting boards to hold back the sand filling. Figure 42, *A*, is the plan of a slice floor, and figure 42, *B*, illustrates the method of breasting back the filling. Slices are 6 to 8 feet wide. When the first slice has reached the fill a second slice is driven alongside it in a similar manner.

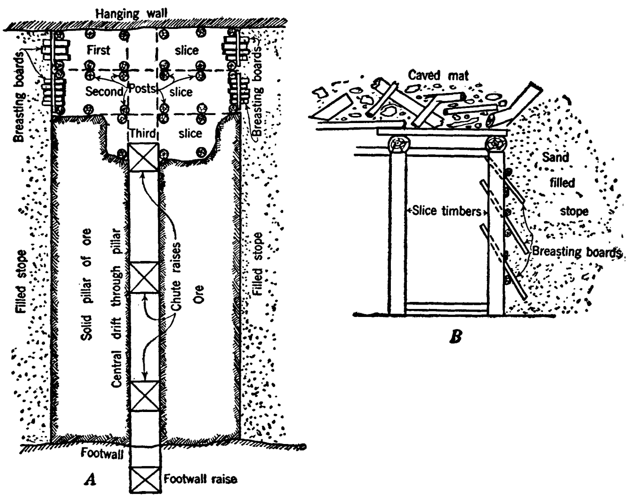

FIGURE 42.—Top-slicing pillars, Franklin, N. J.: *A*, Plan of slice floor; *B*, method of breasting off sand filling.

Sometime three or more slices can be driven before the weight becomes excessive and the timber must be blasted down. Before this is done the bottom of the slice is covered with old timber. After the top sublevel has been sliced back far enough, slicing may be started on the sublevel below, beginning, as before, at the hanging wall. Slicing is continued from the top downward until the entire pillar has been extracted.

Because the hanging wall is hard and firm it " hangs up " over the top slices, leaving an opening due to the mass of timber and fill settling as it is blasted down into the successively lower slices. Since this hanging wall might suddenly collapse when exposed over a large area, waste rock and mill sands are dumped from the level above to fill the space and provide a cushion over the top-slice workings below. In this respect the method differs radically from the usual top-slicing operation where the capping caves readily and fills the excavated areas tightly.

APPLICABILITY OF TOP-SLICING

Top-slicing is applicable to the mining of medium- or high-grade ore where it is so soft or fractured that it will stand only a short time over even small spans, where the capping breaks up and caves readily and quickly over small unsupported spans, and where it is unnecessary to support the capping and surface to prevent subsidence thereof. Top-slicing can also be used for removing pillars between filled stopes as just described. It may be used for mining loose, broken, or caved ground, the chief requirement being that when the slice timbers are blasted down the ground above will cave in and fill the opening tightly without leaving any open holes.

Top-slicing affords a safe and economical method for mining treacherous ground from the top downward (where mining upward from below would be hazardous) and for obtaining complete extraction of ore with little dilution of waste from the walls. It does not permit much sorting in the stopes, although small amounts of waste can be sorted and thrown back in the gob or into a mined-out slice which has not yet been caved.

Although considerable timber is required material of cheaper grade and smaller size can be employed than where it must support the openings over a long period, since it only stands a short time and is blasted down.

Top-slicing has been discussed in detail by Jackson in Information Circular 6410, Mining by the Top-Slicing Method, with Some Notes on Sublevel Caving.

COST OF STOPING

Stoping cost varies widely with the physical characteristics of the deposit (its size, dip, and regularity of outline); the strength of ore and wall rocks; the method of stoping as determined by these general characteristics and by the grade of ore and regularity of its occurrence; the rate of production; and the skill with which operations are conducted. Where ore and waste must be sorted in the stopes the cost per ton of ore mined is increased considerably, although the sorting may result in decreased mining and milling cost per pound of metallic lead and zinc. Likewise, a more costly mining method may give a lower final cost per pound of metal than a lower-cost method which results in loss of ore and high dilution with waste or which does not permit selective mining.

Since conditions seldom, if ever, tally at different mines, obviously direct cost comparisons between any two mines are apt to be misleading unless the conditions at each are fully known.

Some stoping methods are inherently low-cost methods. Block-caving is usually cheapest on a per-ton basis, followed by open-stope methods, especially mill-hole and sublevel variations. In general, shrinkage stoping is the next cheapest, followed by top-slicing, cut-and-fill stoping, and square-set stoping; the latter is the most expensive per ton of ore mined. The order may vary, however, in specific instances; but, in general, block-caving (not employed to any extent in zinc and lead mines in the United States) and open-stoping are cheapest, while square-set mining is the most expensive.

Table 10 gives stoping costs at a number of lead and zinc mines, and illustrates the wide variation in costs under different conditions. The tabulated costs exclude tramming, haulage and hoisting, and general distributive charges and represent only direct stoping costs. Differences in accounting practices at the different mines have made it impossible to segregate stoping costs on exactly the same basis for all mines; however, by making certain deductions from reported distributive charges at some mines, it is believed that the figures given are compiled upon substantially the same basis.

Table 11 shows stoping costs in units of labor, explosives, timber, and power for a group of lead and zinc mines.

TABLE 10.—*Stoping costs, lead and zinc mines*

Mine and location	Type of deposit	Period covered	Ore hoisted, tons	Stoping method	Direct stoping cost per ton of ore hoisted	Remarks
Mine No. 1, Tri-State district	Flat, bedded deposit	6 months, 1928	68,770	Open stopes	$0.532	$0.049 deducted from reported mining cost for development.
Mine No. 2, Tri-State district	do	9 months, 1928	87,563	do	.458	$0.050 deducted for development.
Waco mine, Tri-State district	do	May 1927 to January 1929.	364,285	do	.490	Do.
Barr mine, Tri-State district	do	January to April 1929.	45,344	do	.497	$0.047 deducted for development.
Hartley-Grantham mine, Tri-State district.	do	January to June 1929.	60,282	do	.516	
Hartley mine, Tri-State district	do	6 months, 1930	180,212	do	.572	
No. 8 mine, Southeastern Missouri district.	Flat, bedded deposits of disseminated lead ore.	1928	168,089	do	.483	
Mascot No. 2 mine, Tennessee	Large, bedded deposits; dip, 18° to 22°.	10 months, 1929	511,383	Open stopes; mill-hole system	.222	Based on ore from stopes only.
Bunker Hill & Sullivan mine, Idaho.	Wide, deep shear zone, large ore bodies.	1928	452,345	Square-set-and-fill	2.231	$0.13 deducted from reported cost for transportation.
Do	do	1931	460,366	do	2.476	$0.134 deducted for transportation and $0.518 for distributed charges.
Page mine, Idaho	2 fissure veins in shear zone; dip, 40° to 60°.	1928	90,460	do	2.15	Do.
Silver-King Coalition mines, Utah.	Large, irregular bedded deposits in limestone; some veins in quartzite.	1929	180,208	do	3.50	
Park Utah mine, Utah	Fissures in quartzite and limestone	1928	192,250	Square-set-and-fill; some cut-and-fill.	2.426	
Tintic Standard mine, Utah	Trough ore bodies replacing limestone beds.	November 1929	8,758	Square-set-and-fill	4.973	3 types of ore must be mined separately, and considerable sorting is required.
Ground Hog mine, New Mexico	Ore body 3 to 25 feet wide by 600 feet long in fault fissure.	3 months, 1930	11,146	Square-set-and-fill and timbered rill.	1.762	
Pecos mine, New Mexico	Irregular lenses in shear zone in schist.	1927–1929	194,030	Square-set-and-fill	2.66	Based on ore won by square-setting only.
Do	do	do	270,873	Cut-and-fill	2.11	Based on ore won by cut-and-fill stoping only.
Do	do	do	23,806	Shrinkage and subsequent fill	.93	Based on ore won by shrinkage-stoping only.
Morning mine, Idaho	Large fissure vein in quartzite, continuous to great depth.	1928	276,890	Stringer-set-and-fill	2.371	
Hecla and Star mines, Idaho	Several ore bodies in shear zone, dipping 70° to 80°.	do	312,942	Stringer set-and-fill principally.	1.211	

TABLE 11.—*Stoping costs in units of labor, explosives, timber, and power*

Mine and location	Type of deposit	Period covered	Ore hoisted, tons	Stoping method	Unit costs per ton of ore, stoping only: Man-hours	Explosives, pounds	Timber, board-feet	Power, kilowatt-hours
Mine No. 1, Tri-State district	Flat, bedded deposits	6 months, 1928	68,770	Open stopes	0.784	0.750	None	6.00
Mine No. 2, Tri-State district	do	9 months, 1928	87,563	do	.642	1.265	None	2.51
Mine No. 3, Tri-State district	do	1927	132,384	do	.401	.875	None	1.40
Waco mine, Tri-State district	do	May 1927 to January 1929.	364,285	do	.635	.594	None	
Barr mine, Tri-State district	do	January to April 1929.	45,344	do	.502	.805	None	2.16
Hartley-Grantham mine, Tri-State district.	do	January to June 1929.	60,282	do	.490	.741	None	
Hartley mine, Tri-State district	do	6 months, 1930	180,212	do	.460	1.120	None	2.00
No. 8 mine, Southeastern Missouri	Flat, bedded deposits, disseminated lead ore.	1928	168,089	do	.441	.565	None	
Mine X, Southeastern Missouri	do	Monthly average, 1929.	[1] 49,700	do	.673	.459	None	4.1
Mascot No. 2 mine, Tennessee	Large bedded deposits; dip, 18° to 22°.	10 months, 1929	[1] 511,383	Open stopes, mill-hole system	.316	.502	None	3.95
Edwards mine, N.Y	Lenticular masses in folded beds	1930	[1] 72,742	Open stopes	.928	(?)	(?)	21.16
Silver-King Coalition mine, Utah	Large, irregular bedded deposits in limestone.	1929	180,208	Square-set-and-fill	3.920	(?)	(?)	(?)
Park Utah mine, Utah	Fissures in quartzite and limestone	1928	192,250	Square-set-and-fill; some cut-and-fill.	1.930	(?)	(?)	(?)
Tintic Standard mine, Utah	Trough ore bodies, replacing limestone beds.	November 1929	8,758	Square-set-and-fill	4.694	.500	24.03	26.47
Pecos mine, New Mexico	Irregular lenses in shear zone in schist.	1929	[1] 195,685 [3] 217,121	Square-set-and-fill; cut-and-fill; pillars and shrinkage.	[2] 3.479 [3] 3.135	[2] .730 [3] .658	[2] 3.9 [3] 3.5	[2] 6.97 [3] 6.28
Morning mine, Idaho	Large fissure vein in quartzite	1928	276,890	Stringer-set-and-fill		.693		
Hecla and Star mines, Idaho	Several ore bodies in shear zone; dip, 70° to 80°.	do	312,942	Stringer-set-and-fill; horizontal stoping.	1.318	.533	[4] 1.234	12.66 (for the three Hecla and Star rows)
Do		do		Timbered shrinkage and horizontal stringer-set-and-fill.	1.394	1.174	[5] 11.207	
Do		do		Shrinkage, West ore body	1.410	2.054		
Block P mine, Montana	Narrow fissure; dip, 65° to 88°	1929	106,242	Cut-and-fill	3.956	2.568		
Gilman, Colo	Replacement ore bodies in limestone beds.			Average of 11 square-set stopes	1.724	.67	13.9	
Do	do			Average of nine cut-and-fill stopes	1.250	.53	7.3	
Do	do			Open stope	.879	.38	3.5	
Do	do			Underhand pillar stopes	1.250	.41	6.7	

[1] Stope ore only.
[2] Based on tonnage of ore from stopes only; timber, linear feet.
[3] Based on total ore hoisted; timber, linear feet.
[4] Stulls; linear feet.
[5] Sawed timber.

COST OF MINING

The methods and cost of mine development and stoping have been discussed in the foregoing sections of this paper. As stated the tabulated stoping costs did not include development costs, charges for underground transportation, or general underground and surface expense chargeable to mining, although such charges are inseparable from the cost of mining and constitute an appreciable percentage of total ore-production cost.

This section presents some typical mining costs for a group of lead and zinc mines, including all direct underground mining costs and distributive items properly chargeable to mining. The characteristics of the ore deposits, rate of production, mining method, and other factors are reflected in the costs at each property, and because of the wide range of conditions encountered costs of development and stoping vary greatly at different properties. The same factors affect the total mining costs; and when development, stoping, transportation, and general charges are combined costs at different mines usually vary even more widely. An exception to this may be found where, for instance, one of two properties has a high development and a low stoping cost and the other a low development and a high stoping cost, or there may be compensating factors in other cost items. More often, however, the factors which cause high costs in one department of underground operation cause high costs in other departments.

Due to differences in accounting practices at various mines it is difficult to distribute costs over the several items in table 12 on exactly the same basis, but the figures presented are believed to be substantially on the same basis. Some mines, however, have included certain surface and overhead charges which have not been included in the majority of cases. Direct comparison of the costs at the different mines is therefore difficult.

In columns 5 to 10 the costs are distributed between development (which in some instances includes exploration costs), stoping, transportation (underground tramming, haulage, and hoisting), general underground expense, and surface expense applicable to mining. In columns 11 to 17 costs are distributed between labor and supervision; compressed air, drills, and steel; power; explosives; timber; and other supplies.

Table 13 gives the tonnage of ore produced per man-shift underground, on the surface, and by total mine labor in 1930 at 148 mines which produced 16,753,724 tons of ore during that year. Total labor includes all labor directly chargeable to mining but does not include mill labor and general yard and plant labor on the surface, which is not directly chargeable against mining cost.

TABLE 12.—*Mining costs per ton of ore hoisted*

Mine and location	Period covered	Ore produced during period, tons	Stoping method	Costs of mining ore per ton												
				Development and exploration	Stoping	Transportation	General underground expense	General surface expense	Total	Labor and supervision	Compressed air, drills, and steel	Power	Explosives	Timber	Other supplies	Total
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)	(13)	(14)	(15)	(16)	(17)
Mine No. 1, Tri-State district.	6 months, 1928	68, 770	Open stopes	$0. 049	$0. 532	$0. 217	$0. 187		$0. 985	$0. 624	$0. 127	$0. 041	$0. 110		$0. 083	$0. 985
Mine No. 2, Tri-State district.	9 months, 1928	87, 563	do	. 050	. 458	. 200	$0. 165	$0. 120	. 993	. 576	. 114	. 076	. 157		. 070	. 993
Mine No. 3, Tri-State district.	1927	132, 384	do	. 030	. 450	. 165	. 162	. 194	1. 001	. 675	. 075	. 102	. 117		. 032	1. 001
Waco mine, Tri-State district.	May 1927 to January 1929.	364, 285	do	. 050	. 490	. 155	. 057	. 033	. 785	. 536	. 068	. 109	. 072			. 785
Barr mine, Tri-State district.	January to April 1929.	45, 344	do	. 047	. 497	. 217	. 069	. 032	. 862	. 529	. 066	. 048	. 103		. 116	. 862
Hartley-Grantham mine, Tri-State district.	January to June 1929.	60, 282	do	None	. 516	. 197	. 049	. 020	. 782	. 535	. 052	. 037	. 106		. 051	. 781
Hartley mine, Tri-State district.	6 months, 1930	180, 212	do	None	. 572	. 152	. 063	. 041	. 828	. 507	. 078	. 060	. 150		. 033	. 828
No. 8 mine, Southeastern Missouri district.	1928	168, 089	do	. 045	. 483	. 222	0. 136		. 886	. 687	. 081	. 046	. 072			. 886
Mascot No. 2 mine, Tennessee.	10 months, 1929	528, 626	Open stopes; mill-hole system.	. 053	. 222	. 278	. 086		. 639	. 454	. 047	. 016	. 060		. 062	. 639
Bunker Hill & Sullivan mine, Idaho.	1928	452, 345	Square-set-and-fill	[1]. 053	[2]2.231	[3]. 381	1. 037	0. 195	3. 897	1. 934	. 229	([4])	. 167	$0. 321	1. 246	3. 897
Do	1931	460, 366	do	[5]. 066	[6]2.476	[7]. 308	[8]0. 604		3. 454	1. 875	. 204	. 113	. 165	. 331	. 766	3. 454
Page mine, Idaho	1928	90, 460	do	1. 170	2. 150	. 160	0. 180	0. 160	3. 820	2. 690	. 280	. 120	. 240	. 380	. 110	3. 820
Silver King Coalition mines, Utah.	1929	180, 208	do	2. 086	3. 502	1. 848	. 309		7. 745	5. 418	. 854	. 370	. 486	. 340	. 277	7. 745
Park Utah mine, Utah	1928	192, 250	Square-set-and-fill; some cut-and-fill.	. 828	2. 426	. 282	. 583	. 543	4. 662	3. 062	. 139	. 191	. 227	. 493	. 550	4. 662
Tintic Standard mine, Utah.	November 1929	8, 758	Square-set-and-fill	1. 143	4. 973	1. 158	. 322	[9]1. 888	9. 484	6. 356	. 770	. 261	. 224	. 913	. 960	[10]9. 484
Ground Hog mine, New Mexico.	3 months, 1930	11, 146	Square-set-and-fill and timbered rill.	1. 393	1. 762	. 628	. 080	. 127	3. 990	2. 793	. 169	. 179	. 187	. 318	. 344	3. 990
Pecos mine, New Mexico	1927–29	578, 658	Square-set and cut-and-fill.	[11]1.005	2. 278	. 663	. 423	. 010	4. 379	[12]3.308	[12]. 141	[12]. 091	[12]. 259	[12]. 422	[12]. 677	[12]4. 898

Morning mine, Idaho	1928	276,890	Stringer-set-and-fill	.442	2.371	.530	.698	.204	4.245	3.088	.122	.211	.158	.417	.249	4.245
Hecla and Star mines, Idaho.	do	312,942	do	.638	1.211	.322	.766	.360	3.297	1.868	.070	.145	.112	.357	.317	[13] 2.869

[1] Difference between mining, reported as $2.414, and stoping, reported as $2.361. Probably represents exploration only.
[2] Reported as $2.361; deducted $0.130 for carmen, trammers, and hoistmen. Probably includes some development.
[3] Reported as $0.251; $0.130 added for carmen, trammers, and hoistmen reported under stoping.
[4] Included in other items.
[5] Reported as exploration.
[6] Reported as $3.128; deducted $0.134 for carmen, trammers, and hoistmen and $0.518 distributed charges. Probably includes some development.
[7] Reported as $0.174; $0.134 added for carmen, trammers, and hoistmen.
[8] $0.518 distributed charges reported under stoping plus pumping cost.
[9] Includes a number of charges probably not included in costs given for most of the other mines such as unloading, yard maintenance, loading, and shipping, $0.218; building maintenance, $0.248; heat, water, and lighting, $0.261.
[10] Includes some surface charges probably not included in costs given for other mines listed, such as unloading, yard maintenance, loading, and shipping, $0.218; building maintenance, $0.248; heat, water, and light, $0.261.
[11] Includes $0.146 for diamond drilling.
[12] Year 1929 only and on 217,121 tons of ore.
[13] Development in waste $0.428 not included, working total $3.297.

TABLE 13.—*Ore produced per man-shift in 1930 at lead and zinc mines, tons*

Number of mines reporting	Ore produced, tons	Stoping method employed	Man-shifts worked			Production per man-shift, tons		
			Underground	Surface	Total	Underground	Surface	Total
102	13, 287, 051	Open stopes	1, 676, 562	226, 780	1, 903, 342	7. 925	58. 6	6. 98
26	1, 716, 695	Square-sets	1, 106, 917	208, 886	1, 315, 803	1. 551	8. 2	1. 30
13	1, 098, 823	Cut-and-fill and stringer-set-and fill.	450, 079	120, 965	571, 044	2. 441	9. 1	1. 92
7	651, 155	Shrinkage	184, 554	57, 155	241, 709	3. 528	11. 4	2. 69
148	16, 753, 724		3, 418, 112	613, 786	4, 031, 898	4. 902	22. 3	4. 15

Table 14 presents costs of mining in units of labor, materials, and power. Such costs do not reflect the differences in wage rates and costs of materials prevailing in different districts which affect the dollar costs, and they may be used as a rough measure of the relative productivity of labor and of the operating efficiency under various conditions and with different mining methods.

TABLE 14.—*Mining costs in units of labor, materials, and power*

Mine and location	Period covered	Ore produced during period, tons	Stoping method	Unit costs of ore produced per ton					
				Labor, man-hours			Explosives, pounds	Power, kw.-hr.	Timber
				Development	Stoping only [1]	Total [2]			
Mine No. 1, Tri-State district	6 months, 1928	68,770	Open stopes	0.784		1.341	0.750	7.43	None.
Mine No. 2, Tri-State district	9 months, 1928	87,563	do	.642		.876	1.265	5.75	None.
Mine No. 3, Tri-State district	1927	132,384	do	.401		.905	.875	4.17	None.
Waco mine, Tri-State district	May 1927 to January 1929.	364,285	do	.635		1.041	.594	4.99	None.
Barr mine, Tri-State district	January to April 1929	45,344	do	.502		.888	.805	6.09	None.
Hartley-Grantham mine, Tri-State district.	January to June 1929	60,282	do	.490		.745	.741		None.
Hartley mine, Tri-State district	6 months, 1930	180,212	do	.460		.673	1.120	4.25	None.
No. 8 mine, southeastern Missouri	1928	168,089	do	.490		.875	.634	4.56	None.
Mine X, southeastern Missouri	Month, average 1929	[3] 56,000	do	0.135	0.650	.827	.550	10.30	None.
Mascot No. 2 mine, Tennessee	10 months, 1929	528,626	do	.316		.831	.502	9.68	None.
Edwards mine, New York	1930	[4] 72,742	do	.053	.928	1.503		61.09	
Silver King Coalition mines, Utah	1929	180,208	Square-set-and-fill			8.055	3.607	77.70	
Park Utah mine, Utah	1928	192,250	Square-set-and-fill; some cut-and-fill.	.580	1.930	3.540	1.530	29.23	16.46 board-feet.
Tintic Standard mine, Utah	November 1929	8,758	Square-set-and-fill	1.038	4.694	7.225	1.068	46.60	25.50 board-feet.
Pecos mine, New Mexico	1929	[5] 217,121	Square-set-and-fill; cut-and-fill, pillars, and shrinkage.	[6] 1.463	[6] 3.135	[6] 5.674	[6] 1.338	[6] 13.93	4.7 linear feet.[6]
Morning mine, Idaho	1928	276,890	Stringer-set-and-fill			4.169	.964	39.37	12.75 board-feet.
Hecla and Star mines, Idaho	1928	312,942	do			2.029	.783	17.01	1.234 linear feet plus 11.21 board-feet.
Block P mine, Montana	1929	106,242	Cut-and-fill	.284	3.956	[7] 5.350 [8] .469 5.819	2.854	17.46	7.538 linear feet plus 3.955 board-feet.

[1] Does not include tramming, haulage, hoisting, and general expense.
[2] Includes tramming, haulage, hoisting, and general expense.
[3] Includes 2,000 tons of waste rock.
[4] Not including ore from development.
[5] Includes stope ore, 195,685 tons, and development ore, 21,436 tons.
[6] Based on total tonnage.
[7] Underground.
[8] Surface.

Part 3.—METHODS OF CONCENTRATING LEAD AND ZINC ORES

INTRODUCTION

The Bureau of Mines has prepared and published a number of articles dealing with milling methods and costs at metal mines in the United States. Most of these reports have been written by engineers of the various mining companies in accordance with an outline prepared by engineers of the Bureau, so that the information and operating data presented are actual results obtained when the reports were written.

The object of this part of the bulletin is to summarize milling methods and costs presented in the published articles covering lead and zinc mines in different parts of the United States. Although the list of concentrators represented in this report is incomplete, enough have been included to give a general picture of concentrating methods for lead and zinc ores.

Many changes during the past 15 or 20 years in the methods of concentration have resulted in marked improvement in recoveries of the economic minerals. Since the use of the flotation process has been adopted commercially at most concentrators, much milling equipment previously used has been discarded. Huntington and Chilean mills, and in many cases rolls, have been replaced by rod or ball mills; round tables and vanners have been replaced by flotation machines for the treatment of slimes; screening and classification have been greatly improved; and differential flotation has permitted separation of finely disseminated lead-zinc ores that formerly were not amenable to gravity methods. This paper does not propose to compare the results obtained by earlier methods with those of today, but rather to indicate actual results that are being obtained by present improved methods and equipment.

To bring out the more important points of recent milling practice in the concentration of lead and zinc ores a number of concentrators have been selected, and their flow sheets are presented. In discussion of the method at any particular plant the concentrators are referred to in the text by letter.

ACKNOWLEDGMENTS

The writers are indebted to the engineers of the mining companies who wrote the information circulars published by the Bureau of Mines covering milling practice at different concentrating plants; the texts thereof have been used freely in preparing this report.

