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MINING AND MILLING OF LEAD AND ZINC ORES IN THE MISSOURI-KANSAS-OKLAHOMA ZINC DISTRICT

BY

CLARENCE A. WRIGHT

COOPERATION WITH THE MISSOURI BUREAU OF MINES AND GEOLOGY
H. A. BUEHLER, State Geologist

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MINING AND MILLING OF LEAD AND ZINC ORES IN THE MISSOURI-KANSAS-OKLAHOMA ZINC DISTRICT.

By Clarence A. Wright.

INTRODUCTION.

The Missouri-Kansas-Oklahoma lead and zinc district, better known as the Joplin district, includes the mines in southwest Missouri and in those parts of Kansas and Oklahoma that are directly adjacent. (See Pl. I.) This report of investigations carried on by the Bureau of Mines gives the methods used in mining and milling and indicates in some detail the conditions that affect the efficiency of those methods; it does not attempt to discuss the geology of the district, except incidentally, as this has been described in numerous reports by several geologists. Some of the more important recent publications are those of the United States Geological Survey by Smith and Siebenthal,[a] Bain,[b] and also those of the Missouri State geological survey by Buckley and Buehler.[c] Other reports are those by Schmidt and Leonard,[d] and Winslow,[e] of the Missouri geological survey, by Haworth and Crane,[f] of the Kansas geological survey, and by Jenney.[g]

Although the lead and zinc ores are closely associated, the proportion of zinc ore mined at present is far greater than in former years. For this reason and because the losses in milling zinc ores are proportionately greater than in milling those of lead, the zinc ores are here given most consideration. Only the mines and mills actually examined and tested are discussed in detail, statements of

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[a] Smith, W. S. T., and Siebenthal, C. E., Joplin district folio (No. 148), Geol. Atlas, U. S. Geol. Survey, 1907, 20 pp. Siebenthal, C. E., Origin of the zinc and lead deposits of the Joplin region, Missouri, Kansas, and Oklahoma: U. S. Geol. Survey Bull. 606, 1915, 283 pp. Smith, W. S. T., Lead and zinc deposits of the Joplin district, Mo.-Kan.: U. S. Geol. Survey Bull. 213, 1903, pp. 197–204.

[b] Bain, H. F., Preliminary report on the lead and zinc deposits of the Ozark region: Twenty-second Ann. Rept. U. S. Geol. Survey, pt. 2, 1901, pp. 111–215.

[c] Buckley, E. R., and Buehler, H. A., Geology of the Granby area: Mo. Bureau of Geology and Mines, vol. 4, 2d ser., 1906, 120 pp.

[d] Schmidt, A., and Leonard, A., Lead and zinc regions of southwestern Missouri: Mo. Geol. Survey, vol. 1, 1874, pp. 381–502.

[e] Winslow, A., Lead and zinc deposits: Missouri Geol. Survey, vol. 6, 7, 1894.

[f] Haworth, Erasmus, Crane, W. R., Rogers, A. F., and others. Special report on lead and zinc: Univ. Geol. Survey Kansas, vol. 7, 1904.

[g] Jenney, W. P., Lead and zinc deposits of the Mississippi Valley: Trans. Am. Inst. Min. Eng., vol. 22, 1894, pp. 171–225, 642–646.

results being restricted to the mines in general, with no special reference to any one mine.

Several mill tests were made to determine in a general way the efficiency of the milling practice in this district and to discover at what stage of the process the losses were greatest. Making these mill tests proved difficult, for, because of the arrangement of the mills and the necessity of estimating the tonnage treated, systematic mill testing and sampling can be done at only a few plants. Many tables giving results of mill tests and screen tests have been omitted because they would serve only to emphasize the important points shown by other data given.

The writer has tried to bring out the possibilities of making certain improvements that would effect a greater saving, and has endeavored to avoid the use of technical terms and phrases in order that all statements may be clear and easily understood.

ACKNOWLEDGMENTS.

The writer is greatly indebted to H. A. Buehler, State geologist of Missouri, who not only offered many suggestions, but generously contributed the parts on "Geology" and "Prospecting;" to G. B. Corless, formerly of the Missouri bureau of mines and geology, who prepared the map of the district and assisted in many other ways; and to R. R. Rinker, formerly chemist of the Missouri bureau of mines and geology, for many chemical analyses, only a part of which are presented in this report.