BRIEF SUMMARY AND FLOW SHEETS OF MILLING PRACTICE AT VARIOUS CONCENTRATORS

CONCENTRATOR A—EAGLE-PICHER LEAD CO., NETTA MILL, PICHER, OKLA.[1]

Flow sheet.—Figure 43.

Ore treated.—Economic minerals—galena and sphalerite, which occur principally in large crystals attached to gangue but are also found finely disseminated in the flint and black rock; this necessitates fine grinding to liberate them. Pyrite usually present and occasionally chalcopyrite. Gangue—chiefly flint or chert with some limestone. Considerable black flint or jasperoid usually present; calcite, fairly plentiful. Lead sulphide in ore, 1 to 5 percent; zinc sulphide, 3 to 10 percent.

Capacity of concentrator.—120 tons per hour. In 1930, 60 tons per hour treated. Practice in the district to operate mills one 10-hour shift per day.

Milling method.—Gravity concentration and flotation. Gravity concentration accomplished by roughing and cleaning system in jigs, and fine sands treated on tables. Flotation feed (fines obtained by drag and cone classifiers from gravity concentration) collected in Dorr thickeners. Pulp treated by differential flotation, and products graded by roughing and cleaning system.

Grade of products.—Average grade of zinc concentrates for 1930:

Method	Assays, percent			
	Zn	Pb	Fe	CaO
Gravity concentration	59.12	1.97	1.97	1.04
Flotation	59.72	1.90	2.02	.93

Recoveries.—Total recovery of lead and zinc, 86.23 percent.

CONCENTRATOR B—CANAM METALS CORPORATION, WHITE BIRD CONCENTRATOR, PICHER, OKLA.[2]

Flow sheet.—Figure 44.

Ore treated.—Economic minerals—sphalerite and galena disseminated through jasperoid breccia and filling interstices between blue and white chert boulders. Gangue minerals—blue, white, and gray chert, jasperoid, calcite, dolomite, and small amounts of marcasite. Galena easily freed; most of the sphalerite freed when crushed to 48 mesh.

Capacity of concentrator.—40 tons per hour. During August, September, and October 1929, 24,974 tons treated in 80 operating days of one 10-hour shift per day, or 31.2 tons per hour.

Milling method.—Gravity concentration and flotation. Gravity concentration accomplished by a roughing and cleaning system of jigging; fine sands treated on tables. Feed to flotation made up of slimes from cone and drag classifiers from gravity concentration, collected in Dorr thickeners. Final flotation concentrates taken off direct without previous roughing. Only zinc concentrates produced by flotation.

Period covered.—August, September, and October 1929.

Grade of products.—Average grade of concentrates:

Concentrates	Assays, percent		
	Zn	Pb	Fe
Lead	----------	74	----------
Gravity zinc	61.50	.36	1.07
Flotation zinc	59.80	.07	1.47

[1] Sansom, F. W., Milling Practice at the Netta Mine of the Eagle-Picher Lead Co., at Picher, Okla.: Inf. Circ. 6342, Bureau of Mines, 1930, 13 pp.

[2] Crabtree, E. H., Jr., Milling Practice at the White Bird Concentrator, Canam Metals Corporation, Picher, Okla.: Inf. Circ. 6353, Bureau of Mines, 1930, 10 pp.

Recoveries.—	*Percent*
Lead recovered by jigs	85.6
Zinc recovered by jigs	36.8
Zinc recovered by tables	21.5
Zinc recovered by flotation	26.0
Total zinc recovered	84.3

Ratio of concentration.—Lead concentrates, 51:1; zinc concentrates, 11.4:1; total concentrates, 11.1:1.

Net water consumption.—3,840 gallons per ton of ore treated, or about 16 tons of water per ton of ore. Some water reclaimed from tailings for reuse in mill.

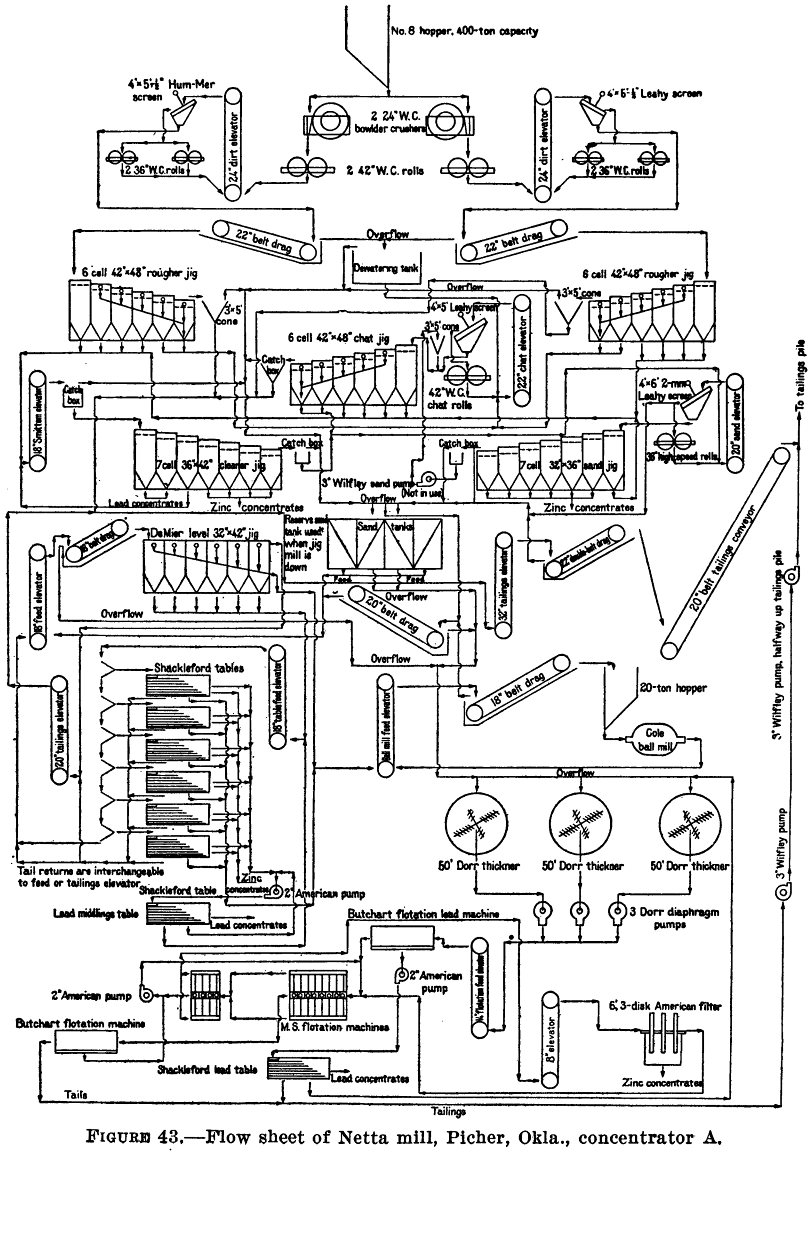

FIGURE 43.—Flow sheet of Netta mill, Picher, Okla., concentrator A.

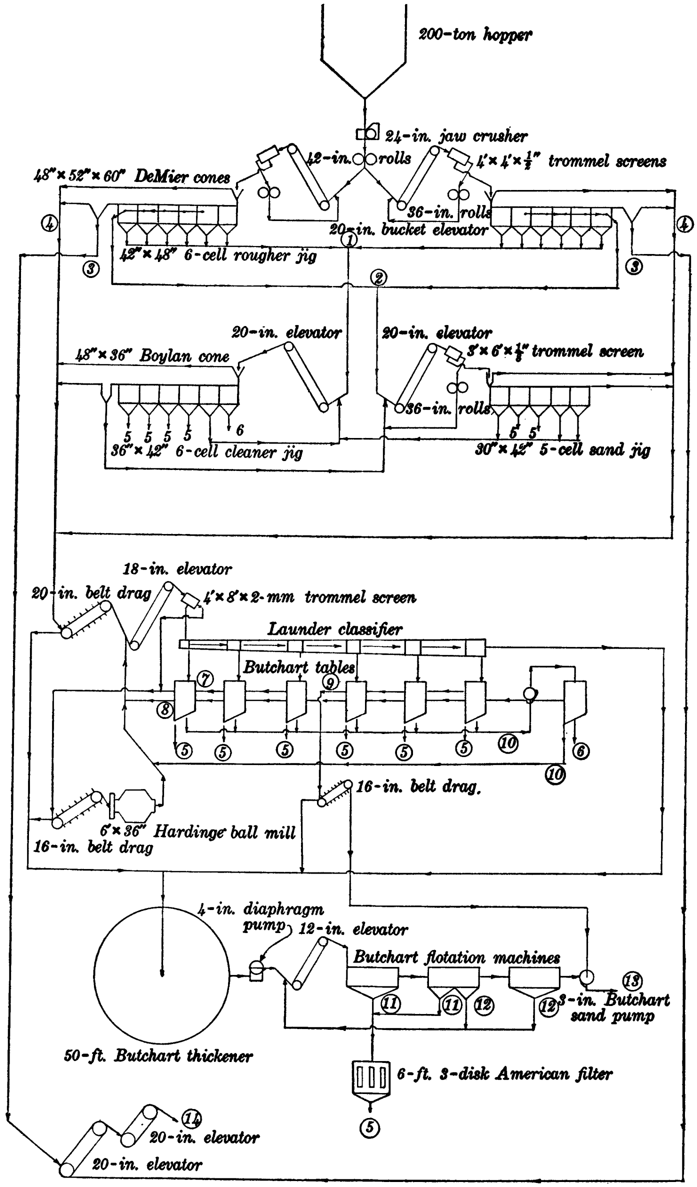

FIGURE 44.—Flow sheet of White Bird concentrator, Picher, Okla., concentrator B: *1*, Hutch product; *2*, chat product; *3*, rougher jig tailings; *4*, sand and slime; *5*, shipping zinc concentrates; *6*, shipping lead concentrates; *7*, table tails (to ball mill); *8*, table middlings; *9*, table tails (to waste); *10*, table lead-zinc middlings; *11*, flotation zinc concentrates; *12*, flotation middling froth; *13*, table and flotation tailings to waste pond; *14*, rougher jig tailings to tailings pile.

CONCENTRATOR C—BUNKER HILL & SULLIVAN MINING & CONCENTRATING CO., WEST MILL, KELLOGG, IDAHO [3]

Flow sheets.—Figure 45.

Ore treated.—Economic minerals—silver-bearing galena and sphalerite. Part of the galena in large masses; most of it included in siderite and quartz in smaller masses, with some microscopic inclusions. Ore fairly easy to crush, but in fine grinding the hardness of quartzite becomes apparent. Approximate mineralogical analysis of ore: Galena, 12.6 percent; sphalerite, 2.0 percent; pyrite, 3.6 percent; siderite, 32.6 percent; quartz, 44.9 percent; and silver, 4.0 ounces. Mill feed contains about 1.7 percent Mn, corresponding possibly to 2.7 percent rhodochrosite.

Capacity of concentrator.—1,300 tons per 24 hours. During 1929, 451,111 tons were milled in 353.66 days of 24 hours, about 53 tons per hour.

Period covered.—January to December 1929.

Milling methods.—Gravity concentration and flotation. Gravity concentration accomplished by treating sized material in jigs and classified fine sands on tables. Lead concentrates only produced by gravity methods. Flotation divided into one circuit treating primary slimes separated from crushed mill feed and producing lead concentrates and another circuit treating ground middlings and tailings from gravity concentration by differential flotation, producing both lead and zinc concentrates.

Grade of products.—

Concentrates	Assays, percent		Assays, ounces
	Pb	Zn	Ag
Lead	58.19	3.13	24.11
Zinc	3.77	47.37	5.9

Recoveries, percent.—

Concentrates	Pb	Ag	Zn
Lead	89.12	89.70	32.15
Zinc	.28	1.05	23.29

Ratio of concentration.—Lead concentrates, 6.27:1; zinc concentrates, 131:1; total concentrates, 5.98:1.

Water consumption.—Total water used in mill, 4,200 gallons per minute, or about 20 tons per ton of ore treated, 52.4 percent being reclaimed water.

CONCENTRATOR D—HECLA MINING CO., CRUSHING, SORTING, AND TREATMENT PLANTS, GEM, IDAHO [4]

Flow sheets.—Figures 46 and 47.

Ore treated.—Economic minerals—galena and sphalerite. Gangue minerals—principally quartzite, siderite, and small amounts of pyrite. Average grade of milling ore for 1930: Pb, 8.7 percent; Zn, 1.2 percent; Fe, 7.1 percent; Ag, 4.77 ounces; and insoluble, 68.8 percent.

Capacity of concentrator.—900 tons per day, or 37.5 tons per hour; 235,413 tons of mill feed treated during 1930.

Period covered.—Year 1930.

[3] Handy, R. S., Milling Methods and Costs at the Northern Idaho Mills of the Bunker Hill & Sullivan Mining & Concentrating Co.; Inf. Circ. 6314, Bureau of Mines, 1930, 53 pp.

[4] Ziegler, W. L., Milling Methods and Costs at the Lead Concentrator of the Hecla Mining Co., Gem, Idaho: Inf. Circ. 6600, Bureau of Mines, 1932, 16 pp.

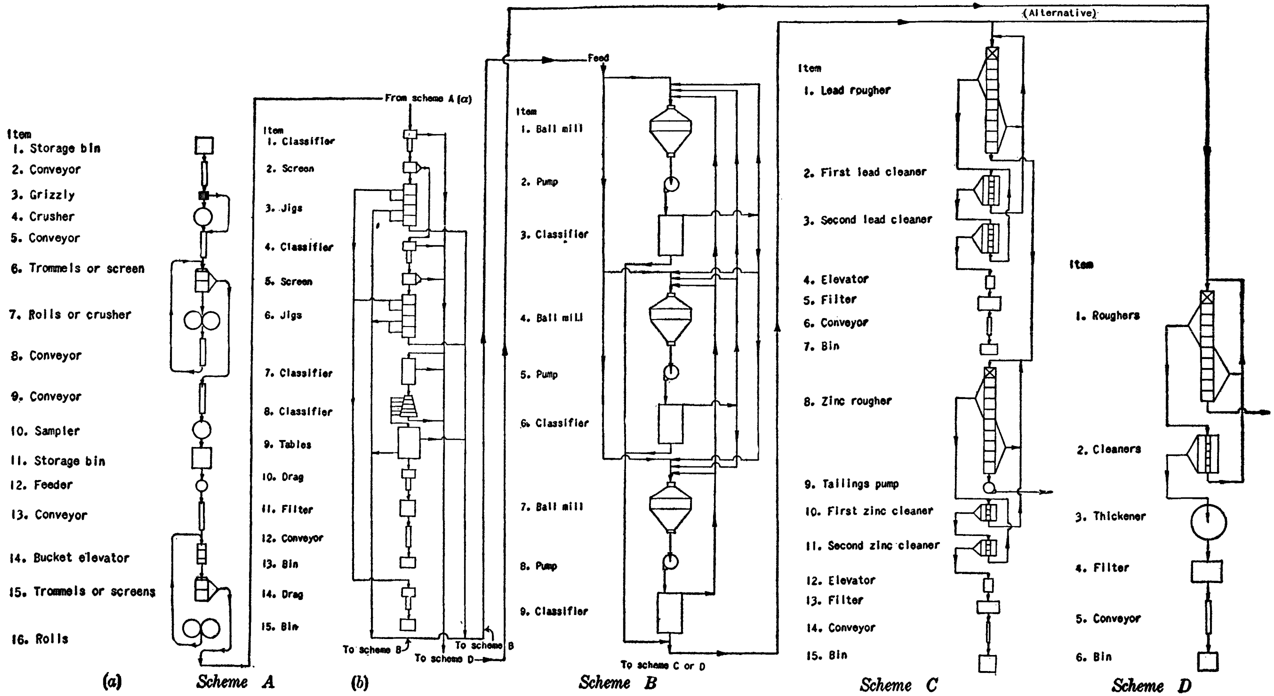

FIGURE 45.—Flow sheets of ore treatment at Bunker Hill & Sullivan mills, concentrator C: Scheme *A* (*a*) reduction of ore, (*b*) gravity flow sheet, one section of West mill; scheme *B*, grinding section; scheme *C*, differential flotation of lead and zinc; scheme *D*, simple flotation.

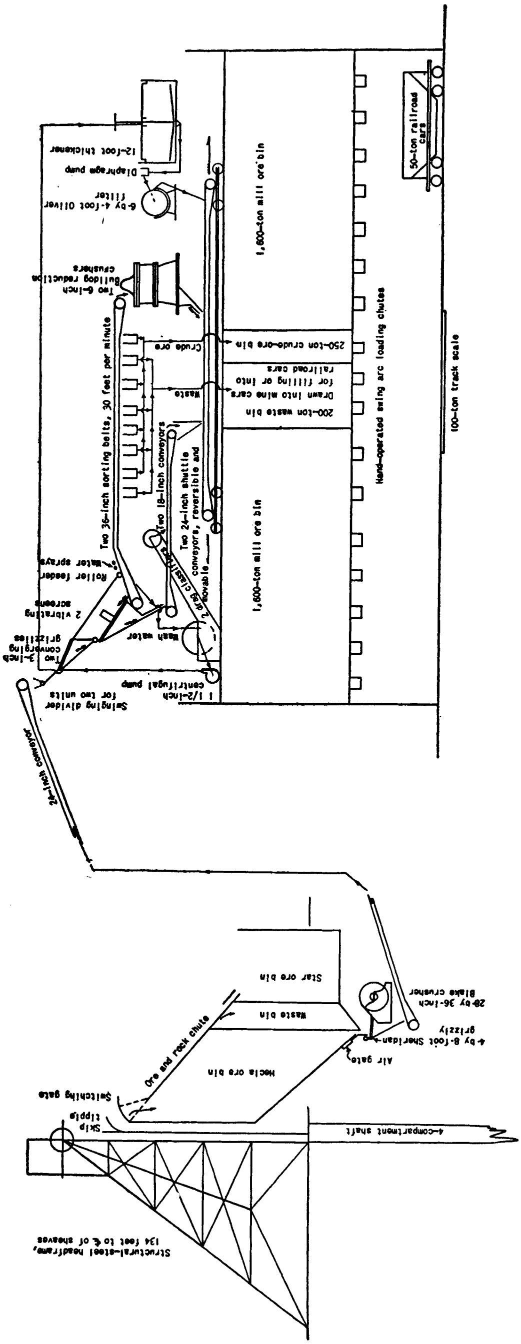

FIGURE 46.—Flow sheet of crushing and sorting plant, Hecla Mining Co., Gem, Idaho, concentrator D.

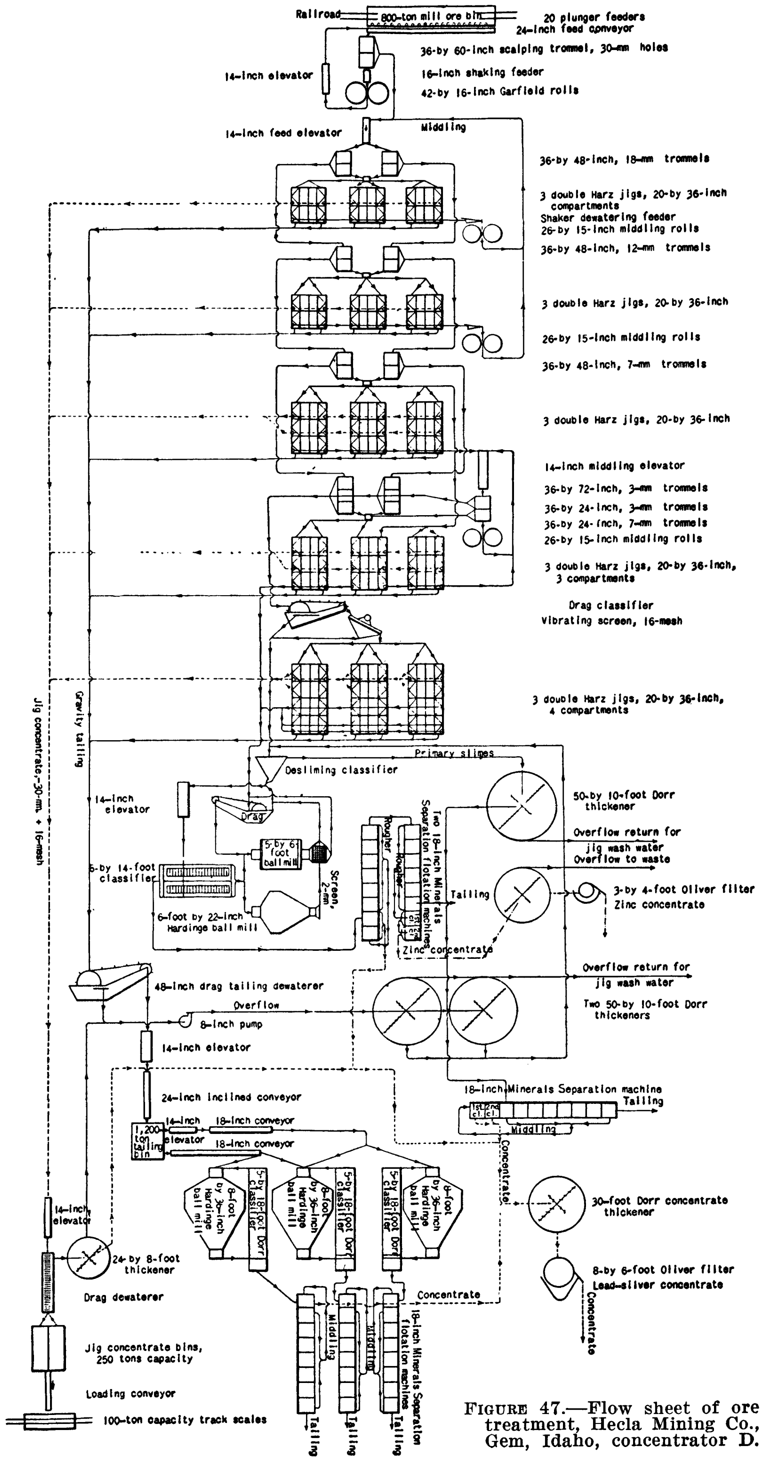

FIGURE 47.—Flow sheet of ore treatment, Hecla Mining Co., Gem, Idaho, concentrator D.

Milling methods.—Crude-lead shipping product and waste rock sorted on picking belt. Gravity concentration limited to jigging of mill feed coarser than 16-mesh material after grading into several sizes. Lead concentrate only produced by jigs. Flotation treatment divided into 3 separate flotation units: (1) Flotation of primary slimes, producing a lead concentrate only; (2) grinding and differential flotation of minus 16-mesh original mill feed and jig middlings, resulting in the production of lead and zinc concentrates; and (3) grinding and flotation of jig tailings, producing lead concentrates.

Grade of products.—

	Assays, percent		Assays, ounces
	Pb	Zn	Ag
Jig concentrates	52.02	3.5	27.09
Flotation:			
Lead concentrates	61.3	5.0	34.6
Zinc concentrates	9.3	46.7	10.1
Flotation of jig tailings: Lead concentrates	46.0	4.0	31.4

Recoveries.—Percent, based on total metal in original mill feed:

	Pb	Ag	Zn
Jig concentrates	65.8	62.5	40.1
Flotation:			
Lead concentrates	25.9	26.9	16.6
Zinc concentrates	.7	1.2	27.7
Flotation of jig tailings: Lead concentrates	6.0	7.2	4.0
Total	97.8	98.4	88.4

Ratio of concentration.—Lead concentrates, 6.3:1; zinc concentrates, 152:1; total concentrates, 6.1:1.

CONCENTRATOR E—AMERICAN ZINC CO., MASCOT, TENN.[5]

Flow sheets.—Figure 48.

Ore treated.—Economic mineral—sphalerite, associated with dolomite, calcite, quartz, and pyrite. Analysis of ore: Calcium carbonate, 48.11 percent; magnesium carbonate, 35.36 percent; silica, 8.73 percent; alumina, 1.04 percent; iron oxide, 1.33 percent; zinc sulphide, 5.43 percent. During 1929 mill heads averaged 2.9 percent zinc.

Capacity of mill.—1,900 tons per day, about 79 tons per hour. During 1929, 624,759 tons milled.

Period covered.—Year 1929.

Milling method.—Gravity concentration and flotation. For gravity concentration jigs only used. Roughing and cleaning system of jigging practiced, tailing from rougher jigs receiving further roughing on bull jigs. Tailing from bull jigs, about 50 percent of tonnage milled, discarded. Flotation feed, slimes obtained from classifier overflows and reground jig middlings; thickened pulp treated by roughing and cleaning system.

Grade of product.—Average grade of zinc concentrates, 60.19 percent zinc.

Recoveries.—Zinc recovery for 1929, 90.15 percent.

CONCENTRATOR F—FLAT RIVER, MO.[6]

Flow sheets.—Figure 49.