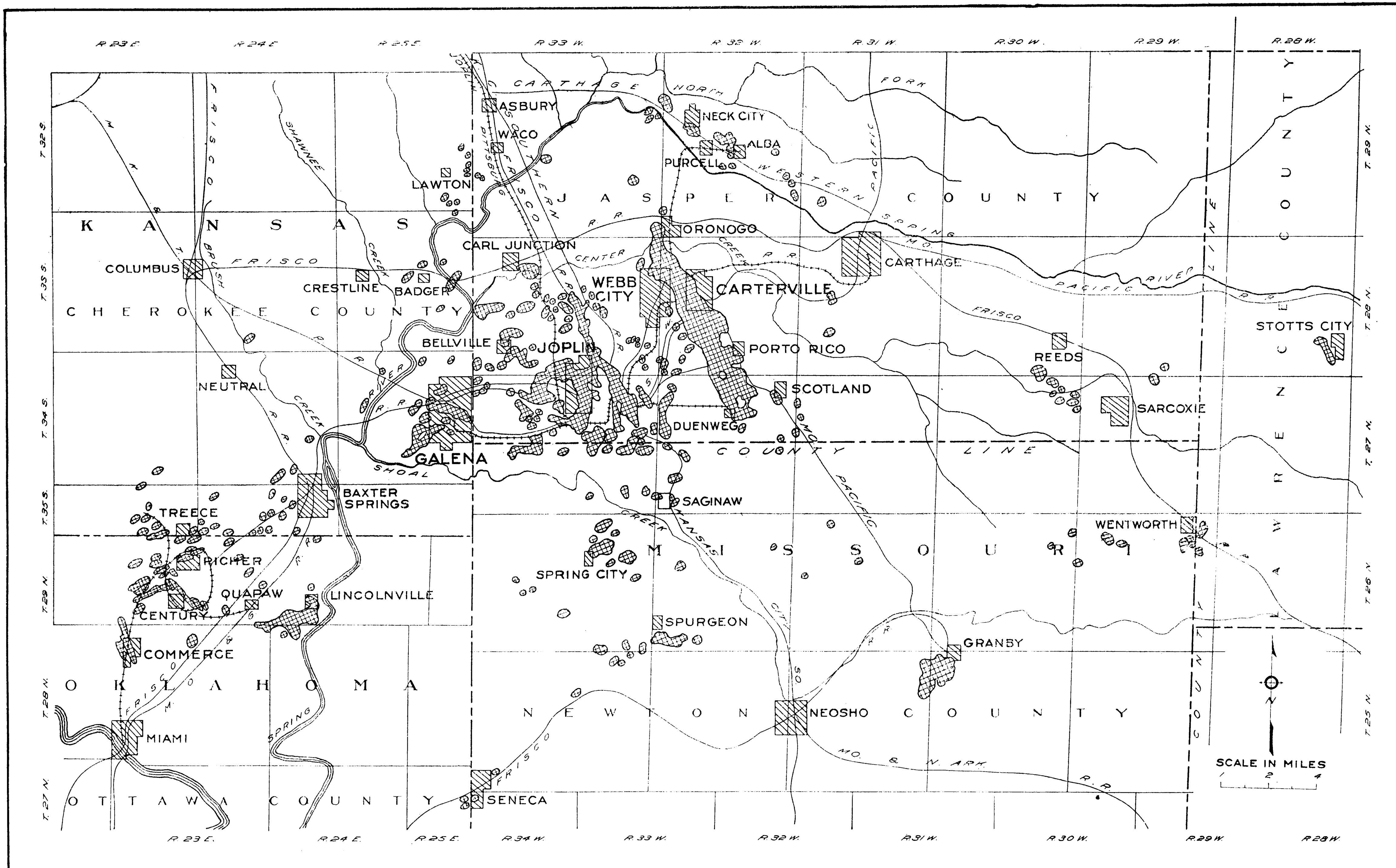
Acknowledgment is also due to operators and mine superintendents for furnishing information, for extending many courtesies and privileges, and for giving assistance whenever possible.

GEOLOGY OF ORE DEPOSITS.

In general the district has a gently rolling, prairie-like surface, although in the immediate vicinity of the larger streams the topography is more broken and frequently becomes decidedly rough.

The ore bodies of the district have definite relations to certain geologic formations and the following brief summary of the stratigraphy is intended simply as a key to the beds encountered in prospecting.

The formations underlying the Joplin district comprise about 1,800 feet of sedimentary beds, ranging in age from Cambrian to Pennsylvanian, resting upon igneous rocks of pre-Cambrian age. The generalized section shown in figure 1, is based upon the record of well No. 6 of the Carthage water works, Carthage, Mo.



MAP OF AREAS WHICH HAVE PRODUCED ZINC AND LEAD ORES IN THE JOPLIN, MO., DISTRICT.

System	Series		Name of Formation	Thickness, Feet	Character of Rocks
Carboniferous	Pennsylvanian		Cherokee formation	0-200+	Drab to black shale and gray to buff sandstone with occasional beds of coal
			–Unconformity–		
	Mississippian		Carterville formation	0-50	Light to dark shales and shaly and oolitic limestones with some massive soft to hard sandstones
			–Unconformity–		
		Boone formation, 250-375	Short Creek oolite	2-8	Massive homogenous bed of oolitic limestone
				80-120	Limestone, in large part crystalline, with interbedded chert
			Grand Falls chert	10-70	Heavy-bedded solid chert
				125-175	Limestone, with interbedded chert in upper part, shaly at the base
			"Blue shale"	1-13	Blue calcareous shale
Cambro-Ordovician			–Unconformity–		
			Jefferson City	500-600	Dolomite with interbedded chert and occasional thin beds of sandstone
			Roubidoux	10-25	Sandstone, angular, clear quartz grains
				700	Undifferentiated light to dark crystalline dolomites with interbedded chert and thin beds of sandstone
			Lamotte?	200	Sandstone, conglomeratic at the base
			–Unconformity–		
Pre-Cambrian					Granite

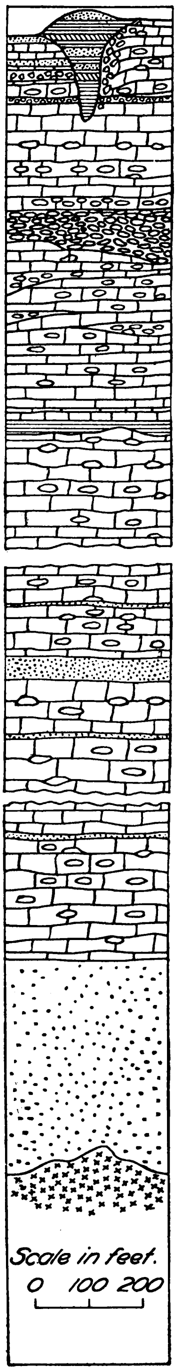


FIGURE 1.—Generalized section of formations in Joplin district.

DESCRIPTION OF BEDS.

The lowest stratified beds comprise 200 feet of sandstone which may be correlated with the Lamotte sandstone of the Ozark region. Overlying this are cherty and noncherty dolomites, approximately 700 feet thick, which have not been differentiated, although they correspond to the interval occupied by the Bonneterre, Davis, Derby, Doe Run, Eminence, Potosi, and Gasconade formations. The Roubidoux sandstone which overlies these beds, at 900 to 1,100 feet beneath the surface, is known to local drillers as the "sea level" or "water" sand and is the formation sought in the deep wells drilled for water supply. The Jefferson City dolomite lies above the Roubidoux and averages 540 feet in thickness.

Above the dolomite is a marked unconformity. In the southern part of the area a black shale (Chattanooga) of Devonian age overlies the dolomite, whereas in the northern part a blue shale of Mississippian age rests directly on it.

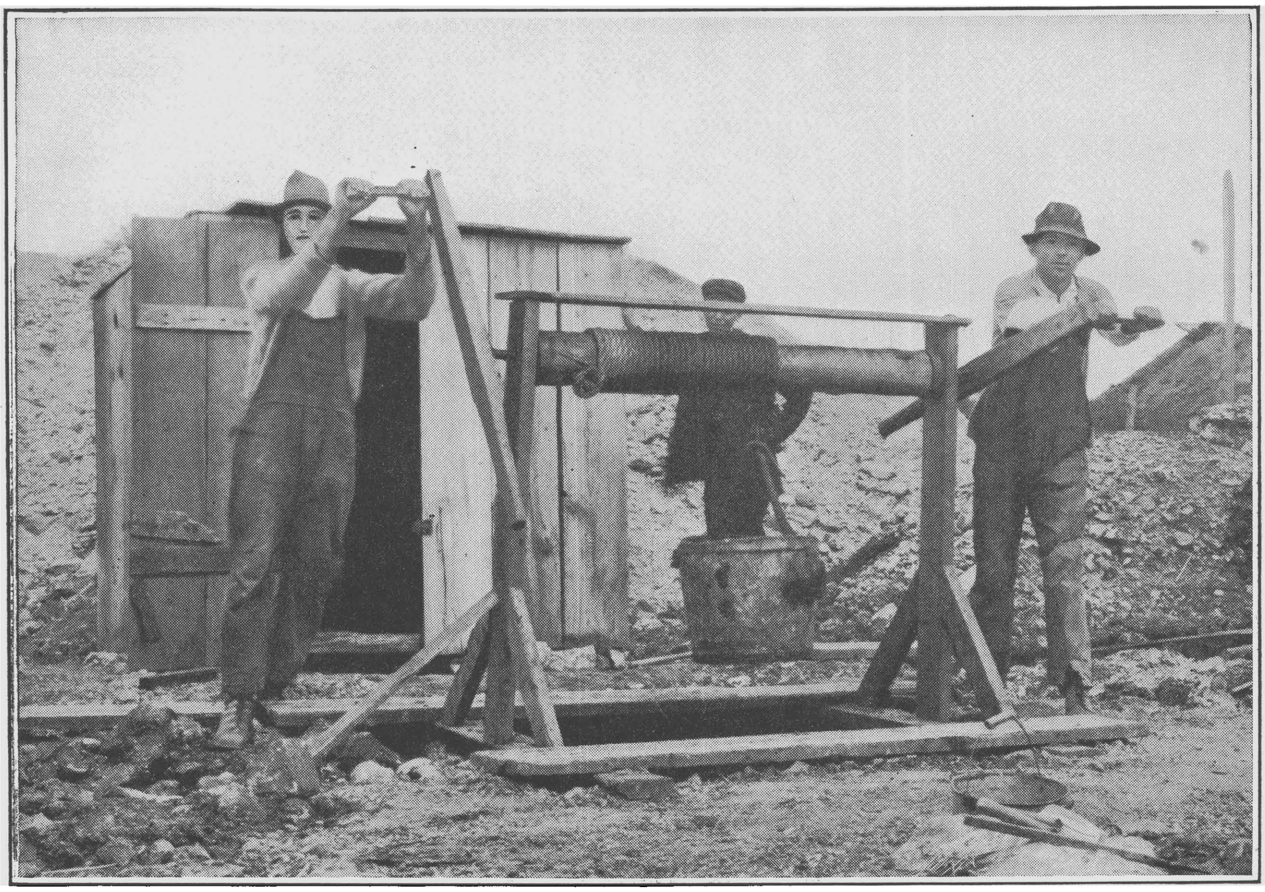
The Boone formation, of Mississippian age, consisting of cherty limestone and chert, overlies the shale and is the chief surface formation of the region; it is approximately 325 feet thick. Above the Boone is the Chester series, also of Mississippian age. On the western edge of the district where this series overlies the Boone almost continuously, it consists of limestone and sandstone; but to the east the Chester beds are chiefly in depressions or "sinks" formed during the erosion interval between the deposition of the Boone and the Chester.