Ore treated.—Economic mineral—galena, with small amounts of pyrite, chalcopyrite, and sphalerite. Gangue—chiefly dolomitic limestone (hard, dense

[5] Strachan, C. B., Milling Methods of the American Zinc Co. of Tennessee, Mascot, Tenn.: Inf. Circ. 6379, Bureau of Mines, 1930, 12 pp.

[6] Coghill, W. H., and O'Meara, R. G. Milling Methods and Costs at a Flat River (Mo.) Mill: Inf. Circ. 6658, Bureau of Mines, 1932, 36 pp.

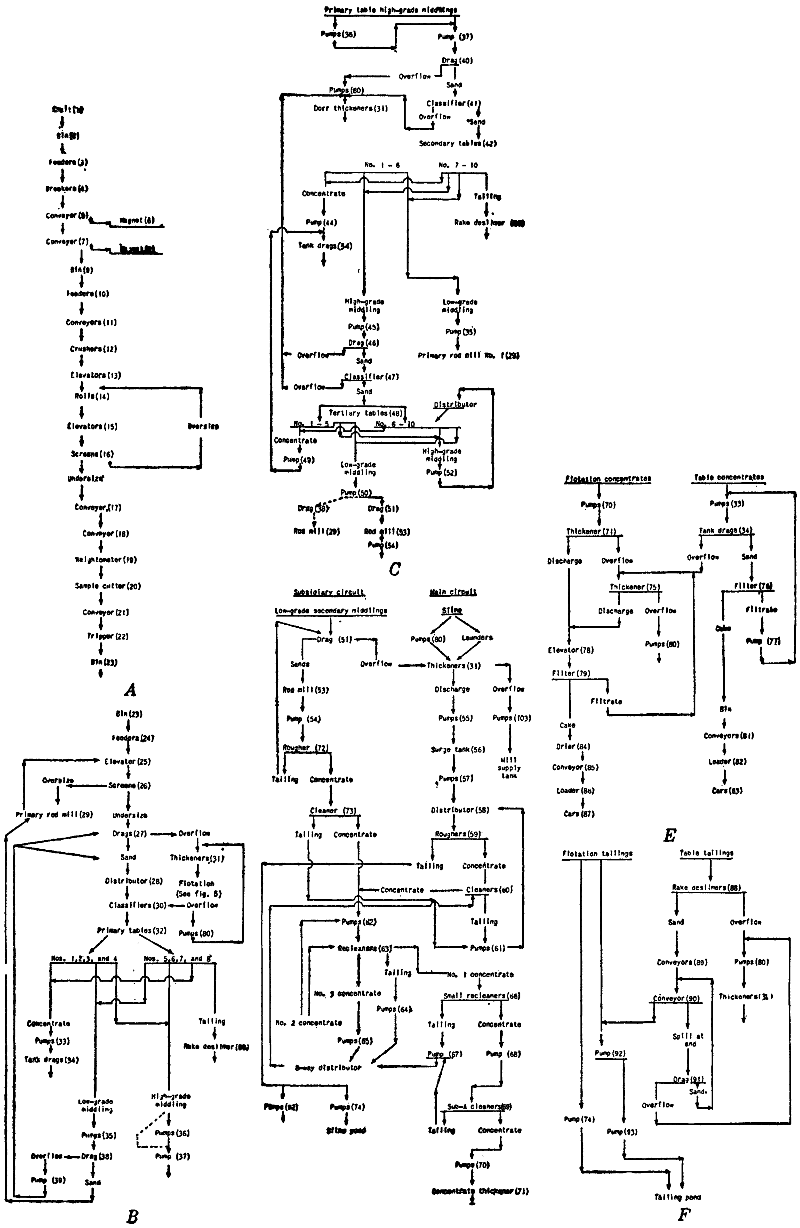

FIGURE 49.—Flow sheets of various plant operations at a Flat River (Mo.) mill, concentrator F: *A*, Dry crushing; *B*, one section of primary wet grinding and gravity concentration; *C*, table-middling section; *E*, concentrate dewatering; *F*, tailing disposal.

variety, fairly resistant to crushing), with small amounts of siliceous minerals. No gangue entirely free from galena. Chemical analysis of ore: Lead, 3.5 percent; silicia, 3.2 percent; alumina, 1.3 percent; iron oxide, 6.6 percent; lime, 27.4 percent; magnesia, 15.5 percent; manganese oxide, 0.7 percent; sulphur, 0.8 percent; copper, nickel, and cobalt, 0.1 percent; zinc, 0.2 percent; carbon dioxide (by difference and undetermined), 40.7 percent.

Capacity of concentrator.—5,000 tons per day, or 208 tons per hour.

Period covered.—1930 and first 5 months of 1931.

Milling method.—Gravity concentration and flotation. All gravity concentration on concentrating tables. Mill feed crushed to pass a 7-mm vibrating screen and, after desliming, fed to 10-spigot hydraulic classifiers of hindered-settling type; spigot discharges fed to tables. High-grade middlings treated on separate group of tables without regrinding; low-grade middlings reground. Tailings from tables treating finer spigot products of primary sections and of high-grade middling circuit discarded. Feed to flotation circuit made up of overflows from hydraulic classifiers and desliming drag classifiers collected in Door thickeners. Flotation roughing and cleaning system practiced. About two thirds of total mill feed eventually treated by flotation.

Grade of products.—

Concentrates	Lead assays, percent	
	1930	1931 (5 months)
Table	76.52	76.03
Flotation	71.36	69.42
Total	74.30	72.84

Recoveries.—Total lead recovery, 96 to 97 percent. About 52 percent of total lead concentrates produced by gravity concentration.

Water consumption.—Ratio of water to ore, about 12:1 by weight.

CONCENTRATOR G—FEDERAL MINING & SMELTING CO., PAGE CONCENTRATOR, KELLOGG, IDAHO [7]

Flow sheet.—Figure 50.

Ore treated.—Economic minerals—galena and sphalerite. Ore consists of intimate mixtures of lead and zinc sulphides in quartzite gangue. Principal accessory minerals—pyrite, siderite, and calcite. Ore distinctly sulphide in character and said to show only slight tendency to oxidize after breaking. Owing to the intimate associations of some minerals fine grinding required for treatment of ore.

Capacity of concentrator.—300 tons per day, or 12.5 tons per hour. Tonnage treated during 1930, 97,447 tons, and during first 6 months in 1931, 30,421 tons.

Period covered.—1930 and first 6 months of 1931.

Milling method.—All-flotation treatment. Ore crushed to pass ¾-inch vibrating screen. Product then ground in ball mills, which are in closed circuit with Dorr and Esperanza classifiers. Classifier overflow comprises flotation feed. Roughing and cleaning system used in both lead and zinc circuits; tailings from the lead circuit comprise the feed to the zinc circuit.

Grade of products.—

Concentrates	1930 assays			1931 (6 months) assays		
	Pb, percent	Ag, ounces	Zn, percent	Pb, percent	Ag, ounces	Zn, percent
Lead	68.1	30.7	7.9	70.8	27.2	6.9
Zinc	3.8	5.6	51.6	3.9	5.3	51.3

[7] Price, G. S., Milling Methods and Costs at the Page Concentrator of the Federal Mining & Smelting Co., Kellogg, Idaho: Inf. Circ. 6590, Bureau of Mines, 1932, 6 pp.

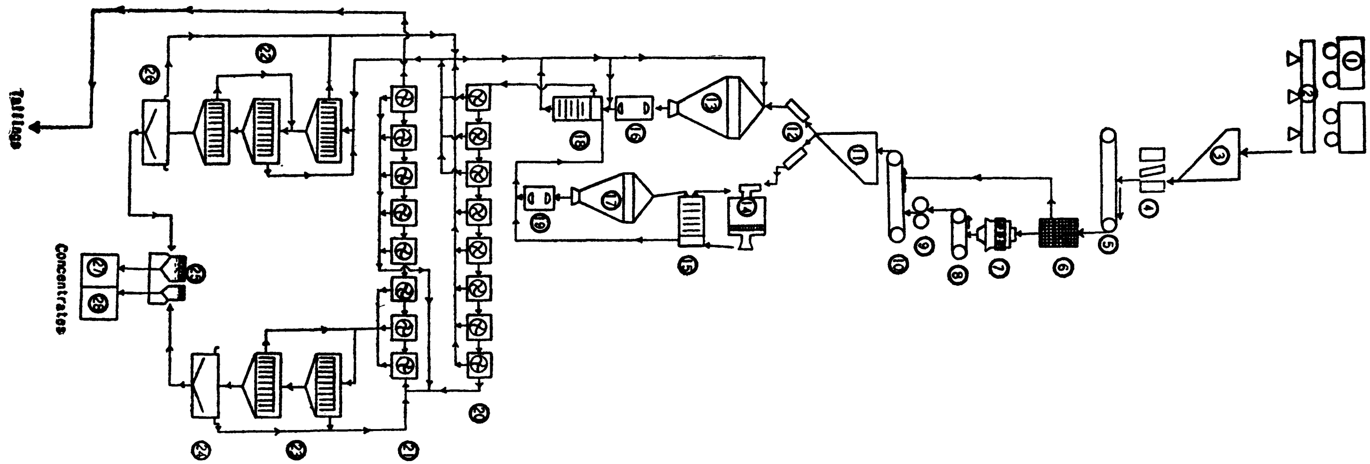

FIGURE 50.—General flow sheet of Page concentrator, Federal Mining & Smelting Co., Kellogg, Idaho, concentrator G: 1, Ore from mine by electric trolley; 2, track scales; 3, coarse-ore bin, 150-ton capacity; 4, Blake jaw crusher, 9 by 15 inch; 5, conveyor, 24 inch; 6, Traylor vibrator screen; 7 Symons cone crusher, 4 foot; 8, conveyor, 24 inch; 9, rolls, 14 by 36 inch; 10, conveyor, 24 inch; 11, fine-ore bin, 200-ton capacity; 12, feeders, cam and spring type; 13, Hardinge ball mill, 8 foot by 36 inch; 14, Marcy ball mill, no. 64½; 15, classifier, 6-foot Dorr type; 16, elevator; 17, Hardinge ball mill, 6 foot by 16 inch; 18, Esperanza classifier; 19, elevator; 20, 2 Minerals Separation Sub-A machines, 8 cell; 21, Minerals Separation Sub-A machine, 8 cell; 22, cleaners for lead circuit, 3 pneumatic-type cells in series—first 10 foot, others 5 foot; 23, cleaners for zinc circuit, two 5-foot pneumatic cells in series; 24, Dorr 20-foot thickener for zinc unit; 25, duplex Oliver filter, 5 foot for lead unit and 3 foot for zinc unit; 26, Dorr 20-foot thickener for lead unit; 27, lead-concentrate bin, 200-ton capacity; 28, zinc-concentrate bin, 160-ton capacity.

Recoveries, percent.—

Concentrates	1930			1931 (6 months)		
	Pb	Ag	Zn	Pb	Ag	Zn
Lead	90.6	87.8	40.2	91.5	88.9	35.1
Zinc	.8	2.7	44.6	1.0	3.7	55.0

Ratio of concentration.—

Concentrates	1930	1931 (6 months)
Lead	7.95:1	6.26:1
Zinc	47:1	29.9:1
Total	6.8:1	5.18:1

CONCENTRATOR H—FEDERAL MINING & SMELTING CO., MORNING CONCENTRATOR, MULLAN, IDAHO [8]

Flow sheet.—Figure 51.

Ore treated.—Economic minerals—galena and sphalerite. Ore complex, consisting of intimate mixture of lead and zinc sulphides containing silver. Gangue minerals—chiefly siderite and quartz. Fine grinding necessary to liberate sulphide minerals. Average tenor of ore milled during 1930: Pb, 9.0 percent; Ag, 3.4 ounces; and Zn, 6.5 percent.

Capacity of concentrator.—1,200 tons per day, or 50 tons per hour. Ore milled during 1930, 361,372 tons.

Period covered.—Year 1930.

Milling method.—All-flotation treatment. All ore crushed to pass 1-inch vibrating screen, then ground in ball mills in closed circuit with drag classifiers, overflows of which comprise flotation feed. Flotation operations divided into lead and zinc circuits, tailings from lead circuit comprising feed for zinc circuit. Roughing and cleaning system followed in both circuits; each circuit provided with ball mill for grinding flotation middlings.

Grade of products.—

Concentrates	Assays, percent		Assays, ounces
	Pb	Zn	Ag
Lead	74.2	5.8	24.9
Zinc	2.8	56.2	4.8

Recoveries, percent.—

Concentrates	Pb	Ag	Zn
Lead	91.0	81.1	9.7
Zinc	3.1	14.2	85.7

Ratio of concentration.—Lead concentrates, 9.1 : 1; zinc concentrates, 10.0 : 1; total concentrates, 4.76 : 1.

---

[8] Dalton, M. P., Milling Methods and Costs at the Morning Concentrator of the Federal Mining & Smelting Co., Mullan, Idaho: Inf. Circ. 6587, Bureau of Mines, 1932, 11 pp.

FIGURE 51.—F
flotation c
handling o

CONCENTRATOR I—ST. JOSEPH LEAD CO., HUGHESVILLE, MONT.[9]

Flow sheet.—Figure 52.

Ore treated.—Economic minerals—galena, sphalerite, marmatite, with pyrite in gangue of altered syenite and rhyolite with subordinate amounts of calcite, barite, quartz, rhodochrosite, and marcasite. Ore contains some gold and silver and crushes readily, but fine grinding necessary to free zinc from pyrite.

Capacity.—400 tons per 24 hours; 13 tons per hour treated in 1929.

Period covered.—Year 1929.

Milling method.—8 percent of the material hoisted rejected by hand sorting; remainder crushed, ground, and treated by all-flotation methods which produce lead concentrates and zinc concentrates.

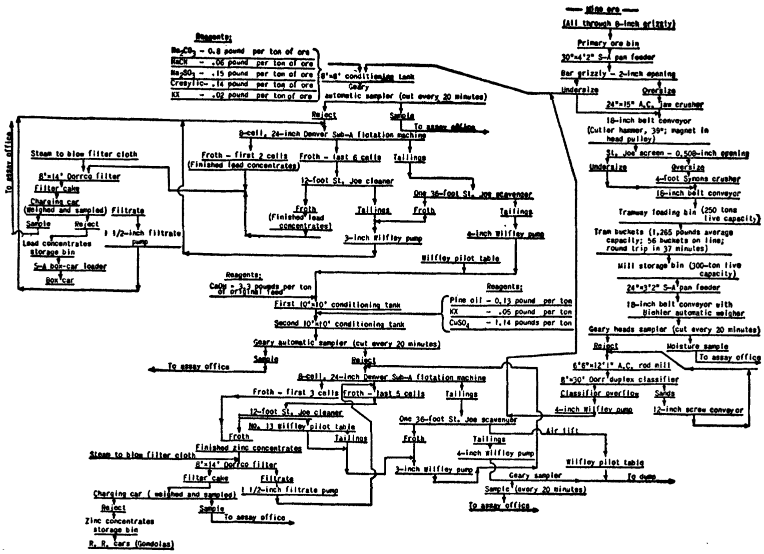

FIGURE 52.—Flow sheet of crushing plant and mill, Hughesville concentrator, St. Joseph Lead Co., Hughesville, Mont., concentrator I.

Grade of products.—

	Assays, percent		Assays, ounces	
	Pb	Zn	Ag	Au
Heads	6.21	5.01	9.09	------------
Lead concentrates	61.0	5.7	50.2	0.05
Zinc concentrates	1.0	51.0	30.9	.02
Tailings	.23	.49	2.2	------------

Ratio of concentration.—Lead section, 10.29:1; zinc section, 12.65:1; total, 11.36:1.

Net water consumption per ton of ore.—6 tons (none reclaimed).

[9] Vanderburg, Wm. O., Milling Methods at the Hughesville Concentrator of the St. Joseph Lead Co., Hughesville, Mont.: Inf. Circ. 6447, Bureau of Mines, 1931, 16 pp.

CONCENTRATOR J—TREADWELL YUKON CO., TYBO, NEV.[10]

Flow sheet.—Figure 53.

Ore treated.—Economic minerals—sphalerite and galena. Ore complex, containing lead, zinc, iron, silver, and gold and partly oxidized in upper levels. Primary slimes in oxidized ore troublesome in flotation and necessitate mixing ores from upper and lower levels to maintain uniform characteristics in mill feed.

Capacity.—320 tons per 24 hours, or 13.3 tons per hour, treated in March 1930.

Period covered.—March 1930.

Milling method.—All flotation, producing lead concentrates and zinc concentrates.

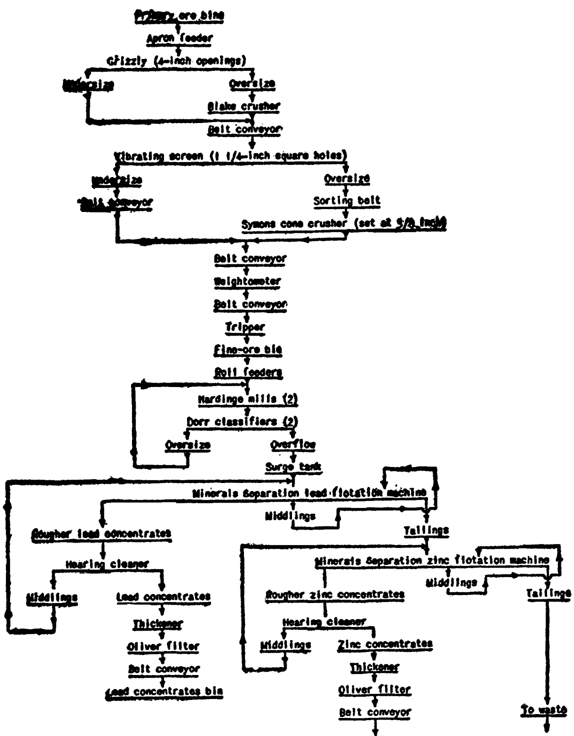

FIGURE 53.—Flow sheet of Tybo concentrator, Treadwell-Yukon Co., Ltd., Tybo, Nev., concentrator J.

Grade of products.—

	Assays, percent		Assays, ounces
	Pb	Zn	Ag
Heads	5.95	4.14	9.44
Lead concentrates	59.90	4.00	87.10
Zinc concentrates	2.55	47.60	14.28
Lead section tailings	1.85	4.17	3.60
Zinc section tailings	1.85	1.01	3.02

[10] Blackburn, W. H., Milling Methods and Costs at the Lead-Zinc Concentrator of the Treadwell Yukon Co., Ltd., at Tybo, Nev.: Inf. Circ. 6430, Bureau of Mines, 1931, 15 pp.

Recoveries, percent.—Pb, 74.0; Zn, 79.2; Ag, 74.9.

Ratio of concentration.—Lead section, 14.12:1; zinc section, 15.89:1; total, 7.47:1.

Water consumption per ton of ore.—5 tons (none reclaimed).

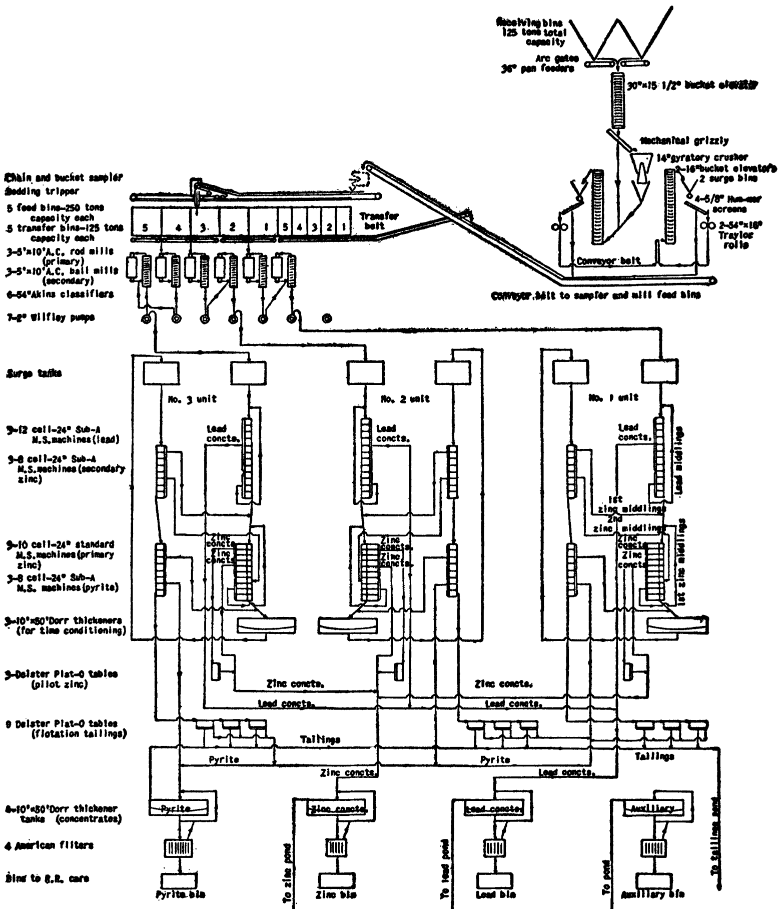

FIGURE 54.—Flow sheet of Midvale flotation plant, U.S. Smelting, Refining & Mining Co., Midvale, Utah, concentrator K.

CONCENTRATOR K—U.S. SMELTING, REFINING & MINING CO., MIDVALE CONCENTRATOR, MIDVALE, UTAH [11]

Flow sheet.—Figure 54.

Ore treated.—Economic minerals—galena and sphalerite. Due to handling much custom ore heads vary widely in metal content and physical characteristics. Much pyrite usually present, oxidation noticeable. Gangue—chiefly quartzite with some lime carbonate, porphyry, and talcose material.

Capacity—1,000 tons per day; during last 3 months of 1929, 982 tons per 24 hours treated, or about 41 tons per hour.

Period covered.—Last 3 months of 1929.

---

[11] Pallanch, R. A., Milling Methods at the Midvale Concentrator of the United States Smelting, Refining & Mining Co., Midvale, Utah: Inf. Circ. 6492, Bureau of Mines, 1931, 18 pp.

Milling method—Lead, zinc, and pyrite concentrates produced by all-flotation treatment. Tabling of 150 to 200 tons of flotation tailings per 24 hours; serves chiefly as pilot operation to measure efficiency of flotation.

Grade of products.—

	Assays, percent			Assays, ounces	
	Pb	Zn	Fe	Au	Ag
Heads (approximate)[1]	9.5	9.0	12.0	0.05	4.1
Zinc concentrates (approximate)	(?)	56.5	5.5		

[1] Represents about 40 percent of total mill heads. Other products, data lacking.

Ratio of concentration.—Lead concentrates, 5.42; zinc concentrates, 9.00; pyrite concentrates, 4.50; total concentrates, 1.93.

Water consumption per ton of ore.—8.3 tons (none reclaimed).

CONCENTRATOR L—CHIEF CONSOLIDATED MINING CO., EUREKA, UTAH [12]

Flow sheet.—Figure 55.

Ore treated.—Economic minerals—oxidized siliceous ores containing lead, silver, gold, and small amounts of copper. Gangue—chiefly quartz with some limonite and limestone.

Capacity.—Crushing plant, 65 tons per hour; grinding and flotation plant, 12.5 tons per hour; 168 tons per 24 hours, or 7 tons per hour, treated during 1929.

Period covered.—Year 1929.

Milling method.—All flotation, producing lead concentrates.

Grade of products.—

	Assays, percent				Assays, ounces	
	Pb	Zn	Fe	Cu	Ag	Au
Heads	11.36	1.65	7.27	0.202	13.00	0.0987
Concentrates	33.43	2.12	5.49	.300	32.05	.2250
Tailings	1.30	1.44	8.08	.157	4.30	.0411

Recoveries, percent.— Pb, 92.14; Zn, 40.19; Fe, 23.63; Cu, 46.72; Ag, 77.26; Au, 71.43.

Ratio of concentration.—3.192 : 1.

Water consumption per ton of ore.—4 tons (1.28 tons new water and 2.72 tons reclaimed from dewatering equipment).

CONCENTRATOR M—EAGLE-PICHER LEAD CO., MONTANA CONCENTRATOR, RUBY, ARIZ.[13]

Flow sheet.—Figure 56.

Ore treated.—Economic minerals—galena, sphalerite with pyrite, and some copper sulphides. Gangue—quartz. Ore contains some free gold. Colloidal slimes give considerable trouble at times.

Capacity.—250 tons per day, or 10.4 tons per hour; 258 tons per 24 hours, or about 11 tons per hour, treated between October 1, 1929 and April 1, 1930.

Period covered.—October 1, 1929 to April 1, 1930.

Milling method.—All flotation, producing lead concentrates and zinc concentrates.