The Cherokee shale of Pennsylvanian age is the youngest formation of the region; it is the surface rock along the western edge of the district, whereas to the east it forms outliers and occupies sinks or structural depressions similar to those occupied by the Chester, which it overlies.

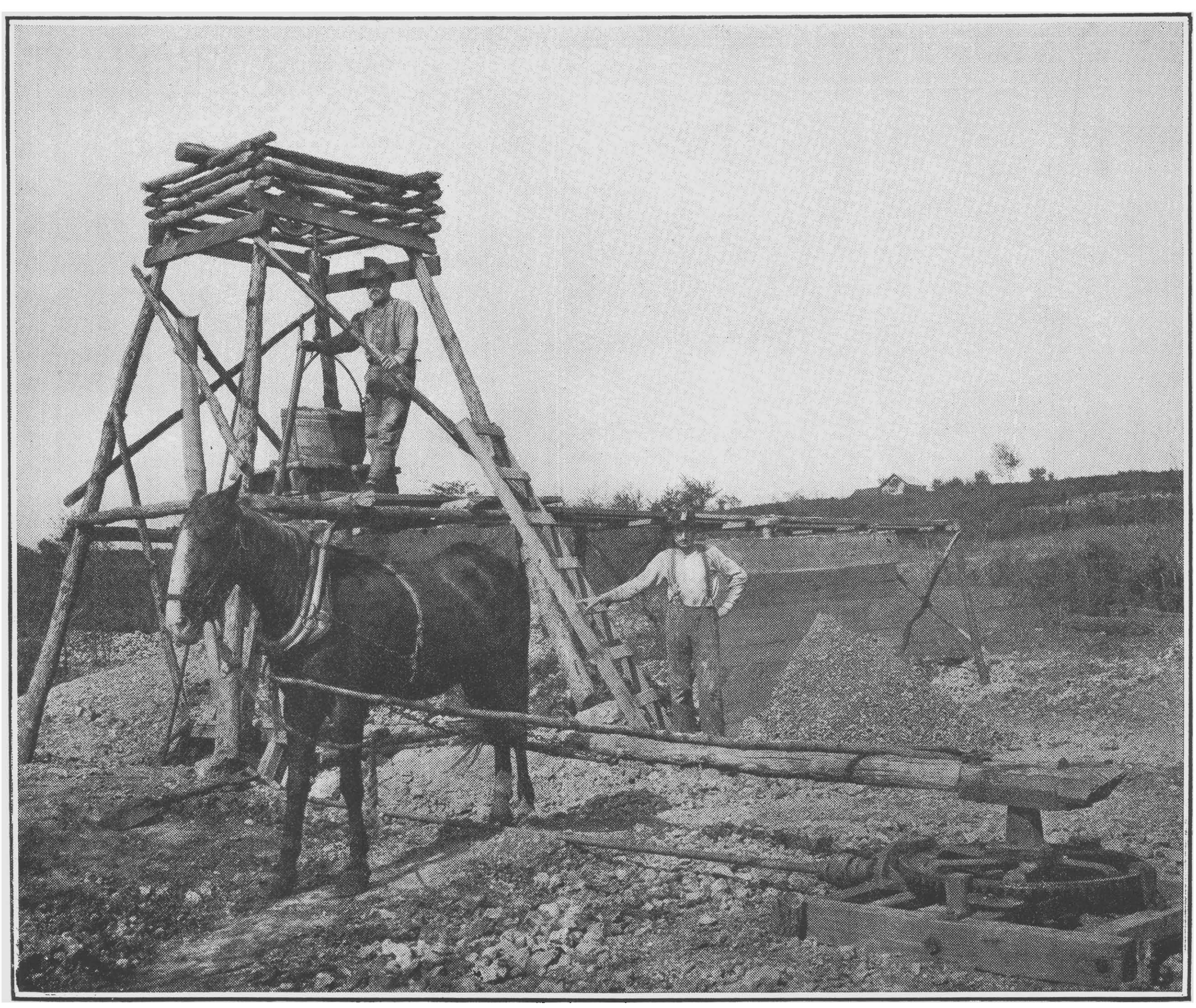
ORE BODIES.

The ore bodies are chiefly in the Boone formation and their character varies with their position. The upper 100 to 150 feet of the formation is cherty, crystalline limestone, underneath which is 2 to 8 feet of oolitic limestone known as the Short Creek oolite. Beneath the Short Creek is 100 feet of cherty, coarsely crystalline limestone similar to that above. Underlying this division is 8 to 30 feet of chert known as the Grand Falls chert. The lower part of the Boone formation consists of 50 to 100 or more feet of bluish limestone and chert which rests upon the Mississippian or Devonian shale.

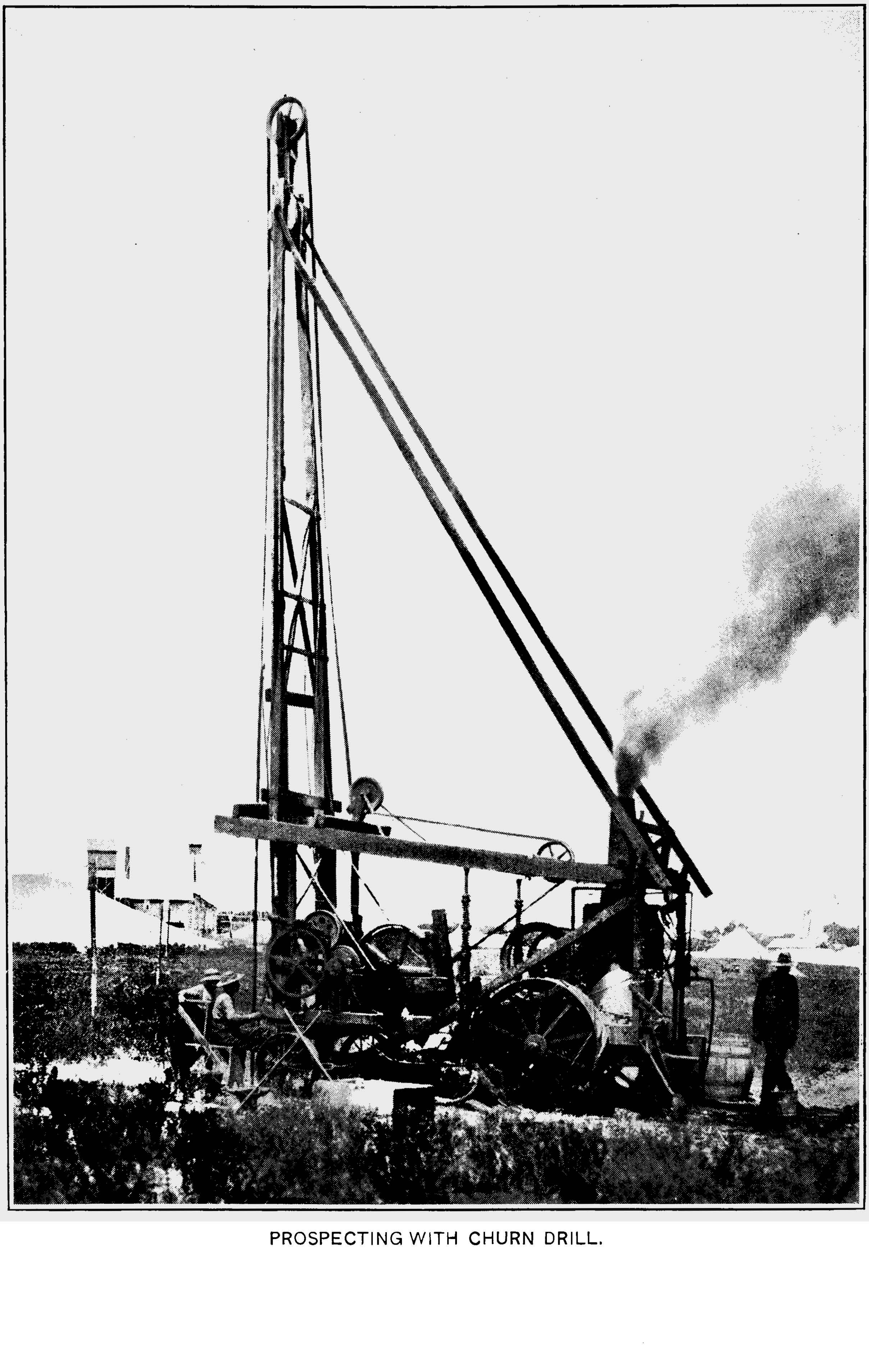
The ore bodies above the Grand Falls chert are usually irregular and lie mainly at the contact of the undecomposed Boone limestone and the shale or sandstone of the Cherokee, which occupies old depressions or solution channels in the limestone. The Grand Falls chert has been mineralized rather uniformly over large areas and