---

[12] Wigton, G. H., Milling Methods and Costs at the Concentrator of the Chief Consolidated Mining Co., Eureka, Utah: Inf. Circ. 6320, Bureau of Mines, 1930, 18 pp.

[13] Andrus, D. E., Milling Methods and Costs at the Montana Mine Concentrator of the Eagle-Picher Lead Co., Ruby, Ariz.: Inf. Circ. 6497, 1931, 15 pp.

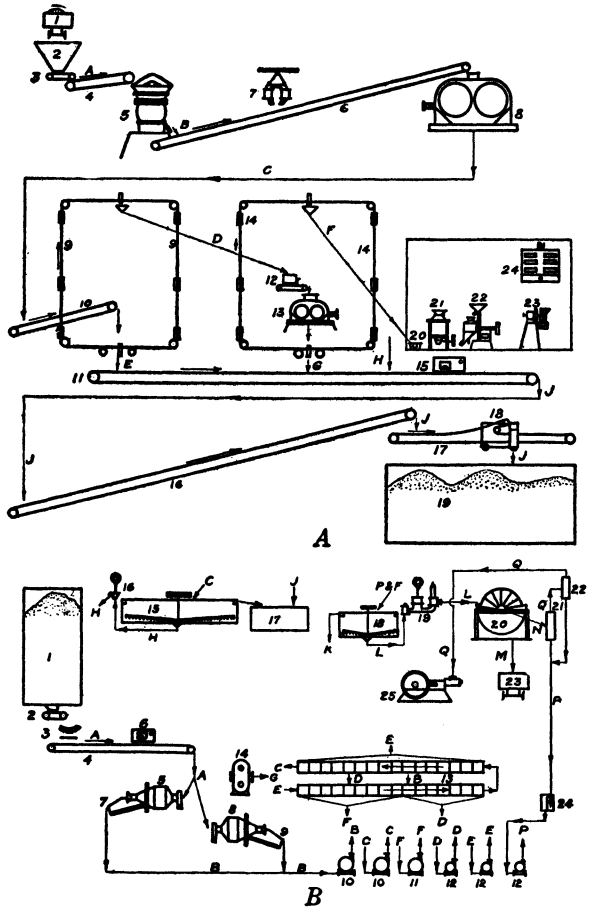

FIGURE 55.—Flow sheets of ore treatment, Chief Consolidated Mining Co., Eureka, Utah, concentrator L: *A*, Crushing and sampling plant—*1*, Denver & Rio Grande Western R. R. car; *2*, track hopper, 40 tons capacity; *3*, apron feeder, 30 by 40 inches; *4*, belt conveyor, 30 inches by 10 feet; *5*, McCully gyratory crusher, 13 by 45 inches; *6*, belt conveyor, 20 inches by 40 feet; *7*, stationary electromagnet; *8*, Garfield rolls, 5 by 2 feet, 80 r.p.m.; *9*, chain-bucket sampler, 6.378 percent cut; *10*, belt conveyor, 20 inches by 14 feet; *11*, belt conveyor, 20 inches by 40 feet; *12*, belt feeder and hopper; *13*, sample rolls, 3 by 16 feet; *14*, chain-bucket sampler, 3.904 percent cut; *15*, Merrick weightometer; *16*, belt conveyor, 20 inches by 115 feet; *17*, belt conveyor, 20 inches by 60 feet; *18*, movable tripper; *19*, concentrator storage bin, 1,100 tons capacity; *20*, receiving tub for sample; *21*, cone grinder; *22*, Braun universal sampler, 19 percent cut; *23*, McCool disk pulverizer; *24*, electric sample dryer; A, mine-run ore, 65 tons per hour capacity; B, crusher product; C, roll product; D, no. 1 sample; E, no. 1 reject; F, no. 2 sample; G, no. 2 reject; H, sample room reject; J, ball-mill feed. *B*, Flotation plant—*1*, concentrator storage bin; *2*, 5 apron feeders, 18 by 38 inches; *3*, collecting belt conveyor, 16 inches by 50 feet; *4*, belt conveyor, 16 inches by 36 feet; *5*, Hardinge ball mill, 8 by 4 feet, 20 r.p.m., capacity 170 tons flotation feed per 24 hours; *6*, Merrick weightometer; *7*, Dorr model D duplex classifier; *8*, Hardinge ball mill, 8 by 3 feet, 19 r.p.m., capacity 140 tons flotation feed per 24 hours; *9*, Dorr model D simplex classifier, 4 by 20 feet; *10*, two 4-inch Wilfley pumps; *11*, one 3-inch Wilfley pump; *12*, three 2-inch Wilfley pumps; *13*, two 16-cell, 21-inch Minerals Separation Sub-A flotation cells; *14*, no. 7 Sturtevant blower, 218 r.p.m., 1,200 c.f.m. at 1 pound pressure; *15*, Dorr thickener, 50 by 10 feet; *16*, Dorr duplex suction pump, 4 inches; *17*, mill-water storage tank, 25 by 10 feet; *18*, 2 Dorr thickeners, 25 by 10 feet; *19*, 2 simplex Dorrco pressure pumps, 4 inches; *20*, 2 American filters, 6-foot diameter, 4 and 6 leaf; *21*, 2 filtrate receivers, 6 by 6 feet; *22*, moisture trap, 2 by 8 feet; *23*, Denver & Rio Grande Western R.R. car, solid bottom; *24*, pump box and well for barometric leg; *25*, Worthington duplex vacuum pump, 14 by 12 inches; A, ball-mill feed, 170 tons per 24 hours; B, classifier overflow, flotation feed; C, flotation tailing; D, primary concentrate for re-treatment; E, scavenger concentrate for re-treatment; F, finished flotation concentrate; G, blower air for flotation; H, thickened tailing to waste; J, fresh water, 52,000 gallons per 24 hours; K, concentrate-thickener overflow for classifier dilution; L, thickened concentrate, 50 percent moisture; M, concentrate filter cake, 53 tons, 15 percent moisture; N, filtrate and air; P, filtrate; Q, air.

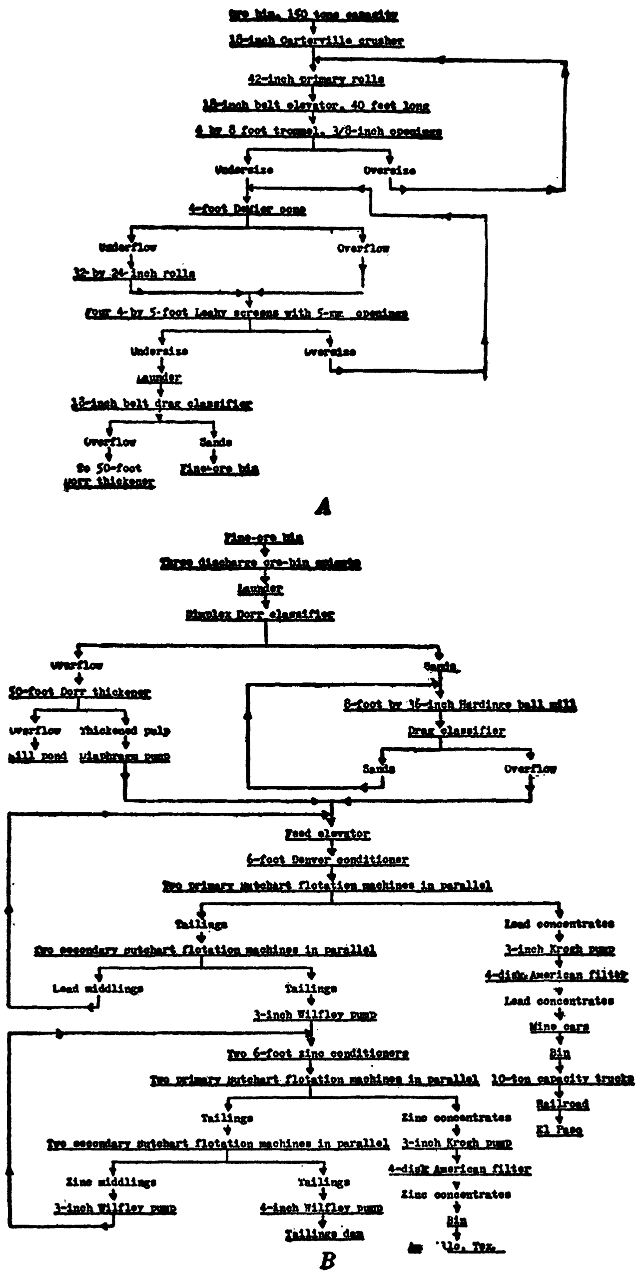

FIGURE 56.—Flow sheets of ore treatment, Montana concentrator, Eagle-Picher Lead Co., Ruby, Ariz., concentrator M: *A*, Crushing plant; *B*, concentrator.

Grade of products.—

	Assays, percent			Assays, ounces	
	Pb	Zn	Cu	Au	Ag
Heads	5.13	5.98	0.39	0.071	8.17
Lead concentrates	57.05	10.06	3.094	.674	74.02
Zinc concentrates	2.79	52.36	.72	.07	12.40
Tailings	.19	1.26	.095	.011	1.22

Recoveries, percent.—

Concentrates	Pb	Zn	Cu	Au	Ag
Lead	92.59	14.02	65.33	78.31	75.49
Zinc	4.23	68.29	14.41	7.89	11.92

Ratios of concentration.—Lead circuit, 12.01:1; zinc circuit, 12.83:1; total, 6.2:1.

CONCENTRATOR N—BLACK HAWK CONSOLIDATED MINES CO., HANOVER, N.MEX.[14]

Flow sheet.—Figure 57.

Ore treated.—Economic minerals—galena, sphalerite with pyrite, and some chalcopyrite. Gangue—limestone-garnet. Ore minerals relatively coarse, liberated without particularly fine grinding.

Capacity.—160 to 180 tons per day, or 6.7 to 7.5 tons per hour.

Period covered.—September and October 1929.

Milling method.—All flotation, following some hand sorting of waste, producing lead concentrates and zinc concentrates.

Grade of products.—

	Assays, percent			Assays, ounces
	Pb	Zn	Cu	Ag
Heads	1.99	9.93	0.43	1.155
Lead concentrates	58.37	11.57	3.68	25.22
Zinc concentrates	1.81	54.77	1.55	1.78

Recoveries, percent.—

Concentrates	Pb	Zn	Cu	Ag
Lead	81.86	-----------	23.45	62.95
Zinc	-----------	90.69	-----------	-----------

Ratios of concentration.—Lead concentrates, 36.1:1; zinc concentrates, 6.3:1; total, 5.35:1.

Net water consumption per ton of ore.—1.69 tons. Balance of water required reclaimed from tailings pond and filters.

[14] Wright, Ira L., Milling Methods at the Black Hawk Concentrator, Hanover, N.Mex.: Inf. Circ. 6359, Bureau of Mines, 1930, 16 pp.

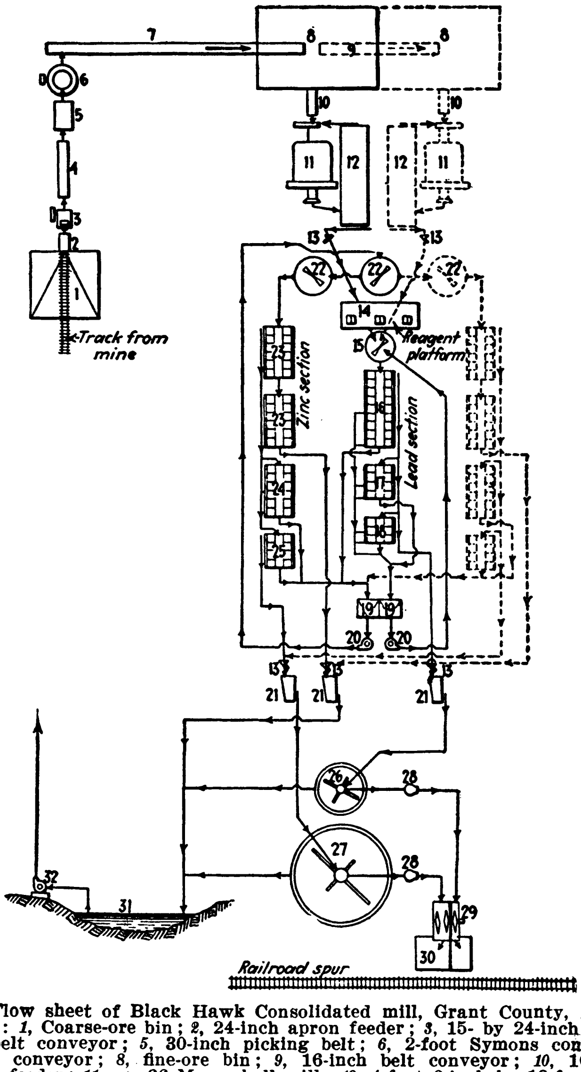

FIGURE 57.—Flow sheet of Black Hawk Consolidated mill, Grant County, N. Mex., concentrator N: *1*, Coarse-ore bin; *2*, 24-inch apron feeder; *3*, 15- by 24-inch jaw crusher; *4*, 16-inch belt conveyor; *5*, 30-inch picking belt; *6*, 2-foot Symons cone crusher; *7*, 16-inch belt conveyor; *8*, fine-ore bin; *9*, 16-inch belt conveyor; *10*, 16-inch Southwestern belt feeder; *11*, no. 66 Marcy ball mill; *12*, 4-foot 6-inch by 18-foot Dorr classifier, *13*, Southwestern pulp samplers; *14*, Southwestern reagent feeders; *15*, 5- by 5-foot Southwestern conditioner; *16*, MB–3012 Southwestern air flotation machine, lead rougher; *17*, MB–3008 Southwestern air flotation machine, lead cleaner; *18*, MB–3006 Southwestern air flotation machine, lead recleaner; *19*, sumps; *20*, 2-inch sand pumps; *21*, no. 13 Wilfley pilot tables; *22*, 6- by 6-foot Southwestern conditioners; *23*, MB–3012 Southwestern air flotation machines, zinc rougher; *24*, MB–3012 Southwestern air flotation machines, zinc cleaner; *25*, MB–3008 Southwestern air flotation machines, zinc recleaner; *26*, 10- by 8-foot Dorr thickener; *27*, 22- by 10-foot Dorr thickener; *28*, 2-inch Dorrco suction pumps; *29*, 6-foot diameter, 3-disk American filter; *30*, concentrate bins; *31*, tailing pond; *32*, recovery water pump.

CONCENTRATOR O—AMERICAN METAL CO., TERERRO, N.MEX.[15]

Flow sheet.—Figure 58.

Ore treated.—Economic minerals—massive sphlerite and galena, with pyrite, chalcopyrite, marmatite, chalcocite, bornite, argentite, and proustite.

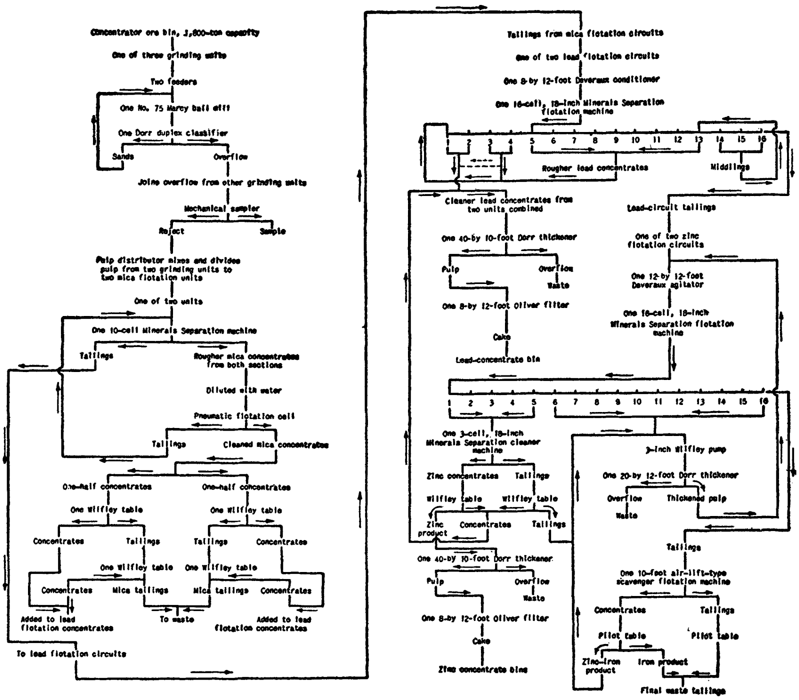

FIGURE 58.—Flow sheet of Pecos concentrator, American Metal Co., Tererro, N.Mex., concentrator O.

Gangue—chiefly mica, quartz, tourmaline, and hornblende. Gold present in intimate association with sulphides and mica. Part of ore minerals very finely divided.

Capacity.—600 tons per 24 hours, or 25 tons per hour; 151,943 tons treated in 1930.

Period covered.—Year 1930.

Milling method.—All-flotation treatment, involving flotation of micaceous gangue material ahead of lead circuit.

Grade of products.—

	Assays, percent				Assays, ounces	
	Pb	Zn	Cu	Fe	Au	Ag
Heads	4.93	15.41	0.83	11.40	0.107	3.37
Lead concentrates	37.81	13.47	4.02	11.47	.768	19.98
Zinc concentrates	1.63	54.45	1.00	6.99	.036	3.05
Tailings	.76	1.33	.26	14.20	.024	.77

[15] Bemis, H. D., Milling Methods and Costs at the Pecos Concentrator of the American Metal Co., Tererro, N.Nex.: Inf. Circ. 6605, Bureau of Mines, 1932, 23 pp.

Recoveries, percent.—

	Pb	Zn	Cu	Au	Ag
Lead concentrates	81.97	9.35	52.02	76.89	63.43
Zinc concentrates	7.91	84.64	28.99	8.06	21.73
Tailings	10.12	6.01	18.99	15.05	14.84

Ratios of concentration.—Lead concentrates, 9.35 :1 ; zinc concentrates, 4.16 : 1; total concentrates, 2.88 : 1.

Net water consumption per ton of ore.—5 tons (practically none reclaimed).

CONCENTRATOR P—ST. JOSEPH LEAD CO., BALMAT, N.Y.[16]

Flow sheet.—Figure 59.

Ore treated.—Economic minerals—ferruginous sphalerite and pyrite, with a little galena. Gangue—dolomite, silicate minerals, and quartz. Some oxidation in upper levels produced hematite, magnetite, and secondary sphalerite intimately associated with primary sulphide. Talcose slimes have caused some difficulties in flotation.

Capacity.—Designed for 500 tons; maximum tonnage milled, 620 tons per 24 hours. During 1930 average tonnage treated was 460 tons per 24 hours, or 19.2 tons per hour. During first 5 months of 1931 average tonnage treated was 545 tons per 24 hours, or 22.7 tons per hour.

Period covered.—Year 1930 (including initial stage of mill operation) and first 5 months of 1931.

Milling method.—All flotation, producing zinc, lead, and pyrite concentrates.

Grade of products (1930).—

	Assays, percent		
	Pb	Zn	Fe
Heads	1.66	12.29	12.93
Lead concentrates	53.05		
Zinc concentrates		55.88	
Pyrite concentrates			43.80
Tailings	.58	1.90	8.95

Recoveries, percent.—

	1930	Jan. 1 to May 1, 1931
Lead	59.01	66.9
Zinc	81.42	82.2
Iron (pyrite)	45.08	66.4

Ratios of concentration.—

	1930	Jan. 1 to May 1, 1931
Lead circuit	54.26 to 1	48.13 to 1
Zinc circuit	5.58 to 1	5.74 to 1
Pyrite circuit	7.57 to 1	6.31 to 1
Total	3.04 to 1	2.82 to 1

Net water consumption per ton of ore.—3.6 tons (approximate) (none reclaimed).

[16] Knaebel, John B., Milling Methods at the Balmat Mill of the St. Joseph Lead Co., Balmat, St. Lawrence County, N.Y.: Inf. Circ. 6574, Bureau of Mines, 1932, 29 pp.

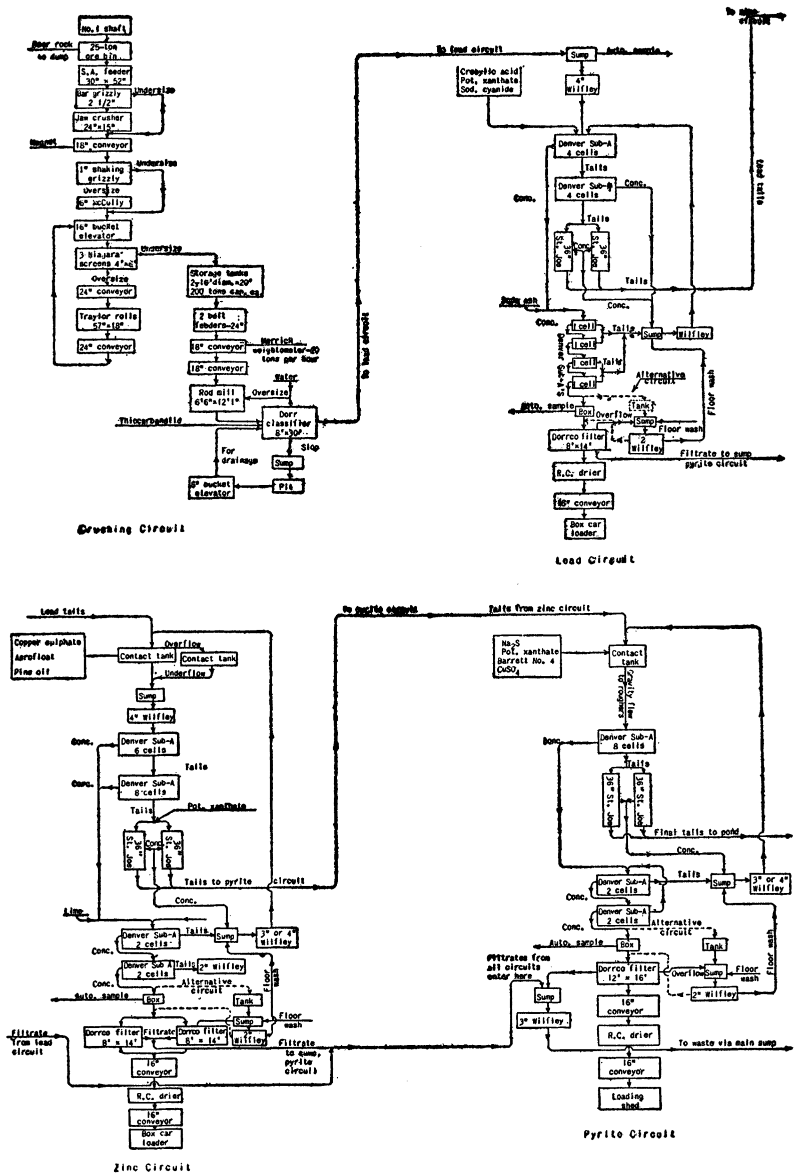

FIGURE 59.—Flow sheet of Balmat 500-ton mill, St. Joseph Lead Co., St. Lawrence County, N.Y., concentrator P.

DISCUSSION OF MILLING OPERATIONS

GENERAL

The best method of treating a given lead-zinc ore would be difficult to prescribe without a thorough study of its composition, mineral distribution, physical characteristics, amenability to gravity and flotation treatment, and other determining factors. Local conditions as well as character and grade of products desired with reference to smelting contracts often have an important bearing on the method of treatment.

Since more or less universal acceptance of the flotation process for better separation and improved recoveries in the treatment of slimes and finely ground ore, many complex and finely disseminated sulphide ores can be treated successfully which formerly would not yield acceptable products by gravity methods. Zinc concentrates with a high pyrite content produced by gravity methods required a flash roast and magnetic separation to yield the desired grade, but it is now possible by use of the proper reagents to depress iron sulphide minerals in the flotation treatment of lead and zinc ores. For many years high-grade products have been obtained by gravity methods in concentrating the Tri-State lead and zinc ores, but recoveries have been greatly improved since flotation was added to the flow sheets for treating slimes, most of which were formerly discarded.

Ores of the Southeastern Missouri, Tri-State, and Mascot (Tenn.) districts, are good examples of those suitable for treatment by gravity methods, and most of the complex and finely disseminated lead-zinc ores of the Coeur d'Alene and other western districts are representative of all-flotation treatment. The different methods applied to lead and zinc ores and the characteristics of the lead and zinc ores treated are indicated in the preceding flow sheets of various concentrators and will be discussed in more detail in the following pages.

HAND-SORTING

Hand-sorting is practiced at relatively few concentrators. Advocates of hand-sorting claim that elimination of waste rock increases mill capacity, raises the grade of mill heads, reduces wear and tear on machinery, and improves metallurgical results. The advantages of this practice will, of course, depend upon the character of the ore, the quantity of waste rock sorted, and local conditions.