A. HAND WINDLASS USED AT PROSPECT SHAFT.



B. HORSE WHIM USED AT PROSPECT SHAFT.



PROSPECTING WITH CHURN DRILL.

hence is known as sheet ground. In parts of the area the limestone underneath the Grand Falls chert carries small mineralized fissures and openings and these deposits, where worked, have usually shown high faces of comparatively low-grade ore. In parts of the district the ore deposits extend into the underlying shale.

In the Miami, Okla., area the shallow deposits are in the Chester sandstones where these have been fractured so as to form a "bowldery" ground that in many respects is similar to that of some shallow deposits in the upper part of the Boone formation.

In general the Short Creek oolite and Grand Falls chert are the chief markers or key beds utilized in determining geologic horizons in prospecting. The Short Creek oolite is approximately 100 feet above the Grand Falls chert and in prospecting sheet ground may be used to indicate the depth of hole necessary. In searching for ore bodies above the Grand Falls it is usual to prospect the juncture of the Cherokee shale and Boone limestone as indicated by exposures of the two. In the Miami district the Cherokee formation can not be used as a guide, for it forms most of the surface.

PROSPECTING.

In the early development of the Joplin district, shallow deposits were sought by sinking shafts or test pits. (See Pl. II.) As the shallow ores were exhausted and mining extended below water level prospecting by shafts became too expensive and it has largely been superseded by drilling. (See Pl. III.)

Churn drills are used exclusively. The brecciated cherts and the many cavities prohibit the general use of the diamond drill. In the Oronogo Circle mine a shot drill was used underground to prospect the lower runs of ore. Near Springfield diamond drills have been used underground a little in prospecting the parallel "gumbo" runs in the shaly, noncherty formations of that area.

Much of the churn drilling is done by contract, the price ranging from \$0.90 to \$1.50 per foot. The maximum depth of hole is seldom more than 350 feet and the average is probably less than 200 feet. The usual diameter is 5⅝ inches. The holes are cased to solid rock. In soft ground the casing goes to the top of the ore body; if a lower run of ore is found a smaller casing is inserted to its top in order to prevent bits of ore from the upper run salting the drillings from the lower.

When ore is struck, samples of cuttings are taken every 2 feet. Usually the samples are collected by emptying the bailer into a bucket or half barrel and decanting the water after the drillings have settled. A part of the sample is placed in a pile on the ground near the drill, tags being used to show the depths of the beds from which the successive piles of cuttings came.

In some places the drillings are washed with water. This procedure, however, may materially affect the percentage of ore in the sample. In other places the drillings are carefully dried and sampled, although a small part of the various piles on the ground is often taken as a sample to represent the ground drilled through. Of course, such sampling is liable to spoil the value of the assays.

Commonly a driller designates the ore as "shines," "good shines," "fair jack," or "good jack." As the real significance of such designations depends upon the opinion of the driller, the future value of many records is virtually lost. Only by careful sampling and assaying of the cuttings can reliable records be had.

In order to verify a drill record a hole is sometimes shot. After the hole is thoroughly cleaned a small sheet-iron pipe containing dynamite is lowered to the ore and the charge exploded. The material loosened by the shot is then broken into particles with the drill and taken out with the sand bucket. This method has given some very reliable results.

Hand jigs are used to some extent in development work and at some small mines that are in the prospect stage. (Pl. IV.) This is especially true of shallow deposits and of prospectors with little working capital.

ORE DEPOSITS.