Both waste rock and the crude lead smelting product are hand-sorted at concentrator D. The hand-sorted lead product contains about 50 percent lead, and the quantity obtained is said to be approximately one half the quantity of concentrates produced by milling operations. At concentrator I about 35 percent of the material broken underground is rejected as waste and is used to fill mined-out portions of the vein and to obtain a higher-grade product for concentration; also, about 8 percent is rejected as waste by hand-sorting at the grizzly ahead of the primary breaker.

CRUSHING

The ore delivered to the crushing-plant hopper is usually broken to pass a grizzly with openings of 10 inches or less. At some plants the ore from the bins passes over grizzlies permitting 1½- to 4-inch material to by-pass the primary crusher and join the crusher product for additional crushing or sizing.

The ore is first crushed in Blake-type jaw crushers or in gyratories, usually to about 2 to 3 inches in size, although at concentrator E the gyratory crusher discharges a product 5½ inches in maximum size and at concentrator D the jaw crusher is set at 5 inches. This coars ecrushing is necessary at concentrator D as a relatively large amount of crude lead and waste rock is sorted out on the picking belt.

In the Tri-State district the crusher product is usually fed direct to rolls for further reduction; but in other districts this product passes over stationary, trommel, or vibrating screens with ½- to 2-inch openings, the oversize being fed to disk or cone crushers, to gyratories, or to rolls (in closed circuit with the screen) and crushed to whatever size is desired for subsequent treatment or grinding.

In plants where gravity concentration is employed jig and table middlings are generally reground in secondary rolls or in ball or rod mills for treatment by sand jigs, tables, or flotation.

Some typical screen analyses of products from crushing plants are given in tables 15, 16, and 17.

TABLE 15.—*Screen analyses of successive crusher products, concentrator F*

Size	Weight, percent			
	Gyratory crusher discharge (Symons crusher feed)	Symons horizontal disk crusher discharge (new feed to rolls)	Composite roll feed (new feed and dry screen oversize)	Roll discharge
+3.0-inch	11.7			
2.1-inch	19.0			
1.05-inch	29.6	45.5	6.2	
0.0525-inch	15.0	34.3	11.9	8.9
3-mesh	9.5	11.3	40.2	35.6
8-mesh	7.9	4.5	37.1	39.9
20-mesh	3.5	1.6	2.7	5.8
48-mesh	1.6	.8	.4	2.6
200-mesh	1.2	.7	.4	1.7
−200-mesh	1.0	1.3	1.2	5.5
Total	100.0	100.0	100.0	100.0

TABLE 16.—*Screen analyses of crusher products, concentrator G*

Size	Weight, percent				
	Mine-run ore (feed to Blake crusher)	Blake crusher discharge	Feed to Symons cone crusher	Symons cone crusher discharge (feed to rolls)	Roll discharge
+3.0-inch	18.0	2.67	1.60		
2.0-inch	6.33	6.25	14.50		
1.0-inch	16.67	11.11	45.60	18.82	
0.5-inch	12.33	10.67	27.40	45.88	4.7
4-mesh	17.97	17.50	2.50	19.80	25.4
8-mesh	7.30	11.20	.60	4.10	17.1
20-mesh	6.90	16.00	1.40	2.80	17.1
48-mesh	4.80	8.10	1.80	2.30	10.7
200-mesh	3.50	6.80	1.50	2.10	7.6
200-mesh	6.20	9.70	3.10	4.30	17.4
Total	100.00	100.00	100.00	100.10	100.0

TABLE 17.—*Screen analyses of crusher and roll products, concentrator L*

Size	Weight, percent		Size	Weight, percent	
	Gyratory discharge (feed to rolls)	Roll discharge		Gyratory discharge (feed to rolls)	Roll discharge
+2.0-inch	16.006		48-mesh	4.397	7.012
1.05-inch	25.529	7.407	200-mesh	4.006	6.813
.525-inch	17.572	24.779	−200-mesh	1.227	2.261
3-mesh	19.758	22.842			
8-mesh	5.944	19.339	Total	100.000	100.000
20-mesh	5.561	9.547			

GRINDING

Rolls, Huntington mills, and Chilean mills, which formerly were used in concentrators to liberate the mineral particles of middlings and finely disseminated ores, have largely been replaced by rod and ball mills. The feed to these mills may comprise middlings from sand jigs and tables, or oversize from fine screens and tailings from tables for further treatment on tables; it may be all middlings and tailings from jigs and tables, or tailings from sand and cleaner jigs only, for further treatment by flotation; or where no gravity concentration is employed it may consist of mill feed that has been crushed, in most cases to 1-inch size or less, which after grinding is subjected to an all-flotation treatment.

In nearly all cases the discharge is classified in drag classifiers of the Dorr or Esperanza types, the sands being returned to the grinding mill or to hydraulic classifiers and the overflow to thickeners for subsequent treatment by flotation. The size of material to be ground depends upon the source of the feed to the mill, and the fineness of grinding depends upon the method of treatment to be used.

The screen analyses of feed and discharge, given in tables 18 to 21, indicate the fineness of grinding practiced at some of the mills covered in this report. In table 19 the feed to the rod mill comprises the screen oversize at the head of the concentrating tables and the low-grade middlings from tables after they have been dewatered by a drag classifier. The rod-mill discharge goes direct to hydraulic classifiers for treatment on tables.

TABLE 18.—*Screen analyses of ball-mill products grinding low-grade middlings in closed circuit with drag classifier, concentrator C*

Size, mesh	Weight, percent			
	Feed to ball mill	Ball-mill discharge	Drag classifier	
			Sand	Overflow
+3	0.9			
8	51.4	2.2	1.0	
20	27.7	6.0	5.4	
48	11.1	24.8	31.4	
100	7.7	32.2	39.0	6.5
200	1.1	14.9	14.3	21.4
−200	.1	19.9	8.9	72.1
	100.0	100.0	100.0	100.0

TABLE 23.—*Screen analyses and tonnages of hydraulic-classifier products, low-grade table middlings in open circuit, concentrator F*

Size, mesh	Weight, percent			
	Screen oversize (1)	Low-grade middlings from drag classifier (1)	Screen oversize and low-grade table middlings	
			Rod-mill feed (1) and (2)	Rod-mill discharge
+8	31.6	0.3	8.3	
20	53.9	23.8	39.0	0.5
48	5.4	58.5	43.0	43.9
100	1.9	15.3	4.5	19.7
200	1.4	0.8	1.2	8.7
−200	5.8	.5	4.0	27.2
	100.0	99.2	100.0	100.0

TABLE 20.—*Screen analyses of rod-mill products grinding mill feed crushed to 1-inch size in closed circuit with drag classifier, concentrator I*

Size	Weight, percent			
	Rod-mill feed	Classifier feed	Classifier sand	Classifier overflow
+1-inch	0.7			
.5-inch	12.1			
3-mesh	37.2			
6-mesh	16.9			
20-mesh	19.6	0.4	0.6	
48-mesh	8.1	14.5	18.3	1.1
100-mesh	2.8	37.7	41.8	23.7
200-mesh	2.6	28.6	30.5	27.7
−200-mesh		18.8	8.8	47.5
	100.0	100.0	100.0	100.0

TABLE 21.—*Screen analyses of ball-mill products in closed circuit with drag classifier, concentrator L*

Size	Weight, percent				Size	Weight, percent			
	Ball-mill feed	Ball-mill discharge	Classifier sands	Classifier overflow		Ball-mill feed	Ball-mill discharge	Classifier sands	Classifier overflow
+1.05-inch	7.41				100-mesh	3.90	25.4	34.4	10.89
.525-inch	24.78				200-mesh	2.91	10.3	10.1	15.84
3-mesh	22.84				−200-mesh	2.26	25.2	10.2	73.27
8-mesh	19.34								
20-mesh	9.55	6.8	7.0			100.00	100.0	100.0	100.00
48-mesh	7.01	32.3	38.3						

Table 22 indicates some conditions under which rod-and-ball mills are operated. It will be noted that at concentrators where all flotation is employed, the grinding mills are operated with a pulp density ranging from 70 to 85 percent solids. The grinding-mill discharge usually contains about 25 percent of minus 200-mesh material, and the circulating load is 200 to 400 percent of the new feed.

At concentrators A and B the grinding mills are operated in open circuit, the discharge being returned for further treatment on tables.

Steel consumption.—According to the data given, the consumption of steel balls and rods averages 2 to 3 pounds per ton of material ground and that of liners 0.4 to 0.5 pound. The report covering milling methods at concentrator P [17] states that during the first 15 months of operation the feed entered the rod-mill classifier circuit at the lower end of the classifier. A change was then made by which the feed went direct to the rod mill, and the rod consumption is said to have been reduced by about 0.4 pound per ton of ore ground.

[17] Knaebel, J. B., Milling Methods at the Balmat Mill of the St. Joseph Lead Co., Balmat, St. Lawrence County, N.Y.: Inf. Circ. 6574, Bureau of Mines, 1932, p. 7.

TABLE 22.—*Operating data of ball and rod mills at various lead and zinc concentrators*

Concentrator	Number and size of grinding mills	Feed to grinding unit	Maximum and minimum size in feed, percent	Maximum and minimum size in discharge, percent	Pulp density of discharge, solids, percent	Circulating load, percent of new feed	Grinding medium	Consumption of grinding medium per ton of material ground, pounds	Mill-liners composition	Consumption of mill liners per ton of material ground, pounds
A	One 6-foot ball mill	Dewatered bed middlings from De Mier level jig and zinc-sand middlings from tables.	31.6 on 20-mesh, 3.65 minus 65-mesh.	4.44 on 20-mesh, 40.67 minus 65-mesh.		(1)	3-inch forged-steel balls.	2.2	Manganese steel.	
B	One 6-foot by 36-inch ball mill.	Oversize from 2-mm screen ahead of tables and dewatered tailings from coarse-sand tables.	Maximum, ⅛-inch.	1.7 on 14-mesh, 17.9 minus 100-mesh.	70	(1)	do	2.0	Titanite	1.0
C	One 8-foot by 36-inch and three 6-foot by 22-inch ball mill.	Dewatered middlings and tailings from jigs and tables; low-grade circuit.	13.1 on 4-mesh, 0.1 minus 200-mesh.	0.5 on 4-mesh, 19.9 minus 200-mesh.	70.2		Three brands of cast-iron balls in use.	Brands A–3.21, B–3.25, C–2.82.	Electric steel and cast iron.	
H	Seven 8-foot by 36-inch ball mills and one 8-foot by 22-inch ball mill.	Mill feed to 5 primary mills in partly closed circuit with classifier, part of sands returning to primary mills and part to 2 secondary mills; one 8-foot by 22-inch mill for regrinding middlings.	Small percentage on 1-inch.				3- and 4-inch cast-iron and steel balls.	2.19	Cast iron	.4
I	One 6-foot 6-inch by 12-foot 1-inch rod mill.	Mill feed. Rod mill in closed circuit with classifier.	12.8 on ½-inch, 2.6 minus 100-mesh.	0.4 on 20-mesh, 18.8 minus 200-mesh.		300	Steel	1.986	Manganese steel.	.247
J	Two 8-foot by 48-inch ball mills.	Mill feed to ball mills in closed circuit with classifiers.	Minus ½-inch	9.75 on 14-mesh, 37.77 minus 200-mesh.	75–80	400	3-inch "Adamantine" steel balls.	2.7	do	.21
K	Three 5- by 10-foot rod mills and three 5- by 10-foot ball mills.	Mill feed to 3 rod mills in closed circuit with classifier; overflow to second classifier in closed circuit with 3 ball mills.	Rod mills: 6.1 on 1-inch, 12.6 minus 200-mesh; ball mills: 0.2 on 14-mesh, 11.2 minus 200-mesh.	Rod mills: 0.4 on 14-mesh, 27.2 minus 200-mesh; ball mills: 0.2 on 14-mesh, 25.7 minus 200-mesh.	75–80		2.5- to 3.0-inch steel rods (C, 0.8 to 1.0 percent; 2.0- to 2.5-inch cast chilled balls).	Rods, 1.2; balls, 1.1.	do	.21

[1] Open circuit.

TABLE 22.—*Operating data of ball and rod mills at various lead and zinc concentrators*—Continued

Concentrator	Number and size of grinding mills	Feed to grinding unit	Maximum and minimum size in feed, percent	Maximum and minimum size in discharge, percent	Pulp density of discharge, solids, percent	Circulating load, percent of new feed	Grinding medium	Consumption of grinding medium per ton of material ground, pounds	Mill-liners composition	Consumption of mill liners per ton of material ground, pounds
L	One 8-foot by 48-inch and one 8-foot by 36-inch ball mill.	Mill feed to ball mills in closed circuit with classifier.	7.407 on 1.05-inch, 2.261 minus 200-mesh.	2.2 on 10-mesh, 25.2 minus 200-mesh.	75	340	5-inch chrome steel balls.	3.24	Titanite	0.71
M	One 8-foot by 36-inch ball mill.	Mill feed from fine-ore bins passes to classifier, the sands from which are ground in ball mill in closed circuit with drag belt.	Minus 5-mm				3-inch cast-iron balls.	3.19		
N	One 6- by 6-foot ball mill.	Mill feed from fine-ore bin to ball mill in closed circuit with classifier.	4.28 on ¼-inch, 5.95 minus 200-mesh.		72	300	4-inch chrome steel.	1.5	Manganese steel	
O	3 no. "75" ball mills	Mill feed to ball mills in closed circuit with classifier.	6.0 on 1-inch, 7.3 minus 200-mesh.	17.2 on 48-mesh, 26.8 minus 200-mesh.	85	400	4½-inch forged-steel balls.	1.8	Manganese steel and chrome-molybdenum steel.	.3
P	One 6-foot 6-inch by 12-foot 1-inch rod mill.	Mill feed from fine-ore bins to rod mill in closed circuit with classifier.	8.1 on 0.742-inch, 5.0 minus 200-mesh.	1.0 on 20-mesh, 19.8 minus 200-mesh.		200	2.5-inch manganese-chrome steel rods.	2.2		

SIZING AND CLASSIFYING

Methods employed.—In reducing ore by crushing, the fine material is separated from the coarse by grizzly or punched-plate stationary screens, by vibrating screens, or by revolving trommels. Crushed material is generally graded into different sizes by trommels for the coarser jig sizes and by vibrating screens for the finer jig and table sizes. Vibrating screens are replacing trammels in many instances.

At the concentrators covered in this report finer sizes of material are classified by desliming in cone classifiers or in classifiers of the Dorr or Esperanza types, and material to be treated on concentrating tables is classified in hydraulic classifiers of the sorting-column type.

Since adoption of flotation at most lead-zinc concentrators throughout the country much greater emphasis has been placed on the prevention of slime losses by proper desliming and thickening equipment. Although flotation has largely replaced vanners and canvas plants much improvement in table concentration has been made possible by proper classification in multiple-spigot hydraulic classifiers. Table 23 gives an example of hydraulic classification of material fed to eight concentrating tables, as practiced at concentrator F.

TABLE 23.—*Screen analyses and tonnages of hydraulic-classifier products, concentrator F*

Size, mesh	Feed	Spigot 1	Spigot 2	Spigot 3	Spigot 4	Spigot 5	Spigot 6	Spigots 7 and 8	Spigots 9 and 10	Overflow
On 8	0.2	1.6	0.4	0.1						
10	3.7	17.8	9.8	2.7	0.3	0.1				
14	8.0	28.0	19.7	9.3	2.3	1.5				
20	6.6	17.9	16.4	11.3	4.9	4.1	0.4	0.3	0.2	
28	11.1	16.0	18.5	20.2	13.1	11.5	3.5	2.6	2.2	
35	13.2	9.6	15.1	22.0	22.3	16.9	8.8	7.4	6.6	
48	14.3	4.4	9.6	16.2	24.5	19.8	21.4	18.0	16.1	
65	20.3	3.2	7.0	12.2	22.9	34.1	41.7	45.0	38.0	
100	7.4	.8	1.9	3.1	5.0	6.7	13.1	14.8	21.4	1.2
150	5.1	.4	.9	1.4	2.5	3.0	7.0	8.0	12.3	14.3
200	2.3	.1	.4	.5	.9	.3	.4	1.2	1.1	17.1
Through 200	7.8	.2	.4	.7	1.3	2.0	3.7	2.7	2.1	67.4
	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Mean mesh ... mm	0.305	0.828	0.623	0.451	0.323	0.289	0.215	0.207	0.195	
Per 24 hours ... tons	[1] 418.4	52.1	78.4	41.4	48.8	49.4	47.6	52.3	48.4	

[1] This tonnage does not include the slime in the overflow.

From 1924 to 1929 slimes were separated at concentrator C as follows: [18]

* * * By the use of 8-foot steel cones equipped with Fahrenwald classifying mechanism such as is used on the Fahrenwald sizer. The overflow of the Esperanza classifier and of the Fahrenwald sizer, being 92 percent minus 150-mesh in size, was divided equally between these cones, and the sand discharge from each cone was fed to a 6-foot Frue vanner. The overflow of the cones was thickened and sent to the flotation plant.

The comparison of efficiency of the vanner when treating the feed to these cones, thickened to 7 parts of ater to 1 of solids, and when treating the sand from the cones is shown as follows:

[18] Handy, R. S., Milling Methods and Costs at the Northern Idaho Mills of the Bunker Hill & Sullivan Mining & Concentrating Co.: Inf. Circ. 6314, Bureau of Mines, p. 41.

One Frue vanner treating feed to Fahrenwald cones

Product	Tons per 24 hours	Lead assay, percent	Lead, tons	Total lead, percent
Feed	2.5	7.0	0.175	100.0
Concentrates	.17	55.0	.096	55.0
Tailings	2.33	3.3	.079	45.0

One Fahrenwald cone handling overflows of hydraulic classifiers

Product	Tons per 24 hours	Weight, percent	Lead assay, percent	Lead, tons	Total lead, percent
Feed	27.23	100.0	6.90	1.875	100.0
Sand to vanner	23.43	86.0	6.80	1.593	85.0
Overflow	3.80	14.0	7.44	.282	15.0

One Frue vanner treating sand from Fahrenwald cone

Product	Tons per 24 hours	Lead assay, percent	Lead, tons	Total lead, percent
Feed	23.43	6.8	1.593	100.0
Concentrates	2.09	69.0	1.442	90.5
Tailings	21.34	.7	.151	9.5

Thus, the percentage of the total lead in the feed to the cone classifiers that was recovered by the vanner was 90.5 percent of 85.0 percent, or 76.9 percent, as compared with 55 percent when treating the feed to the cones without classification.

Pulp densities and fineness of overflow.—In most of the plants where the grinding units are operated in closed circuit with Dorr and Esperanza type classifiers the classifier overflows have a pulp density of about 25 to 40 percent solids. The amount of minus 200-mesh material in the classifier overflows at the different plants is 30 to 80 percent of the total solids, depending upon the fineness of grinding necessary to liberate the mineral particles and to effect the desired grades and recoveries of the final products.

At the concentrators in the Tri-State district the feed to the rougher jigs and the jig tailings are deslimed by cone or drag classifiers.

GRAVITY CONCENTRATION

Of the 16 concentrating plants covered in this report 6 employ gravity concentration supplemented by flotation. At 3 of the 6 mills gravity concentration is accomplished by jigs and tables, at 2 by jigs only, and at 1 by tables only. Jig tailings are discarded at 3 of these plants, and at the other 2 they are ground and treated by flotation.

JIGGING

Jigging methods practiced in lead-zinc concentrators may be divided into two classes: (1) Concentration of sized material and (2) concentration by roughing and cleaning. The following brief description will indicate the jigging methods practiced at various concentrators.

Concentration of sized material.—At concentrator C the mill feed is crushed to 7-mm size; a sized product between 7 mm and 0.13 inch is treated on one set of jigs and the material between minus 0.13 inch and plus 0.0325 inch on another set of jigs. A lead product is obtained from the first two cells of the jigs, and the middlings and tailings are ground in ball mills for further treatment by flotation. The fine sands are treated on tables.

At concentrator D the mill feed is graded by trommel screens into products of minus 30-mm, 18-mm, 12-mm, 7-mm, 3-mm, and 16-mesh sizes. The jigs which handle material coarser than 16 mesh produce concentrates, middlings, and tailings. The middlings from the two coarser jigs are crushed in rolls and returned to the 18-mm screen, and those of the next two finer jigs are crushed and returned to the 3-mm screen. The middlings from the jigs treating the finest sizes join the minus 16-mesh mill feed and are dewatered in preparation for grinding and flotation. No tables are used at this concentrator.

Concentration by roughing and cleaning.—A roughing and cleaning system of jigging is practiced at concentrators A, B, and E. The mill feed, which has been crushed to about five eighths inch or less, is deslimed and fed to 6- or 7-cell rougher jigs. The hutch products from the first 4 or 5 cells go to a cleaner jig, which produces final lead and zinc concentrates; and the hutch products from the last rougher cells, the rougher bed middling, and the cleaner-jig tailing are reduced in rolls and treated in middling or sand jigs by roughing and cleaning, or by cleaning only. The middlings and tailings from sand jigs in plants A and B are treated on tables. A lead concentrate is produced from the first one or two cells of the cleaner jig, a mixed lead-zinc product from the second or third cell which is returned to the head of the cleaner, and a zinc concentrate from the remaining cells.

At concentrator E the rougher-jig tailings are treated in "bull jigs" from which the hutch products go to the rolls and sand jig and the dewatered tailings to waste. The sand-jig product goes to the cleaner jig, and the tailing goes to the rod mill and to flotation. The cleaner tailing joins the sand-jig tailing. No tables are used at concentrator E.

TABLE CONCENTRATION

The treatment of fine sands on concentrating tables, as practiced at the different mills, is described briefly in the following paragraphs.

The dewatered fine material from various sources in the jig section of concentrator A is fed to a DeMier level jig,[19] which is said to have proved very effective in jigging sand. The hutch products from this jig go to a battery of six concentrating tables, while the bed middlings are ground in a ball mill and returned to the jig.

[19] The DeMier level jig is built similarly to the standard jig except that all cells are at the same level; that is, there is no drop from one cell to the next. To facilitate travel of the material on the jig, a raking device, similar to that of a Dorr drag classifier, is suspended over the full length of the jig, with rakes about 18 inches apart. These rakes drag the surface of the bed intermittently forward toward the tail of the jig.

The tables produce a zinc concentrate, a zinc-lead product which is separated on another table, a zinc-sand middling which joins the jig-bed middlings in the ball mill, and tailings which are returned to the DeMier jig. The tailing from the jig is discarded.

At concentrator B the sand discharge of the belt drag classifier at the head of the table section is elevated to a 2-mm trommel screen. The oversize from this screen is ground in a ball mill and returned to the trommel screen, and the undersize passes to hydraulic classifiers and seven concentrating tables. These tables produce clean zinc concentrates, a lead-zinc product which is separated on another table, a zinc-sand middling which is returned to the head of the table circuit, and tailings. The tailings from the first 3 tables are ground in a ball mill with the trommel-screen oversize, and the tailings from the last 4 tables are deslimed and discarded.

A simpler flow sheet of table concentration is that used at concentrator C. The fine sands are treated over tables after classification in a Fahrenwald hindered-settling classifier. The lead product from tables goes to lead bins, and the middlings and tailings are ground in ball mills and later treated by flotation.

Gravity concentration at concentrator F is accomplished entirely by concentrating tables in six similar table sections. One section is described here. The mill feed is crushed to pass a 7-mesh vibrating screen and, after desliming, is fed to two 10-spigot hydraulic classifiers. The first 6 spigots feed one table each, while the seventh and eighth spigots and the ninth and tenth spigots are combined to feed 2 more tables. A concentrate, a high-grade middling, and a low-grade middling are made on each of the first 4 tables, while the last 4 tables make similar products and a tailing. The low-grade middlings are reground in primary rod mills and returned to the primary table circuit. The primary high-grade middlings from all 6 primary sections are treated on 10 tables, where they are concentrated without additional grinding, producing a concentrate, a high-grade middling, and a low-grade middling on each table and a tailing on the last 4 tables. The low-grade middlings of this circuit are reground in one of the primary rod mills and returned to the primary circuit, while the high-grade middlings are treated in another 10-table circuit. This last circuit yields only concentrates and high-grade and low-grade middlings; the high-grade middlings are circulated and distributed to the last five tables, and the low-grade middlings are deslimed and ground in a secondary rod mill. The rod-mill discharge is treated direct by a rougher flotation machine, and the flotation tailing is returned to a drag classifier in closed circuit with the rod mill.

The system of table concentration just described may seem elaborate, but it is more logical to treat the high-grade middlings by this method than to return and mix them with the feed or low-grade middlings.

Tables are sometimes used incidentally in all-flotation plants. For example, at concentrator I tables are used to recover sulphide minerals that may have been brought over with the froth of the mica flotation circuit, while at concentrator K the flotation tailings are passed over tables whose primary purpose is to serve as pilot machines and to indicate the efficiency of flotation operations, the degree of oxidation, and the degree of grinding obtained. Concentrates from the first cell of the zinc flotation circuit at this plant are spread out on a table for observation.