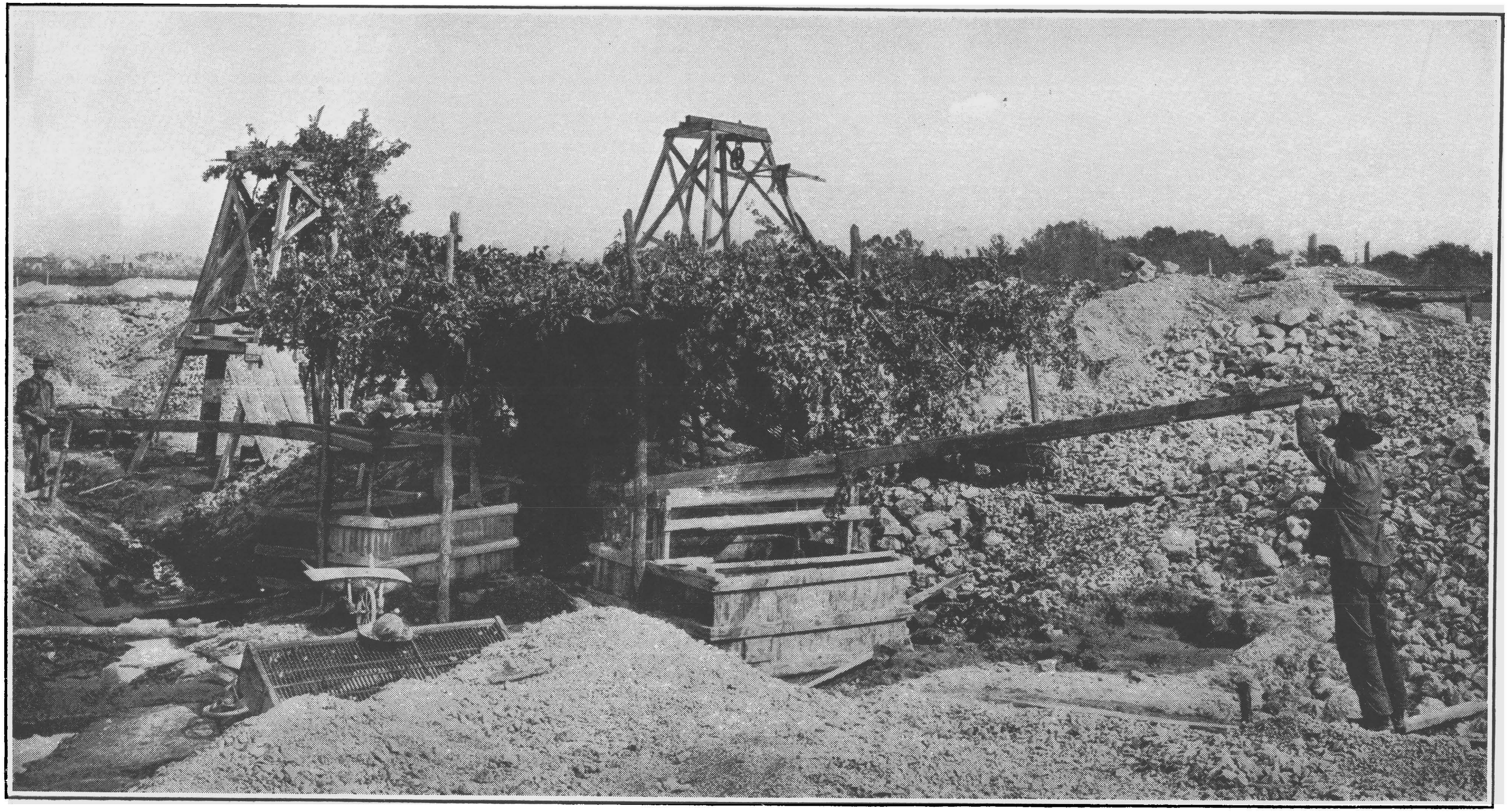
OCCURRENCE.

SHAPE OF ORE BODIES.

The lead and zinc ores are widely distributed throughout in rather well-defined blanket deposits, better known as "sheet-ground" deposits, or in irregular deposits known as "runs."

In the sheet-ground deposits the ore bodies lie nearly horizontal between the bedded layers of cherts. The seams of ore vary in thickness from a fraction of an inch to several inches, and here and there contain cavities lined with crystals of lead and zinc minerals, with minor amounts of iron sulphides and other associated minerals. The ore-bearing beds are 6 to 20 feet thick, and are more uniform in richness and less liable to terminate abruptly than the "runs." Brecciation is relatively unimportant in the sheet-ground deposits.

The "runs" and the other irregular bodies of ore may be divided into two classes—those in fairly hard ground, where little timbering is necessary, and those known as "soft-ground" deposits, which require timbering in nearly all the drifts. These ore bodies usually have an elongated form and many are of considerable length and thickness. A few deposits are circular or semicircular in shape and are known as "circle" deposits. The so-called runs and the irregular ore bodies are found at shallower depths than deposits of the sheet-ground type,



HAND JIGS. SUCH JIGS ARE STILL USED AT SOME SMALL MINES AND IN DEVELOPMENT WORK.

and are as a rule considerably richer in mineral content, often containing large pockets of nearly pure blende and galena. These minerals are commonly associated with a dark-colored secondary chert as the main filling material of fractures and cavities.