FLOTATION

Many different flow sheets and types of flotation equipment have been adopted by mining companies in treating lead and zinc ores by flotation. Each ore is a separate problem and requires careful study of chemical composition, the distribution of its mineral constituents, and the fineness of grinding necessary to liberate the mineral particles. Although certain flotation agents used in separation of the economic minerals from the gangue and in differential flotation of two or more sulphide minerals have become more or less standard, the quantities used vary considerably. Conditioning the pulp before flotation also requires careful study, not only for effecting the best economic separation but also for determining the quantity of reagents necessary. These points are clearly indicated in the information circulars covering each concentrator published by the Bureau of Mines.

Classification of methods.—The flotation methods adopted at the various plants may be classified as follows: (1) Treatment of fines collected in thickeners from classifier overflows as a result of dewatering gravity-concentration products; (2) treatment of all fine material, not included in the feed to jigs, and middling and tailings from sand jigs; (3) treatment of all fines, not included or recovered in gravity concentration units, and reground middlings and tailings from jigs and tables; and (4) treatment of entire mill feed by grinding and flotation.

General practices.—Except in method (1) the material for flotation treatment is ground in rod or ball mills, generally in closed circuit with Dorr or Esperanza type classifiers. Table 24 gives some idea of the fineness of material fed to the flotation units at various concentrators.

The classifier overflows generally are collected in Dorr thickeners, and the thickener discharge is pumped to the flotation units. At concentrator P sumps are used instead of thickeners; the classifier overflows are permitted to flow into them, and from there the pulp is pumped to the flotation circuit.

Although at some plants a final flotation product is taken off in one operation without further cleaning, generally a system of roughing and cleaning is practiced. Where both lead and zinc products are recovered the final tailings of the lead circuit comprise the feed to the zinc circuit after conditioning with the proper flotation agents. The rougher concentrates are cleaned one to four times at the different plants, the tailing or middling of the last machine usually being returned to the head of the preceding machine. At many concentrators the rougher tailings are given a final treatment in scavenger machines before they pass to the next circuit or to waste. At concentrators K and P a third circuit is employed to recover a pyrite product from the zinc-circuit tailings.

Three separate flotation units are employed at concentrator D: (1) Flotation of primary slimes producing lead-silver concentrates; (2) grinding and differential flotation treatment of minus 16-mesh original feed and jig middlings producing both lead and zinc concentrates; and (3) grinding and flotation of jig tailings producing lead concentrates only. The ratio of lead to zinc in the milling ore treated at this particular plant is about 8 to 1.

TABLE 24.—*Screen analyses of flotation feed from grinding units, percent*

Concentrator	Screen sizes, mesh							Total
	35	48	65	100	150	200	−200	
A		2.51	8.21	16.15		35.37	37.76	100.00
C		.3	2.2	8.4	11.3	12.2	65.6	100.00
D		.3	2.5	5.5		21.0	70.7	100.00
E	1.7		11.7	14.4	10.0	13.4	48.8	100.00
F			.8	3.4	9.6	10.2	76.0	100.00
G		.21	1.48	6.83	9.41	9.97	72.10	100.00
H			.2	2.3	6.0	10.1	81.4	100.00
I		1.1	9.0	14.7	14.1	13.6	47.5	100.00
J			5.09	9.10	8.86	8.10	68.85	100.00
K	.4	1.3	3.7	9.1	4.2	21.2	60.1	100.00
L			1.37	8.22	17.80	26.02	73.98	100.00
N				7.0		18.0	75.0	100.00
O			5.13	4.23	8.66	6.11	75.87	100.00
P	.93	3.39	5.73	16.26	9.35	25.74	38.60	100.00

FLOTATION REAGENTS

The kind and quantity of flotation reagents used at concentrators producing lead or zinc concentrates, or both, vary, depending largely upon character of ore, operating conditions, and grade of products desired. Table 25 indicates the quantities and kinds of flotation agents used at certain concentrators to produce lead and zinc concentrates.

Conditioning or mixing tanks are usually employed to bring the flotation reagents in intimate contact with the pulp. At some concentrators the flotation or conditioning agents are added at various points in the circuit. For example, at concentrator J part of the soda ash, sodium cyanide, and zinc sulphate used is added to the ball-mill circuit and the remainder to the flotation circuit. At concentrator N the reagents are added as follows:

Reagent	Where added	Quantity per ton of ore treated, pound	Reagent	Where added	Quantity per ton of ore treated, pounds
Soda ash	Ball-mill feed	0.80	Sodium cyanide	Lead cleaner	0.30
Thiocarbanalid	do	.05	Do	Lead recleaner	.10
Pine oil	Classifier overflow	.07	Copper sulphate	Lead-circuit tailings	1.40
Cresylic acid	do	.05	Barrett no. 634	do	.05
Sodium xanthate	Lead rougher	.10	Sodium xanthate	Zinc conditioner	.20
Zinc sulphate	Lead cleaner	.50	Lime	Zinc cleaner	.20

TABLE 25.—*Flotation reagents used to effect differential flotation for producing lead and zinc concentrates at various concentrators*

QUANTITY OF REAGENT PER TON OF ORE TREATED, POUNDS

Flotation reagents	Concentrator							
	D	G	H	I	J	N	O	P
Lead circuit:								
Soda ash	0.10		0.21	1.466	1.90	0.80		(1)
Sodium sulphite				.1818				
Sodium cyanide				.0616	.65	.40	0.060	0.0954
Zinc sulphate	.30	0.88	.20		.90	.50	1.395	
Pine oil						.07		
Cresylic acid	.10	.07			.50	.05	[2] .175	.0307
Aerofloat	.03	.18						
Sodium xanthate				.1826	.10	.10		
Potassium xanthate				.0246			.161	.0239
Thiocarbanalid						.05		.1073
Barrett no. 4			.05					
Minerec "A"			.11					
Lime							.574	
Zinc circuit:								
Copper sulphate	.30	.61	.51	1.210	1.30	1.40	.783	2.2964
Lime				3.337	1.90	.20	1.848	.7956
Pine oil		.03		.125	.01		.151	.1840
Sodium xanthate		.04	.19		.10		.50	
Potassium xanthate				.0399				.0208
Barrett no. 4	.05	.06	.18					
Barrett no. 634						.05		
Aerofloat	.05							.1280

GRADE OF PRODUCTS [3]

	D	G	H	I	J	N	O	P
Lead concentrates:								
Lead ... percent	61.3	70.8	75.2	61.06	59.9	58.37	37.81	53.05
Zinc ... do	5.0	6.9	5.7	5.71	4.0	11.57	13.47	
Zinc concentrates:								
Lead ... do	9.3	3.9	2.7	1.17	2.55	1.81	1.63	
Zinc ... do	46.7	51.3	56.8	51.20	47.60	54.77	54.45	55.88

[1] Added when alkalinity falls too low.
[2] Added to mica circuit ahead of lead circuit.
[3] These are average grades over definite periods, whereas the use of reagents indicated may have been modified during these periods.

At concentrator B, where zinc concentrates only are produced by flotation, and at concentrator F, where lead concentrates only are produced, the following quantities of flotation reagents are used per ton of ore treated:

Concentrator B	*Pounds*
Potassium xanthate	0.26
Pine oil	.08
Copper sulphate	1.14

Concentrator F	*Pound*
Sodium cyanide	0.04
Cresylic acid	.20
Pine oil	.06
Potassium xanthate	.083

The flotation of lead carbonate is described in Information Circular 6320, covering concentrator L. At this concentrator a reagent has been developed to float the oxidized valuable minerals from the gangue, thus eliminating the necessity of sulphidizing. The reagent is said to be made by reconstructing a mixture of an amino compound ($R.NH_2$) and an alcohol with phosphorous pentasulphide at low temperatures. At concentrator L conditioning of the flotation feed with sulphuric acid seems necessary for successful flotation; enough sulphuric acid is added in the ball mill to maintain a pH value of 6.6 to 6.8 in the ball-mill discharge. The other reagents used are sodium silicate and potassium xanthate. In 1929 the grade of lead product averaged 33.43 percent lead with a 92.14-percent recovery of the lead. A volatilization plant is part of this concentrator but is not being used at present.

Soda ash maintains definite alkalinity of the pulp, which in terms of pH values is usually about 8 to 8.5. Zinc sulphate, sodium sulphite,[20] and sodium cyanide are used as depressants of sphalerite and pyrite in the lead circuit. In the zinc circuit copper sulphate is used to reactivate sphalerite which has been depressed in the lead circuit, and lime is employed as a depressant for pyrite.

Often, small amounts of sodium and potassium xanthate are used as collecting agents, and many other flotation agents have proved effective as frothers or collectors.

The use of xanthates at concentrator O [21] is of interest. It is stated that when amyl xanthate is added to the lead circuit only very small quantities need to be added to the zinc circuit and that at times the zinc circuit has been operated for several days without additional xanthate. This condition seems to indicate an excess of xanthate in the lead circuit, yet reducing the amount in this circuit gives lower recoveries. When pentasol xanthate is used in the lead circuit the amount required is said to be much less than when amyl xanthate is used, but considerable additional xanthate must be added to the zinc circuit. Attempts to increase the pentasol xanthate in the lead circuit and thereby reduce the additional xanthate required in the zinc circuit have failed owing to difficulties in the functioning of the lead circuit. It is said, however, that there is no difference in the metallurgical results when each xanthate is used to the best advantage. The xanthate used most recently at this plant is understood to be a secondary butyl xanthate.

DEWATERING OF CONCENTRATES

The concentrates produced by flotation, which contain 5 to 25 percent solids when they enter the thickeners, usually receive preliminary dewatering in Dorr-type thickeners. The discharge from thickeners, which generally contains 50 to 75 percent solids, is fed to filters. The moisture content of the filter cake varies at different plants from about 7 to 12 percent, although at concentrator L it averages between 15 and 16 percent. The filter cake of zinc concentrates is usually 1 or 2 percent higher in moisture content than that of lead concentrates.

In a few instances the filter cake is dewatered further on Lowden driers, or in revolving cylinder driers of the Ruggles-Cole type, as at concentrator P. The sizes of the Lowden driers used at concentrators E and F are 12 by 32 feet and 12 by 28 feet, respectively, and the speeds at which the rakes are operated are 2⅓ and 2⅖ r.p.m. with forward strokes of 10 and 11.5 inches.

Table concentrates are also dewatered by filters at concentrators C and F, and at concentrator E the jig concentrates after being crushed and the dewatered flotation product from the filter are fed to a Lowden drier.

---

[20] It is claimed that at concentrator I an excess of sodium sulphite in the lead circuit has no appreciable effect on the appearance of the froth or on recovery of lead.

[21] Bemis, H. D., Milling Methods and Costs at the Pecos Concentrator of the American Metal Co., Tererro, N.Mex.: Inf. Circ. 6605, Bureau of Mines, 1932, p. 12.

Lime is sometimes used to assist in settling the flotation concentrates in the thickener, and at plant H a small quantity of pine oil is added to the thickened pulp in the filter tank; the pine oil is said to produce a drier cake, eliminating the sticky condition of the cake, which is apt to give trouble in the storage bins.

The filtrates from the filters are usually returned to the respective circuits or pumped to the return-water storage tank at the head of the concentrator.

GRADE OF PRODUCTS AND RECOVERIES

The grade of lead and zinc concentrates produced and the recoveries obtained at various concentrators are shown in table 26. The tenor of the ore, as given under " Heads assays ", represents average assays over a period of time, except for two or three assays in which the figures have been estimated from the available data.

Table 26 shows that good results have been obtained by differential flotation, both as to grade and recoveries, which would not have been possible by gravity methods alone. At several concentrators results are especially noteworthy.

TABLE 26.—*Average grade of concentrates and recoveries at various concentrators treating lead and zinc ores*

Concentrator	Heads assays			Method of concentration	Lead concentrates						Zinc concentrates					
					Assays			Recoveries, percent			Assays			Recoveries, percent		
	Ag, ounces	Pb, percent	Zn, percent		Ag, ounces	Pb, percent	Zn, percent	Ag	Pb	Zn	Ag, ounces	Pb, percent	Zn, percent	Ag	Pb	Zn
A				Gravity [1]								1.97	59.12			
				Flotation								1.90	59.72			
B		1.7	6.35	Gravity [1]		74.0	5.2		85.6			.36	61.50			58.3 } 84.3
				Flotation								.07	59.80			26.0 }
C	4.29	10.42	1.55	Gravity [1] } Flotation }	24.11	58.19	3.13	89.70	89.12	32.15	5.90	3.77	47.37	1.05	0.28	[2] 23.29
D	4.77	8.7	1.2	Gravity [3]	27.09	52.02	3.5	83.8	84.4	63.4	10.1	9.3	46.7	4.7	2.6	55.5
				Flotation	34.6	61.3	5.0	89.5	93.2	33.3						
				Flotation of jig tails	31.4	46.0	4.0	90.2	90.1	68.8						
E			2.9	Gravity [3] } Flotation }									60.19			90.15
F		3.5	.2	Gravity [4]	1.2	75.89	.5	([5])	([5])	([5])	([5])	([5])	([5])	([5])	([5])	([5])
				Flotation	1.9	73.03	2.4									
G	4.32	10.95	2.79	do	27.2	70.8	6.9	88.9	91.5	35.1	5.3	3.9	51.3	3.7	1.0	55.0
H	3.4	9.0	6.5	do	24.9	74.2	5.8	81.1	91.0	9.7	4.8	2.8	56.2	14.2	3.1	85.7
I	9.09	6.21	5.01	do	50.19	61.06	5.71	53.61	95.46	11.08	30.86	1.17	51.2	26.81	1.48	80.76
J	9.44	5.95	4.14	do	87.10	59.9	4.0	65.4	71.3	6.8	14.28	2.55	47.6	9.5	2.7	72.4
L	13.0	11.36	1.65	do	32.05	33.43	2.12	77.26	92.14	40.19						
M	8.17	5.13	5.98	do	74.02	57.05	10.06	75.49	92.59	14.02	12.4	2.79	52.36	11.92	4.23	68.29
N	1.155	1.99	9.93	do	25.22	58.37	11.57	62.95	81.86		1.78	1.81	54.77			90.69
O	3.37	4.93	15.41	do	19.98	37.81	13.47	63.43	81.97	9.35	3.05	1.63	54.45	21.73	7.91	84.64
P		1.5	11.6	do		53.05			59.01				55.88			81.42

[1] Jigs and tables.
[2] Based on mill feed.
[3] Jigs only.
[4] Tables only.
[5] Total lead recovery between 96 and 97 percent, of which table concentrates constitute about 52 percent of total output.

DISPOSAL OF TAILINGS

The tailings from concentrators where all or part of the tailings from jigs and tables go to waste either flow by gravity or are elevated by bucket elevators or belt conveyors to the tailings pile. At plants where the tailings water is to be returned as needed for milling purposes tailings are impounded by dams, forming ponds in which the solids are permitted to settle out, and the clear water is returned by pumps to storage tanks.

The table tailings from concentrator F [22] are dewatered, mixed with part of the flotation tailings, and pumped to slime ponds. The pond overflow is recovered by a vertical concrete standpipe connected at the bottom to a horizontal tunnel extending through the dam. The intake lines of the pumps, used for returning clear water to the mill, are connected with the bottom of the vertical tower through this tunnel.

The tailings from all-flotation plants are either conveyed by gravity in a launder or flume or pumped to waste. Whether or not tailings are impounded depends on the area available for tailings disposal or on the water requirements of the mill. At some plants flotation tailings are permitted to discharge into some nearby creek or to spread out over flat areas, without having been dewatered, while at others the tailings are pumped to thickeners (the discharge passing to waste and the overflow being pumped to storage tank for reuse).

WATER SUPPLY

The water used in concentrating plants may be either fresh or reclaimed water. The fresh-water supply is usually obtained from the mine, from a nearby river or lake, or from wells. It may be pumped through pipe lines, sometimes for considerable distances, or it may flow by gravity in flumes to a storage tank situated above the concentrator.

Reclaimed water is obtained from tailing ponds or dewatering equipment and is returned by pumps to the water-storage tanks. The amount of water reclaimed for milling purposes varies a great deal at different plants, usually depending upon the available fresh-water supply and the cost of conveying the water to the concentrator. For example, at one concentrator [23] the fresh water is pumped from the 1,900-foot mine level; 32 percent of the total water used is fresh water, and the remaining 68 percent is reclaimed from dewatering equipment. At some plants no water is reclaimed for milling purposes.

Practically no bad effects from the use of reclaimed or return water have been experienced in flotation operations at most concentrators. Where the fresh water, obtained from a creek or river, receives sewage from a nearby town considerable difficulty may be experienced in flotation, especially during periods of low water.[24]

---

[22] Coghill, W. H., and O'Meara, R. G., Milling Methods and Costs at a Flat River (Mo.) Mill: Inf. Circ. 6658, Bureau of Mines, 1932, 36 pp.
[23] Wigton, G. H., Milling Methods and Costs at the Concentrator of the Chief Consolidated Mining Co., Eureka, Utah: Inf. Circ. 6320, Bureau of Mines, 1930, 10 pp.
[24] Zeigler, W. L., Milling Methods and Costs at the Lead Concentrator of the Hecla Mining Co., Gem, Idaho: Inf. Circ. 6600, Bureau of Mines, 1932, p. 13.

72670°—35——13

The quantity of water used in concentrators A to P ranges from about 3 to 20 tons per ton of ore treated, depending upon the quantity of ore and the method of treatment.

POWER

Most concentrators now employ electric motive power at 220 to 440 volts for all but the largest motors. Some of the largest motors, such as those used with heavy crushing machinery, employ higher voltages. The power is generally supplied by public-utility companies at a cost of about 0.5 cent to 1.5 cents per kilowatt-hour. At concentrator O the power is supplied at 2,200 volts from the company's own power plant, but for use in lighting and for small motors this voltage is stepped down to 110 and 440 volts, respectively. The power purchased for concentrator P is brought in at 23,000 volts and stepped down to 2,300 and 440 volts for large and small motors, respectively. Both gas engines and electric power are used in operations at concentrator B.

Table 27 indicates the distribution of power at various concentrators.

TABLE 27.—*Distribution of power consumed per ton of ore treated at various concentrators*

Item	Concentrator									
	B		F		I		J		K	
	Power, kw.-hr.	Percent of total	Power, kw.-hr.	Percent of total	Power, kw.-hr.	Percent of total	Power, kw.-hr.	Percent of total	Power, kw.-hr.	Percent of total
Crushing	1.86	15.1	3.4	20.4	2.682	8.6	1.77	7.4	[1] 1.51	5.57
Elevating	3.26	26.4								
Grinding	2.01	16.3	4.0	24.0	11.174	36.0	15.84	65.9	9.15	33.79
Classification										
Jigging	.79	6.4								
Tabling	.43	3.5	1.7	10.2					.25	.92
Flotation	1.22	9.9	4.6	27.5	11.514	37.0	4.85	20.2	8.24	30.40
Dewatering concentrates					4.111	13.2	.39	1.6	.59	2.18
Pumping	2.76	22.4							6.15	22.71
Water supply			2.0	12.0					.72	3.66
Tailings disposal			.9	5.3						
Miscellaneous			[2] .1	.6	1.609	5.2	1.19	4.9	[3] .48	1.77
	12.33	100.0	16.7	100.0	31.090	100.0	24.04	100.0	27.09	100.00

[1] Includes conveying.
[2] Loading of concentrates.
[3] Thickeners.

TABLE 27.—*Distribution of power consumed per ton of ore treated at various concentrators*—Continued

Item	Concentrator							
	L		M		O		P	
	Power, kw.-hr.	Percent of total	Power, kw.-hr.	Percent of total	Power, kw.-hr.	Percent of total	Power, kw.-hr.	Percent of total
Crushing	1.935	3.78	[4] 6.05	21.33	--------	--------	[5] 41.17	53.96
Grinding	19.180	37.48	9.95	34.94	[6] 10.96	35.08	4.74	6.21
Classification	.214	.42			--------	--------	8.20	10.74
Screening	--------	--------	--------	--------	--------	--------		
Conveying	1.611	3.15	--------	--------	--------	--------		
Tabling	--------	--------	--------	--------	2.08	6.66	--------	--------
Flotation	16.050	31.36	7.94	27.96	11.73	37.53	19.61	25.70
Dewatering concentrates	4.890	9.56	1.73	6.10	2.34	7.49	--------	--------
Pumping	5.760	11.25	--------	--------	--------	--------	--------	--------
Water supply	--------	--------	1.90	6.71	1.65	5.29	--------	--------
Tailings disposal	.319	.62	.85	2.96	--------	--------	.86	1.13
Miscellaneous	[7] 1.221	2.38	--------	--------	[8] 2.48	7.95	--------	--------
	51.170	100.00	28.42	100.00	31.24	100.00	76.30	100.00

[4] Includes screening.
[5] Crushers and rolls.
[6] Crushing not included as ore is crushed at mine to 1½-inch size.
[7] Control and mill sampling 0.712 kw.-hr.
[8] Thickening, sampling, lighting, and laboratory.

LABOR

The number of men necessary for the operation of a concentrator varies considerably. In general, a plant treating 5,000 tons per day will require a much smaller number of men proportionately than a plant treating only 500 tons per day, and obviously a plant in which two or more different concentrates are produced will require a larger number of men than one of equal capacity producing only one product under like conditions.

Table 28 indicates the labor costs at seven concentrators of different capacities, in terms of man-hours per ton and of tonnage treated per man, in each of the various departments of the milling operation.

Table 29 gives average over-all labor costs in similar terms for 14 lead concentrators, 19 zinc concentrators, and 83 concentrators producing both lead and zinc concentrates.

Labor costs per ton of ore treated are generally lower for plants treating relatively large tonnages. The wages paid for labor, which in the lead-zinc mining districts consists mostly of native Americans, range in different districts from about $3 per 8-hour shift for mill helpers to $5 or $6 for head operators and shift foremen. Some plants pay a bonus based on the price of metal.

TABLE 28.—*Distribution of labor in man-hours per ton treated and in tons per 8-hour shift*

Item	Concentrator and tonnage treated					
	B, 40 tons per hour			F, 208 tons per hour		
	Man-hour per ton	Tons per 8-hour shift	Percent of total	Man-hour per ton	Tons per 8-hour shift	Percent of total
Sorting	(1)	(1)				
Crushing	0.0374	214	9.1	0.0235	340	22.0
Grinding	.0187	428	4.5	.0141	567	13.25
Classification, screening, and conveying	.0187	428	4.5			
Concentration:						
Gravity	[3].1496	[3] 53.5	36.4	[4].0141	[4] 567	13.25
Flotation	.0748	107	18.2	.0094	850	8.8
Dewatering concentrates	.0374	214	9.1	[5].0047	[5] 1,700	4.4
Sampling				.0047	1,700	4.4
Tailings disposal	.0374	214	9.1			
Maintenance				.0283	283	26.6
Supervision				.0078	1,020	7.3
Power	.0374	214	9.1			
Total labor	.4114	19.45	100.0	.1066	75	100.00

Item	Concentrator and tonnage treated					
	H, 50 tons per hour			I, 17 tons per hour		
	Man-hour per ton	Tons per 8-hour shift	Percent of total	Man-hour per ton	Tons per 8-hour shift	Percent of total
Crushing	0.114	70	21.4	0.1015	78.79	16.5
Grinding	[2].114	[2] 70	21.4	.0768	104.14	12.5
Concentration: Flotation	.176	45.5	33.0	.0768	104.14	12.5
Dewatering concentrates	.030	267	5.6	.0768	104.14	12.5
Weighing and loading				.0512	156.11	8.35
Sampling				[6].0256	[6] 312.43	4.2
Tailings disposal				.0512	156.11	8.35
Maintenance				.0512	156.21	8.35
Supervision				.0256	312.43	4.2
Assaying				.0512	156.21	8.35
Warehouse				.0256	312.43	4.2
Miscellaneous	.099	81	18.6			
Total labor	.533	15	100.0	.6135	13	100.00

Item	Concentrator and tonnage treated								
	J, 13 tons per hour			L, 7 tons per hour			P, 20 to 25 tons per hour		
	Man-hour per ton	Tons per 8-hour shift	Percent of total	Man-hour per ton	Tons per 8-hour shift	Percent of total	Man-hour per ton	Tons per 8-hour shift	Percent of total
Sorting	0.027	296	8.0						
Crushing	.047	170	113.9	0.1040	77.0	11.5	0.0955	83.3	31.4
Grinding	.078	102.6	23.1	.0995	80.4	11.0	.0579	138.2	19.0
Classification, screening, and conveying							.0222	360	7.3
Concentration: Flotation	.097	82.5	28.7	.3125	25.6	34.5	.1067	75	35.0
Dewatering concentrates	[6].022	[6] 364	6.5	.0994	80.5	11.0			
Sampling				.0957	83.6	10.6			
Tailings disposal				.0762	104.9	8.4	.0222	360	7.3
Miscellaneous	.067	119.4	19.8	.1180	67.7	13.0			
Total labor	.338	23.7	100.0	.9053	8.83	100.0	.3045	26.3	100.0

[1] Repairs included in the different items.
[2] Includes classifying.
[3] Jigs and tables.
[4] Table concentration.
[5] Includes loading.
[6] Includes mixing reagents.

TABLE 29.—*Over-all milling costs in man-hours per ton and tons per man-shift*

Number of mills reporting	Total ore milled, tons	Principal metal produced	Average ore milled per man-shift, tons	Average man-hour per ton
14	7, 359, 063	Lead	33. 170	0. 2412
19	2, 630, 082	Zinc	[1] 13. 135	. 6091
83	7, 957, 868	Zinc and lead	[1] 21. 437	. 3739

[1] Some mills reported 10-hour mill shifts; figures from these mills have been adjusted to an 8-hour-shift basis.

MILLING COSTS

Cost data covering milling operations at a number of concentrators are given in table 30. The total milling cost per ton of ore treated is divided into different items of operation; and the cost of each item is subdivided into labor, supplies and miscellaneous, and power. As the method of keeping costs by the different mining companies varies it is usually difficult to present cost data covering the operations of several concentrators in one table wherein the separate cost items are in complete accord; however, in table 30, where these items for the different plants do not correspond, or two or more items are covered by a single cost figure, this fact is explained by a footnote.

Table 31 gives the milling costs in terms of combined lead and zinc metal contained in the concentrates produced, that is, per pound of lead in the lead concentrates and per pound of zinc in the zinc concentrates, the zinc in lead concentrates and lead in the zinc concentrates not being included. In this table no credits are included in the cost of combined lead and zinc metal for precious or other metals; however, the quantities of silver, gold, and copper contained in the lead and zinc concentrates are given under "Remarks" where such data were available.

TABLE 30.—*Milling costs per ton of ore treated at various concentrators*

Item	Concentrator and period covered													
	B; August, September, October 1929				C; year 1929				E; year 1929	G; year 1930				H; summarized average
	Labor	Supplies and miscellaneous	Power	Total	Labor	Supplies and miscellaneous	Power	Total	Total costs	Labor	Supplies and miscellaneous	Power	Total	Total costs
Crushing	$0.014	$0.049	$0.023	$0.086	$0.035	$0.034	$0.027	[1] $0.096	$0.0671	$0.037		$0.017	$0.054	$0.060
Grinding		.054	.024	.078	.019	.109	.045	.173	.0532	.070	$0.210	.131	.411	.130
Elevating and conveying	.014	.012	.039	.065				•	[3] .0119					.020
Screening and classifying					.019	.023	.006	[2] .048						.020
Jigging	.052	.002	.010	.064	.055	.042	.041	.138	.0634					
Tabling	.039	.015	.005	.059	.010	.010	.011	.031						
Flotation	.036	.039	.015	.090	.066	.135	.032	.233	.0773	.058	.120	.114	.292	.242
Dewatering concentrates									[4] .0288	.018		.013	.031	[5] .020
Disposal of tailings									.0045					
Water supply		.004	.033	.037					.0306	.009		.007	.016	.010
Pumping									[6] .0044					.020
Power														.270
Sampling										.003		.002	.005	
Assaying		.018		.018										
Maintenance														[7] .090
Supervision										.043			.043	
Miscellaneous		.007		.007						.023	.030		[8] .053	[9] .130
Overhead								.359						
Total	.155	.200	.149	.504	.204	.353	.162	1.078	.3412	.261	.360	.284	.905	1.012

Item	Concentrator and period covered—Continued																
	J; March 1930				L; year 1929				M; Oct. 1, 1929–Apr. 1, 1930				N; September, October, 1929	O; year 1930			
	Labor	Supplies and miscellaneous	Power	Total	Labor	Supplies and miscellaneous	Power	Total	Labor	Supplies and miscellaneous	Power	Total	Total costs	Labor	Supplies and miscellaneous	Power	Total
Sorting	$0.017	$0.001		$0.018									$0.0472				
Crushing	.035	.021	$0.023	.079	$0.0854	$0.0183	$0.0195	$0.1232	$0.1493	$0.0954	$0.0765	[10] $0.3212	.1110				
Grinding	.054	.181	.210	[11] .445	.0778	.1830	.1938	.4546	.0233	.1029	.1258	[12] .2520	.3553	$0.0500	$0.1546	$0.0670	$0.2716
Screening and classifying					.0053		.0022	.0075									
Tabling														.0226	.0028	.0128	.0382
Flotation	.073	.365	.064	.502	.2299	1.2652	.1621	1.6572	.1347	.3631	.1003	.5981	.5262	.1157	.3320	.0723	.5200
Dewatering concentrates	.015	.001	.005	.021	.0993	.0195	.0494	.1682	.0204	.0080	.0219	.0503	[13] .0817	.0202	.0035	.0161	[14] .0398
Disposal of tailings					.0535	.0054	.0032	.0621	.0408	.0110	.0105	.0623		.0002	.0006		.0008
Water supply						.2540		.2540	.0506	.0008	.0241	.0755		.0003	.0432	.0102	.0537
Sampling					.0742	.0143	.0072	.0957						.0124	.0022	.0059	.0205
Assaying						.1710		.1710					.1114		.0595		.0595
Supervision													.0731	.0888	.0007		.0895
Miscellaneous	.050	.151	.016	[15] .217	.1049	.0389	.0796	.2234	.1675	.6751		[16] .8426		.0201	.0478		[17] .0679
Overhead													[19] .2447		.0293		[18] .0293
Total	.244	.720	.318	1.282	.7303	1.9696	.5170	3.2169	.5866	1.2563	.3591	2.2020	1.5506	.3303	.6752	.1843	1.1898

1 Includes conveying.
2 Screening only.
3 Tramming crushed ore to concentrator.
4 Includes drying.
5 Filters and thickeners.
6 Pumping slimes to flotation plant.
7 Plant maintentnce and repair shops.
8 Includes $0.013 for heating.
9 Includes $0.01 for launders, $0.02 for transmission, $0.01 for lubrication, $0.02 for loading,, $0.04 for lighting an dheating, and $0.01 for change.
10 Includes screening.
11 Includes classifying.
12 Includes classifying.
13 Filters and thickeners.
14 Includes thickening.
15 General mill expense.
16 Includes assaying, overhead, shop, and operation of pumping plant and pipe line from river when used.
17 Includes $0.0071 for heating, $0.018 for mill clean-up, $0.032 for experimental work, $0.004 for lubrication, and $0.0045 for lighting.
18 Royalties.
19 General expense.

TABLE 31.—*Milling costs per pound of combined lead and zinc metals at various concentrators*

Concentrator	Period covered	Ore milled, tons	Lead concentrates produced, tons	Zinc concentrates produced, tons	Lead in lead concentrates, pounds	Zinc in zinc concentrates, pounds	Combined lead and zinc, pounds	Total cost of milling	Cost per pound of combined lead and zinc	Remarks
B	Aug. 1 to Oct. 30, 1929.	24,974	49	2,192	72,520	2,673,176	2,745,696	$12,586.90	$0.00458	No precious metals.
C	1929	451,111	72,002.56	3,448.19	83,796,579	3,266,815	87,063,394	486,297.66	.00558	Lead concentrates contained 24.11 ounces Ag per ton and zinc concentrates 5.9 ounces Ag per ton, or a total of 1,756,671 ounces Ag in combined concentrates.
C	1931 [1]	460,366	85,542	4,296	88,253,071	4,642,383	92,895,454	384,098.47	.00414	Lead concentrates contained 19.95 ounces Ag per ton and zinc concentrates 4.36 ounces Ag per ton, or a total of 1,726,293 ounces Ag in combined concentrates.
E	1929	624,759	----------	[2] 27,136	----------	32,666,740	32,666,740	213,167.77	.00653	No precious metals.
G	1930	97,447	12,244	2,073	16,676,328	2,139,336	18,715,664	88,189.54	.00471	Lead concentrates contained 30.7 ounces Ag per ton and zinc concentrates 5.6 ounces Ag per ton, or a total of 387,500 ounces Ag in combined concentrates.
H	do	361,372	39,833	36,008	59,112,172	40,472,992	99,585,164	371,851.79	.00373	Lead concentrates contained 24.9 ounces Ag per ton and zinc concentrates 4.8 ounces Ag per ton, or a total of 1,164,680 ounces Ag in combined concentrates.
J	March 1930	9,930	703.5	625	842,793	595,000	1,437,793	12,730.26	.00886	Lead concentrates contained 87.1 ounces Ag per ton and zinc concentrates 14.28 ounces Ag per ton, or a total of 70,200 ounces Ag in combined concentrates.
L	1929	37,834	11,852.7	----------	7,924,715	----------	7,924,715	121,708.19	.01536	Lead concentrates contained 0.255 ounce Au and 32.05 ounces Ag per ton, or a total of 2,666.86 ounces Au and 379,879 ounces Ag.
M	Oct. 1, 1929, to Apr. 1, 1930.	41,246	3,435	3,215	3,919,335	3,366,748	7,286,083	90,823.69	.01246	Lead concentrates contained 0.674 ounce Au and 74.02 ounces Ag per ton and zinc concentrates 0.07 ounce Au and 12.4 ounces Ag, or a total of 2,504.25 ounces Au and 294,125 ounces Ag.
N	September and October 1929.	9,222	255.2	1,465.8	297,920	1,605,637	1,903,557	14,299.63	.00752	Lead concentrates contained 25.22 ounces Ag per ton and 3.68 percent Cu and zinc concentrates 1.78 ounces Ag per ton and 1.55 percent Cu, or a total of 9,045 ounces Ag and 64,223 pounds of Cu.
O	1930	151,943	[2] 16,244	[2] 36,406.6	12,283,715	39,645,584	51,929,299	180,781.82	.00348	Lead concentrates contained 0.768 ounce Au and 19.98 ounces Ag per ton and 4.02 percent Cu and zinc concentrates 0.036 ounce Au and 3.05 ounces Ag per ton and 1 percent Cu, or a total of 13,783.8 ounces Au, 435,693 ounces Ag, and 2,034,004 pounds Cu.

[1] Bunker Hill & Sullivan Mining & Concentrating Co., Forty-fourth Annual Report for the Year Ended Dec. 31, 1931.
[2] Calculated.

CONCLUSION

In the body of this bulletin the authors have endeavored to indicate the extent to which the physical, mineralogical, and other characteristics of different types of lead and zinc deposits govern the methods and costs of exploring, sampling, developing, stoping, and milling the ores.

Descriptions of typical ore deposits and of the methods employed in their exploitation, together with actual operating costs, have formed the basis of the discussions. Much of the descriptive matter and data have been taken from information circulars previously published by the Bureau, most of which were written by members of the staffs of the operating companies. The discussions, while based largely on the data contained in these circulars, have been supplemented in many instances by the personal observations of the authors of this bulletin.

COSTS OF MINING AND MILLING

Costs for the various operations connected with mining and milling and summaries of mining and milling costs at a number of mines and mills have been given. In only 1 or 2 instances were dollar costs of both mining and milling available for a given property for the same year, and no tables have been presented in which mining and milling costs are combined; however, data were available from which the man-hours per ton and tonnage per man-shift could be calculated for mining and milling at a large number of properties during 1930. These data have been combined and grouped under lead, zinc, and lead-zinc mines in table 32, the figures given being weighted averages in each instance. Labor usually accounts for about 60 percent of the total mining cost and 20 to 40 percent, or an average of about 30 percent, of the total milling cost, and the figures in the table are therefore of interest in a study of production costs.

COSTS PER POUND OF LEAD AND ZINC

With the exception of table 31, page 194, the costs so far presented have been costs per ton of ore mined or per ton of ore milled. Since the value of an ore is based upon the quantity and value of the recoverable metals contained, costs per unit of metal produced have greater economic significance than costs per ton of ore mined and milled. Unfortunately, data upon which to figure costs on this basis are available from comparatively few companies. Most companies do not publish their costs, and where they do the figures are with few exceptions incomplete.

For either straight lead ores or straight zinc ores costs of production per pound of contained metal up to the point where the shipping ore or concentrates leave the mine or mill en route to the smelter may be calculated readily from the cost sheets. For a complex ore containing lead and zinc, copper, and precious metals there may be a difference of opinion as to how costs should be allocated between the different base metals and how to apply credits for the precious metals. A system of cost accounting which would be rational for one company mining a certain type of ore and producing one or more metals in the form of concentrates would not usually be suitable for another company mining a different type of ore and producing a different combination of metals. Furthermore, a company operating its own smelter ordinarily would employ a different cost set-up than one shipping its product to a custom smelter.

Rational presentation of the costs of a number of companies in tabular form would be difficult if not impossible, even if the majority of companies published their detailed costs. Differences in accounting systems, complexities introduced by differences in the types and grade of the ores, freight and smelter charges, and smelter deductions, penalties, and credits all contribute to the difficulty of reducing costs to a common denominator.

The subject of smelting is beyond the scope of this paper and has not been considered. Obviously, the costs of producing lead and zinc in usable form are incomplete unless the smelting cost is included. As pointed out later, published data in one instance show a smelting cost equal to the combined mining and milling cost. Nevertheless, any data available on the cost of mining and milling per pound of lead and zinc produced are deemed of sufficient interest to be presented, incomplete as they admittedly are.

Table 33 gives production costs per pound of lead and zinc contained in direct shipping ore or/and concentrates for mining, milling, general expense, plant depreciation, and taxes. These costs have been computed from published annual reports of producing companies.

For simplicity and uniformity, the costs are based only on the lead contained in lead concentrates and the zinc contained in zinc concentrates, the lead in zinc concentrates and zinc in lead concentrates being disregarded; also, no account is taken of the value of precious metals, which at some mines may provide a large revenue and, if entered as a credit against the cost of producing the base metals, would materially reduce tabulated costs. The footnotes furnish some data regarding the production of byproduct metals which are not considered in the body of the table.

To indicate what costs may be beyond those covered in table 33 (that is, through the smelting stage of processing), the reader is referred to the annual report of the Bunker Hill & Sullivan Mining & Concentrating Co. for the year ended December 31, 1931. The approximate cost and selling value of the year's production are tabulated on page 20 of that report. Costs are all charged against lead production, although the production for the year included 1,759,251 ounces of silver and 10,019,605 pounds of zinc. The costs given are $0.224 per pound of lead (Kellogg cost, essentially the cost of mining and milling) and $0.0234 per pound of lead for smelting, or a total cost of $0.0458 per pound of lead. Excluding the value of the zinc and silver, the cost per pound of lead for smelting was therefore slightly greater than the mining and milling cost.

TABLE 32.—*Costs of mining and milling in tons per man-shift and man-hours per ton*

Number of mines and mills reporting	Total ore, tons	Principal metal	Tons per man-shift				Man-hours per ton			
			Underground	Surface	Mill	Total	Underground	Surface	Mill	Total
13	7,211,115	Lead	9.030	75.439	33.314	6.494	0.8859	0.1060	0.2401	1.232
19	2,630,082	Zinc	5.1396	29.149	[1] 13.135	3.278	1.5564	.2744	.6091	2.440
83	6,547,219	Lead and zinc	4.524	21.464	20.595	3.162	1.7683	.3728	.3884	2.5295

[1] Some mines reported 10-hour mill shifts; figures from these mines have been adjusted to an 8-hour shift basis.

TABLE 33.—*Operating costs per pound of lead and zinc (not including smelting)*

Company	Year	Lead in lead concentrates, pounds	Zinc in zinc concentrates, pounds	Combined lead and zinc pounds	Cost per pound of combined lead and zinc					
					Mining and development	Milling	Mining and milling	General expense, local taxes, etc.	Depreciation and Federal taxes	Total
Bunker Hill & Sullivan	1931	88,253,071	4,642,383	[1] 92,895,454	$0.0171	$0.0041	$0.0212	([2])		$0.0212
Hecla	1931	47,893,419	758,322	[3] 48,651,741	.0132	[4] .0025	.0157	$0.0041		.0198
Tintic Standard	1931	[5] 27,986,298		27,986,298	.0214	.0033	.0247		$0.0019	.0266
Silver King Coalition	1931	[6] 25,725,936	12,292,038	[7] 38,017,974			.0301	.0019	.0009	.0329
Park-Utah	1930	28,040,599	34,073,606	[8] 62,114,205	.0213	[9] .0017	.0230	.0020	.0017	.0267
Treadwell Yukon	1931	8,287,978	5,994,946	[10] 14,282,924						.0230
Tri-State district[11]	1930	[12] 72,787,200	[13] 473,350,800	546,138,000			.0189	[14] .0037	[15] .0043	.0269

[1] 209,400 pounds of lead in zinc concentrates and 5,284,804 pounds of zinc in the lead concentrates not included; 1,725,733 ounces of silver also produced.
[2] Included in mining and milling.
[3] Includes metal in direct smelting ore. Concentrates and smelting ore contained 1,271,869 ounces of silver.
[4] Milling and ore haulage.
[5] Lead in ore instead of in concentrates. Ore contained 2,995.39 ounces of gold, 1,991,745 ounces of silver, and 424,754 pounds of copper.
[6] Includes 40,530 pounds of lead in lease ore.
[7] 390,924 pounds of lead in zinc concentrates, 793,010 pounds of copper, 2,203.47 ounces of gold, and 1,531,236 ounces of silver not included.
[8] Total lead and zinc content; contained also 804,957 pounds of copper, 17,679 ounces of gold, and 1,598,983 ounces of silver.
[9] Milling and ore expense.
[10] Concentrates contained 922.59 ounces of gold and 622,177 ounces of silver.
[11] Preliminary figures, Tri-State Lead and Zinc Producers Association.
[12] Estimated from preliminary figures of Tri-State Lead and Zinc Producers Association, assuming 80 percent lead in lead concentrates.
[13] Estimated on assumption that zinc concentrates average 60 percent zinc.
[14] Includes taxes excluding income tax, insurance, and royalty of $0.0026 per pound.
[15] Depreciation and depletion.

INDEX

Page

All-flotation plants, tables, use 180
American Metal Co. *See* New Mexico.
American Zinc Co. *See* Tennessee.
Andrus, D. E., work 162
Arizona, Eagle-Picher Lead Co., Montana concentrator, capacity 162
flow sheet 164
milling method 162
ore treated 162
products, grade 165
ratio of concentration 165
recoveries 165
Netta mill, capacity 147
flow sheet 148
milling method 147
ore treated 147
products, grade 147

Ball mills, feed 172
operating data, table 175
Ball-mill products, screen analyses, tables 173, 174
Balmat mill. *See* New York, St. Joseph Lead Co.
Banks, Leon M., work 52, 54
Barker district. *See* Montana.
Barr mine. *See* Tri-State district.
Battle Mountain district. *See* Colorado.
Bemis, H. D., work 167, 184
Berg, J. E., work 57, 112
Black, W. S., work 85
Blackburn, W. H., work 160
Black Hawk concentrator. *See* New Mexico.
Black Rock mine. *See* Montana.
Block P mine. *See* Montana.
Borcherdt, W. O., work 62, 107, 121, 128
Boutwell, J. M., work 27
Bowen, —, work 45
Brecciated zones, mineralization 15
Brown, U. E., work 26, 110
Brush, G. J., work 8
Bunker Hill & Sullivan Mining & Concentrating Co. *See* Idaho.
Butler, B. S., work 29
Butte district. *See* Montana.

Calkins, F. C., work 25
Canam Metals Corporation. *See* Oklahoma.
Cherokee County. *See* Kansas.
Chief Consolidated mines. *See* Utah.
Chimney ore bodies, development 63
Churn drills, exploration 50
use 47
Classifier overflow, fineness 178
pulp density 178
Classifying, methods employed 177
Coady, L. J., work 26, 57, 129
Coeur d'Alene district. *See* Idaho.
Coghill, W. H., work 154, 187
Collecting agents, use 184
Colorado, Battle Mountain district, cut-and-fill stoping 128
exploration 62
sampling practice 78
square-set stoping 121

Page

Colorado, etc.—Continued.
Empire Zinc Co., cut-and-fill stoping 128
figure 125
pillar removal 128
stoping methods 107
Eagle Canyon, gold-silver ores 62
Eagle mine, exploration 62
Gilman, horizontal cut-and-fill stoping 128
figure 125
sampling practice 78
square-set stoping 121
Concentrates, dewatering 184
grade, table 186
prices 73
Concentrating plants, gravity methods, use 178
water used 187
Concentrating tables, use 179
Concentration, methods 146
Concentrators, flotation units 181
tables 182, 183
labor costs, tables 190, 191
labor distribution, table 190
labor employed 189
lead and zinc, ball- and rod-mill data, table 175
milling operations, costs 191
tables 192, 194
milling practice, flow sheets 147
summary 147
power employed 188
distribution, table 188
recoveries, table 186
tailings disposal 187
Cooper district. *See* New Mexico, Pecos district.
Coy, Harley A., work 54, 102
Crabtree, E. H., Jr., work 147
Cretaceous age, ore deposits 16
Crosscutting, cost 90
summary 64
Crosscutting data, table 92
Crosscuts, size 89
Crusher products, screen analyses, tables 171, 172
Crushing, discussion 171
Cushing, H. P., work 41
Cut-and-fill stoping,
advantages 131
disadvantages 131
discussion 123
figures 124, 125, 127, 130
sequence of operations 131
summary 130

Dailey, M. J., work 28, 58, 112
Dalton, M. P., work 158
Dana, E. S., work 8
DeMier level jig, description 179
Deposits, flat-lying 6
igneous metamorphism 6
replacement 6
tabular 6
Development, cost 96
per ton of ore mined, table 98
definition 80
discussion 80
ore produced per foot, table 97
type, determining factors 81

Page
Diamond drilling, cost 49
use 47, 49
Dobbel, Chas. A., work 60
Dorr-type thickeners, use 184
Drifts, location 94
size 89
Drifting, cost 90
per foot, table 92
data, table 92
summary 64
Drill-hole samples, assays, table 72
Drilling, advantages 65
deep-hole, costs 54
summary 63
Drill samples, definition 67

Eagle Canyon. *See* Colorado.
Eagle Mine. *See* Colorado.
Eagle-Picher Lead Co. *See* Arizona.
Eastern Tennessee district. *See* Tennessee.
Edwards district. *See* New York.
Edwards Mine. *See* New York.
Emmons, W. H., work 8, 14
Empire Zinc Co. *See* Colorado.
Enrichment zone, character 13
Estimating practice, examples 68
summary 79
table 80
Exploration, definition 47
discussion 47
methods, examples 48
summary 63
surface 47
underground, methods 48
Exploratory openings, use 81

Face samples, difinition 67
Fahrenwald cones, slime separation by, efficiency, table 178
Faults, ore shoots 14
Federal Mining & Smelting Co. *See* Idaho.
Flat River mill. *See* Missouri.
Float, as surface indication of ore 12
Flotation, general practices 181
methods, classification 181
Flotation feed, screen analyses 182
Flotation process, discussion 170
Flotation reagents, quantities used 182
kinds used 182
tables 182, 183
Folds, mineralization 15
Ford, Wm. E., work 8
Foreman, Charles H., work 58, 75, 129
Formational contacts, mineralization 15
Fowler, F. M., work 14, 19, 20
Fowler, George M., work 53
Franklin Furnace district. *See* New Jersey.
Frothers, use 184
Frue vanners, efficiency, tables 178

Galena, occurrence 17
Gardner, E. D., work 86, 91, 109
Gilman. *See* Colorado.
Gordon, C. H., work 36, 46
Grab samples, definition 67
Graton, L. C., work 36, 46
Gravity concentration, discussion 178
jigging method 178
table method 179
Grinding, discussion 172
Ground Hog mine. *See* New Mexico.

Haight, C. M., work 134
Hammer drill, deep-hole, use 60
use 48, 56
Hand sorting, discussion 170
Handy, R. S., work 150, 177
Heading-and-bench mining, discussion 101
Healy, A. J., work 33, 61, 77, 117
Hecla Mining Co. *See* Idaho.

Page
Heikes, V. C., work 29
Henry, R. J., work 85
Hewitt, D. F., work 14, 39
Hewitt, E. A., work 28, 59, 114
Hoag, C., work 34, 61, 77, 85, 126
Hughesville concentrator. *See* New York, St. Joseph Lead Co.
Hydraulic-classifier products, screen analyses, table 177
tonnages, table 177

Idaho, Bunker Hill & Sullivan mine, exploration 57
ore bodies, occurrence 26
section 24
square-set-and-fill mining 110
stages, figure 111
square-set stope, figure 109
stope development 95
cross-section 95
Bunker Hill & Sullivan Mining & Concentrating Co., annual report, reference 196
costs, table 197
West mill, capacity 150
flow sheet 151
milling methods 150
ore treated 150
products, grade 150
ratio of concentration 150
recoveries 150
water consumption 150
Coeur d'Alene district, Algonkian rocks, section, table 25
Belt series, ore deposits 26
cut-and-fill stoping 129
estimation practice 75
exploration 57
ore, tenor 27
ore deposits, geology 24
ore shoots, characteristics 26
sampling 75
square-set stoping 110
Federal Mining & Smelting Co., Morning concentrator, capacity 158
flow sheet 158
milling method 158
ore treated 158
products, grade 158
ratio of concentration 158
recoveries 158
Page concentrator, capacity 156
flow sheet 157
milling method 156
ore treated 156
products, grade 156
ratio of concentration 158
recoveries 158
Hecla mine, exploration 58
sampling 75
stope preparation 95
stringer sets and filling, use 109
stull-sets-and-fill stoping 129
figure 130
Hecla Mining Co., costs, table 197
crushing and sorting plant, flow sheet 152
lead concentrator, capacity 150
flow sheets 152, 153
milling methods 154
ore treated 150
products, grade 154
ratio of concentration 154
recoveries 154
ore treatment, flow sheet 153
Morning mine, exploration 57
sampling 75
stringer sets and filling, use 109
stringer-sets-and-fill stoping 129
figure 125
ventilation raises 95
vertical longitudinal projection 55
Page mine, exploration 57
sampling 75
shaft development 83
square-set stoping practice 112

Page
Jackson, C. F., work 50, 68, 70, 79, 82, 85, 90, 107
Jigging, methods 178
sized-material concentration 179
roughing and cleaning 179
Johnson, J. Fred, work 86
Joint fissures, mineralization 15
Jones, E. L., work 26

Kansas, Cherokee County, prospect holes, drilling 51
See also, Tri-State district.
Keener, Oliver W., work 85
Kemp, J. F., work 45
Knaebel, J. B., work 41, 54, 68, 74, 79, 85, 90, 105, 168, 174

Labor, cost 189
tables 190, 191
Lary, Howard N., work 62, 78
Leached zone, character 12
Lead, milling, introduction 1
mining, introduction 1
mining districts 16
production 2
costs, table 197
value 2
Lead carbonate, flotation 183
Lead deposits, changes with depth 12
geologic age 16
geology 6
mineralogy 6
minerals contained, table 8
Lead minerals, occurrence, table 8
Lead ore, classifying 177
concentrates produced, table 186
concentration, methods 146
deposits, types 6
principal sources, map 5
production 3
tables 4
sizing 177
Lead ore bodies, outcrops 11
Lead-zinc concentrators, jigging methods 178
Lead-zinc ore, production 3
tables 4
Levels, development, discussion 86
figure 96
interval, discussion 86
short, advantages 89
disadvantages 89
table 88
plan 93
influence of stoping method 90
Lewis, Robert S., work 75
Lindgren, W., work 8, 14, 29, 33, 36, 44, 45
Loughlin, G. F., work 29
Lowden driers, use 184
Lyden, J. P., work 14, 19, 20, 53

Matson, J. T., work 34, 61, 77, 85, 126
McGilvra, D. B., work 33, 61, 77, 117
Metallic minerals, associated, occurrence, table 9
Midvale concentrator. *See* Utah, U. S. Smelting, Refining & Mining Co.
Mill-hole system, discussion 103
Milling, costs 191
conclusions 195
over-all, table 191
per pound of metals, table 194
per ton of ore treated, table 192
Milling operations, discussion 170
Mine development, cost 96
Minerals, in lead and zinc ores 7
Mines, development 80
lead and zinc, drifting and crosscutting, table 92
employing cut-and-fill stoping 123
employing open stoping, list 106
mining costs 141
tables 142, 145
stope production 101
stoping costs 137
tables 139, 140

Page
Mine valuation, method 71
probable net profit, table 73
Mining, cost 141
conclusions 195
tables 142, 145. 197
Missouri, Bonneterre limestone, ore occurrence 17
Chester formation, ore deposits 19
Flat River mill, concentrator, capacity 156
flow sheet 155
milling method 156
ore treated 154
products, grade 156
recoveries 156
water consumption 156
No. 8 mine, ore estimation 68
polygonal areas, figure 69
sampling practice 68
stoping methods, figure 102
St. Louis Smelting & Refining Co., diamond drilling 49
southeastern district, estimating practice 68
exploration 47, 48
geological formations, cross-section 17
heading-and-bench system 101
figure 102
mine workings, plan 19
cross-section 20
ore deposits, geology 17
sampling 68
shafts 83
test-hole drilling 49
typical mineralization, figure 18
See also Tri-State district.
Montana, Barker district, cut-and-fill stoping 125
exploration 61
ore, assay 34
ore deposits, geology 34
sampling 77
Black Rock mine, booms 119
cribbed bulkheads 118
estimating practice 77
exploration 61
geology 33
ground conditions 117
heavy ground, top slicing with square sets 120
main hoisting shaft 84
sampling practice 77
square-set stoping practice 117
figure 118
underhand square-set stoping 119
figure 119
overhand square-set stoping 117
figure 118
Block P mine, exploration 61
horizontal cut-and-fill stoping 125
figure 124
main shaft 84
rock walls, use 126
sampling 77
stope preparation 96
Butte district, copper production 32
estimating practice 77
exploration 61
ore deposits, geology 32
prospecting 61
sampling practice 77
square-set stoping 117
veins, characteristics 33
Silver Bow County, geology 32
Montana concentrator. *See* Arizona, Eagle-Picher Lead Co.
Morning concentrator. *See* Idaho, Federal Mining & Smelting Co.
Morning mine. *See* Idaho.

Netta mill. *See* Arizona, Eagle-Picher Lead Co.
Netzeband, W. F., work 53, 71, 85

	Page
Nevada, Treadwell Yukon Co., costs, table	197
Tybo concentrator, capacity	160
flow sheet	160
milling method	160
ore treated	160
products, grade	160
ratio of concentration	161
recoveries	161
water consumption	161
Yellow Pine district, ore deposits, geology	39
Yellow Pine mine, plan	40
section	40
New Jersey, Franklin Furnace district, ore deposits, geology	44
plan	45
section	45
shrinkage stoping	134
top slicing	135
figure	136
zinc deposits	45
Franklin mine, shrinkage stoping	134
figures	132, 133
Sterling Hill, ore bodies	44
shrinkage stoping	134
typical ore, analysis	44
Newland, D. H., work	41
New Mexico, American Metal Co., Pecos concentrator, capacity	167
flow sheet	167
milling method	167
ore treated	167
products, grade	167
ratio of concentration	168
recoveries	168
water consumption	168
Black Hawk concentrator, ore milled, assay	37
mineralogy	37
Black Hawk Consolidated mill, capacity	165
flow sheet	166
milling method	165
ore treated	165
products, grade	165
ratios of concentration	165
recoveries	165
water consumption	165
Ground Hog mine, estimating, practice	78
exploration	62
geology	37
rill stoping	121
sampling	78
section	38
shaft	84
square-set stoping	121
figure	120
stope development	96
Magdalena, ore deposits	46
Pecos district, cut-and-fill stoping	126
estimation practice	78
exploration	61
ore deposits, geology	34
sampling	77
Pecos mine, cut-and-fill stoping	126
section	127
estimating practice	78
exploration	61
geology	34
metal production	35
ore bodies, development, figure	35
sampling	77
shafts	84
stope preparation	96
stopers, use	128
San Miguel County, geology	34
Santa Rita-Hanover region, estimating practice	78
exploration	62
ore deposits, geology	36
sampling practice	78
square-set stoping	121

	Page
New York, Edwards district, diamond drilling	56
estimating practice	74
exploration	54
long-hole drilling	56
ore deposits, geology	41
sampling practice	74
Edwards mine, discovery	55
exploration	56
main shaft	84
open stoping	105
figure	105
ore bodies, figure	43
stope preparation	95
St. Joseph Lead Co., Balmat mill, capacity	168
flow sheet	169
milling method	168
ore treated	168
products, grade	168
ratios of concentration	168
recoveries	168
water consumption	168
Balmat property, channel sampling	74
Hughesville concentrator, capacity	159
flow sheet	159
milling method	159
ore treated	159
products, grade	159
ratio of concentration	159
water consumption	159
No. 2 mine. *See* Tennessee, mascot.	
No. 8 mine. *See* Missouri.	
Nonmetallic minerals, gangue, occurrence, table	10
Oklahoma, Canam Metals Corporation, White Bird concentrator, capacity	147
flow sheet	149
milling method	147
ore treated	147
products, grade	147
ratio of concentration	147
recoveries	147
water consumption	147
See also Tri-State district.	
O'Meara, R. G., work	154, 187
Open stopes, figures	102, 103, 104, 105
Open-stope methods, advantages	107
disadvantages	107
summary	107
use	101
Open-stope mining, heading-and-bench variation	101
Ore, angle of repose	106
flotation treatment	181
lead and zinc, classifying	177
concentration	146
sizing	177
production per man-shift, table	144
sampling, summary	79
table	80
Ore body, brecciated, figure	22
types	22
semisheet, ground type, figure	22
Ore deposits, estimation	65
exploration	47
sampling	65
Ore reserve, advantages	99
estimating, summary	79
table	80
Ore shoots, definition	13
in dolomitized areas	14
localization	13
influence of geologic structures	14
Outcrops, character	11
Overhand stoping, discussion	106
Oxidized zone, character	12
Page concentrator. *See* Idaho, Federal Mining & Smelting Co.	

Page
Paleozoic ores, occurrence 16
Pallanch, R. A., work 161
Park City district. *See* Utah.
Park Utah mine. *See* Utah.
Pecos concentrator. *See* New Mexico, American Metal Co.
Pecos district. *See* New Mexico.
Pehrson, Elmer W., work 2
Penrose, R. A. F., Jr., work 15
Picher field. *See* Tri-State district.
Plant costs, per pound of lead and zinc, table 197
Poston, Roy H., work 49, 50, 68, 85
Power, distribution, table 188
Pre-Cambrian ores, occurrence 16
Price, G. S., work 156

Ransome, F. L., work 25
Reigart, John R., work 50
Richard, F. W., work 37, 62, 78, 121
Ries, H., work 8, 14, 45
Rod mills, feed 172
  operating data, table 175
Rod-mill products, screen analyses, tables 173
Roll products, screen analyses, table 172
Roughing and cleaning, concentration 179

St. Louis Smelting & Refining Co. *See* Missouri.
Sales, Reno H., work 33
Sampling, methods 66
  table 80
  summary 79
  table 80
Sampling practice, examples 68
San Miguel County. *See* New Mexico.
Sansom, F. W., work 147
Santa Rita-Hanover region. *See* New Mexico.
Shafts, cost 85
  table 85
  development, discussion 81
  inclined 82
  sinking, costs 85
    table 85
  size 83
  typical plan, figure 85
  vertical 82
Shale drilling, use 52
Shrinkage stoping, advantages 134
  disadvantages 135
  discussion 131
  figures 132, 133
  summary 134
  variations 132
Silver, in lead and zinc ores, occurrence 7
Silver Bow County. *See* Montana.
Silver King Coalition Mines. *See* Utah.
Sinking, summary 64
Sized material, concentration 179
Sizing, methods employed 177
Slimes, separation 177
Slushing, discussion 104
Smith, Arthur, work 23
Smith, Lewis A., work 2
Snow, Fred W., work 86
Southeastern Missouri district. *See* Missouri.
Spencer, A. C., work 46
Sphalerite, occurrence 17
Square-set stoping, discussion 107
  figures 109
    111, 114, 116, 118, 119, 120
  methods, advantages 122
  disadvantages 123
  summary 122
  mines employing 109

Page
Square-set timbering, figure 108
  method of using 108
Steel balls, consumption 174
Steel rods, consumption 174
Stellited bits, use 63
Stope, development 94
Stoping, cost 137
  tables 139, 140
  discussion 98
  rate 99
Stoping methods, classification 100
  discussion 99
  selection 99
Stoping operations, period 98
Stott, Charles E., work 36
Strachan, C. B., work 23, 154
Sulphide enrichment, economic significance 13

Table concentration, discussion 179
Tailings, disposal 187
Tennessee, American Zinc Co., concentrator, flow sheet 154
    mill capacity 154
    milling method 154
    ore treated 154
    product, grade 154
    recoveries 154
  Eastern district, exploration 54
    ore desposits, geology 23
  Mascot, heading-and-bench stoping 102
    figure 104
    No. 2 mine, shaft development 83
    open-stope mining, figure 104
    ore bodies, description 23
      figure 55
    ore body, figure 24
    stope preparation 95
  zince deposits, exploration 48
  zinc ore, analysis 24
Tertiary age, ore deposits 16
Test holes, drilling 49
Tillson, B. F., work 134
Tintic district. *See* Utah.
Tintic Standard mine. *See* Utah.
Top slicing, applicability 137
  discussion 135
Transverse shrinkage stope, mining, successive stages 134
  figure 133
Treadwell-Yukon Co. *See* Nevada.
Tri-State district, Barr mine plan 21
  Boone formation, ore deposits 19
  churn-drill exploration 50
  estimating practice 71
  exploration 50
  exploratory drilling, table 52
    cost, table 52
  heading-and-bench mining 101
    figure 103
  mine valuation 71
  operating costs, table 197
  ore, grade 23
  ore bodies, figure 72
    mineralization, figure 22
  ore deposits, discovery 47
    geology 18
  Picher field, exploration 51
    maps, use 53
  sampling practice 71
  shafts 83
  stope development 94
  test-hole drilling 53
  See also Kansas; Missouri; Oklahoma.
Tunnel development, discussion 81
Tybo concentrator. *See* Nevada, Treadwell Yukon Co.

Umpleby, J. B., work 26
Underhand stoping, discussion 106
U. S. Smelting, Refining & Mining Co. *See* Utah.

Page

Utah, Carboniferous Wasatch limestone, ore occurrence 27
Chief Consolidated mines, prospecting 60
Chief Consolidated Mining Co., concentrator, capacity 162
flow sheet 163
milling method 162
ore treated 162
products, grade 162
ratio of concentration 162
recoveries 162
water consumption 162
Park City district, deposits, ore minerals 29
exploration 58
ore, estimation 75
grade 29
ore deposits, types 28
geology 27
sampling 75
square-set stoping methods 112
Park City limestone, ore occurrence 27
Park-Utah mine, costs, table 197
cross-section 27
estimation practice 76
exploration 59
raises 96
sampling 76
square-set-and-fill stoping 114
figure 114
Silver King Coalition mines, bedded lead ore, figure 28
costs, table 197
exploration 58
sampling 75
shaft development 84
stoping method 112
stringer sets, figure 113
Thaynes formation, ore occurrence 27
Tintic district, estimation prac-
exploration 59
ore deposits, geology 29
sampling practice 76
square-set stoping 115
Tintic Standard mine, costs, table 197
diamond bracing, use, figure 116
exploration 59
figure 31
geology 29
ground conditions 117
lead-ore shipments, analysis 32
ore estimation 77
raises 96
sampling 76
shaft development 84
siliceous ore, analysis 32
square-set stoping practice 115
figure 116

Page

Utah, etc.—Continued.
U.S. Smelting, Refining & Mining Co., Midvale concentrator, capacity 161
flow sheet 161
milling method 162
ore treated 161
products, grade 162
ratio of concentration 162
water consumption 162

Vanderburg, Wm. O., work 34, 61, 77, 109, 125, 158

Waco section, drilling, costs 51
Wade, James W., work 29, 60, 76, 115
Water, reclaimed, use 187
Weaver, Lamar, work 86
Weber quartzite, ore occurrence 27
Weed, Walter H., work 33
Wells, F. G., acknowledgment 1
West mill. *See* Idaho, Bunker Hill & Sullivan Mining & Concentrating Co.
Wethered, C. E., work 26, 57, 129
White Bird concentrator. *See* Oklahoma, Canam Metals Corporation.
Wigton, G. H., work 162, 187
Wisconsin, zinc ores, open-stope mining 106
Wolff, J. E., work 45
Woolsey, L. H., work 27
Wright, C. W., work 3
Wright, Ira L., work 37, 165

Yellow Pine district. *See* Nevada.

Ziegler, W. L., work 150, 187
Zinc, deposits 36
milling, introduction 1
mining, introduction 1
mining districts 16
production 2
costs, table 197
value 2
Zinc deposits, changes with depth 12
geologic age 16
geology 6
mineralogy 6
minerals contained, table 8
Zinc minerals, occurrence, table 9
Zinc ore, concentrates produced, table 186
concentration, methods 146
deposits, types 6
principal sources, map 5
production 3
tables 4
Zinc ore bodies, outcrops 11

U. S. GOVERNMENT PRINTING OFFICE: 1935

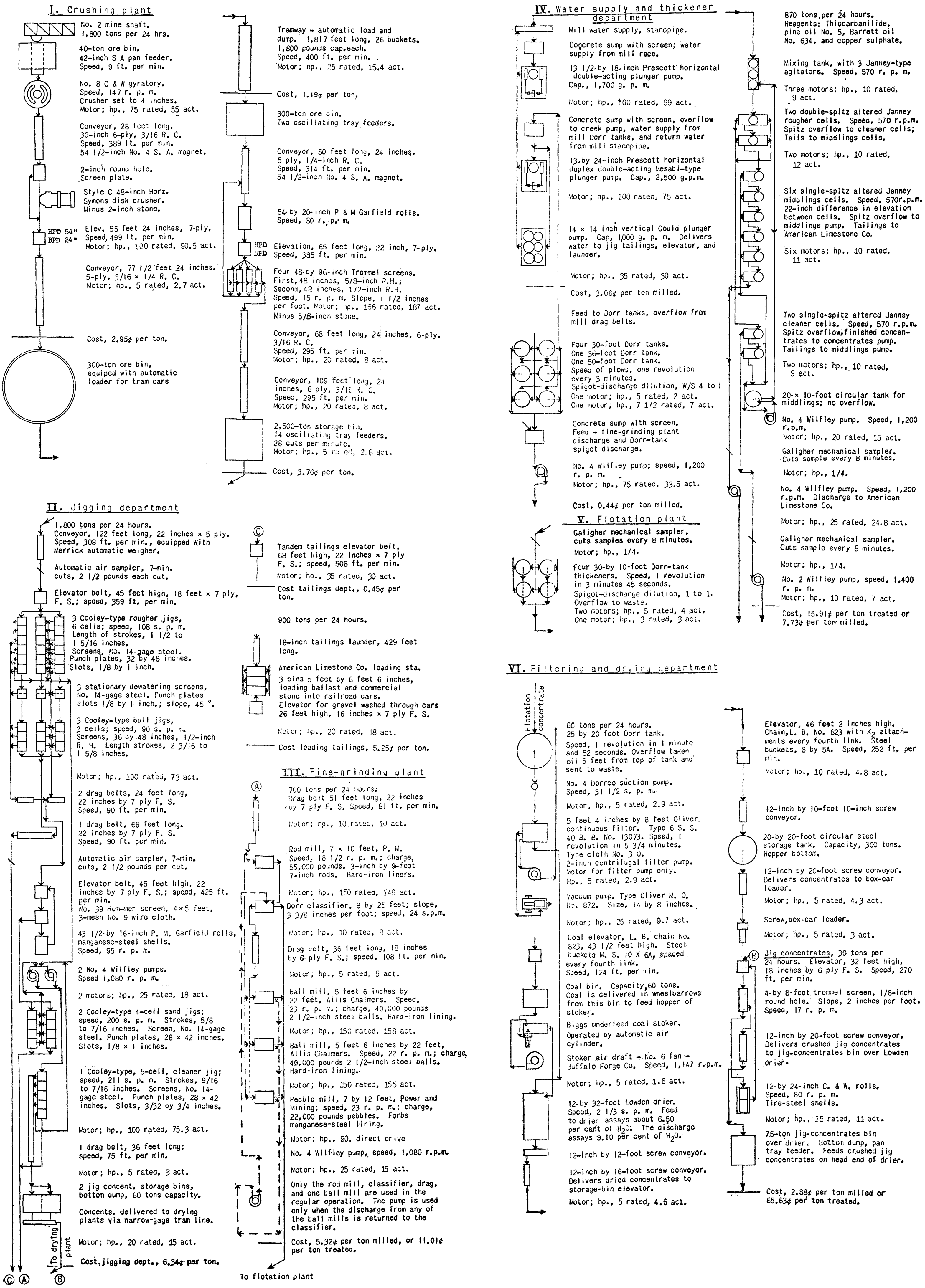

FIGURE 48.—Flow sheet of ore treatment, American Zinc Co. of Tennessee, Mascot, Tenn., concentrator E: I, Crushing plant; II, jigging department; III, fine-grinding plant; IV, water-supply and thickener department; V, flotation plant; VI, filtering and drying department.

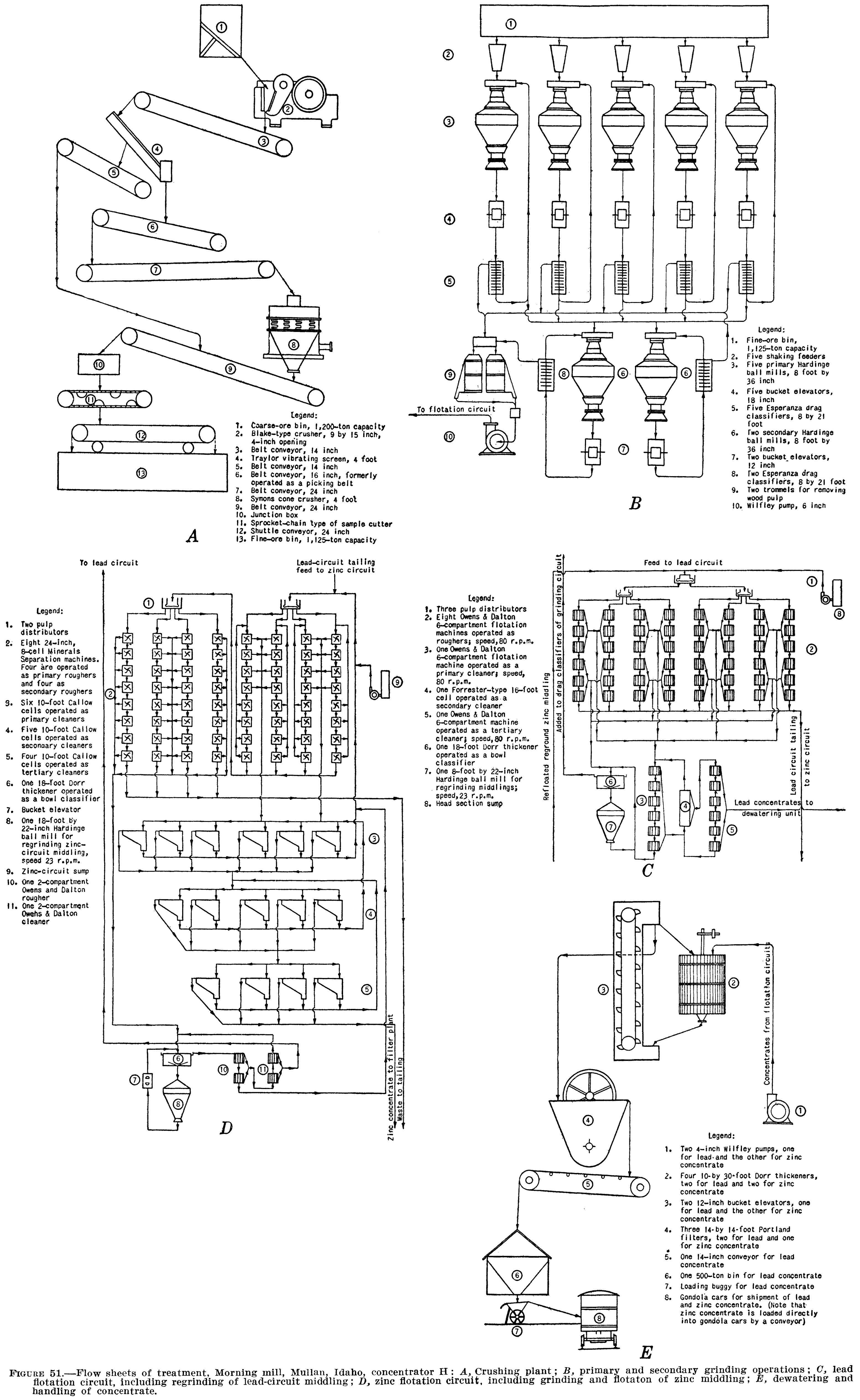

FIGURE 51.—Flow sheets of treatment, Morning mill, Mullan, Idaho, concentrator H: *A*, Crushing plant; *B*, primary and secondary grinding operations; *C*, lead flotation circuit, including regrinding of lead-circuit middling; *D*, zinc flotation circuit, including grinding and flotaton of zinc middling; *E*, dewatering and handling of concentrate.