GANGUE MATERIALS.

Chert and dolomite are the most important gangue minerals. Limestone, shale, "soapstone," and mud are less important. In some of the deposits the chert and dolomite are intimately mixed, the zinc blende being embedded in a black secondary chert that cements the gangue and the blende. Chert, or flint, is the characteristic gangue material of the ore deposits and can be recognized in three different varieties: The white chert, the blue to gray chert, and the black or secondary chert. The blue chert is known as "live flint," and its occurrence usually indicates the presence of blende. The black variety, which has already been mentioned as filling fractures, cavities, and the interstices of the brecciated rock, is generally found in the brecciated areas closely associated with sphalerite, or in places with galena, and as a cementing material or matrix for the blue and white cherts with zinc blende. The so-called "cotton rock," a decomposed chert, dead white in color, often forms layers several inches thick near the surface or overlying the ore bodies. It is noticeable in the roof of some of the "sheet-ground" mines, where it usually has to be taken down as waste with the ore or has to be propped with posts at frequent intervals.

DISTRIBUTION OF OXIDIZED AND SULPHIDE ORES.

Generally the oxidized ores of zinc and lead lie above and near the level of groundwater, the sulphide of zinc becoming dominant below, the proportion of galena growing less, and that of iron sulphides greater as the depth increases. Zinc blende is seldom found in the weathered zone, for it becomes oxidized, generally to silicate or carbonate of zinc, by the action of surface waters. As the general course of these waters is downward, and as the blende is more soluble and more easily oxidized than galena, it is found at a greater depth, below the influence of the oxidizing agents. Hence, the finding of a large body of galena with blende scattered through it indicates the probability of a good deposit of blende being found below.

MINERAL CONSTITUENTS OF THE ORES.

Galena, cerussite, sphalerite, calamine, and smithsonite are the minerals of chief commercial importance. These minerals are locally better known as "lead," for galena; "blende" or "rosin jack," "ruby jack," "steel jack," or "blackjack" for sphalerite;

and "silicate" for calamine and smithsonite. Closely associated with these minerals are pyrite, marcasite, chalcopyrite, calcite, dolomite, and, in a few mines, small crystals of quartz. Local names are "mundic" for the iron sulphides, "tiff" for calcite, and "spar" for dolomite. Besides these, in a few of the mines there are small deposits of bituminous compounds, which are termed by the miners "coal tar" or "asphalt." These compounds, which are described in detail on page 12, and are discussed by Siebenthal,[a] occupy small openings or crevices and are usually viscous at the ordinary temperature in the mines.

A yellow to light-brown clay is associated with the ore, especially in the "soft-ground" deposits, and this clay, although not a definite mineral, in many places contains considerable zinc silicate. Several other less common and less important minerals are associated with the ores.

DETAILED DESCRIPTION OF CHIEF MINERALS.

As the specific gravity and composition of the chief minerals have a direct bearing on the methods of concentration, these minerals are described separately. The physical and chemical properties of the minerals are taken from Dana.[b]

SPHALERITE OR ZINC BLENDE.

Composition.—Pure zinc sulphide (ZnS)=33 per cent sulphur and 67 per cent zinc, but natural sphalerite often contains small quantities of iron, manganese, and cadmium, and rarely other impurities. The sphalerite of the Joplin district is unusually pure, containing little iron and cadmium.

Physical properties.—Specific gravity, 3.9 to 4.1; hardness, 3.5 to 4. Luster, resinous to adamantine. Color, commonly yellow to brown ("rosin jack"), and dark brown to black ("blackjack"); also red ("ruby jack"), and almost colorless when pure. Streak, white to yellowish-brown.

Sphalerite, or zinc blende, is the principal ore mineral of the Joplin district. It occurs as crystals and crystal aggregates lining cavities; and also massive in seams and in the chert breccia.

SMITHSONITE.

Composition.—Zinc carbonate ($ZnCO_3$)= carbon dioxide 35.2 per cent, zinc oxide 64.8 per cent. Content of metallic zinc, when pure, 52 per cent. The impurities, found in small amounts, are the carbonates of iron and manganese.

Physical properties.—Specific gravity, 4.30 to 4.45; hardness, 5. Luster, vitreous inclining to pearly. Color white, often grayish, greenish, brownish-white. Streak, white. Fracture uneven to imperfectly conchoidal. Brittle.

Smithsonite does not occur in large quantities, as compared with sphalerite, but is an important zinc ore in the district. It is called

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[a] See Siebenthal, C. E., Origin of the zinc and lead deposits of the Joplin region—Missouri, Kansas, and Oklahoma: U. S. Geol. Survey Bull. 606, 1915; pp. 205, 206.

[b] Dana, E. S., Text book of mineralogy, 1908.

"silicate," as is calamine, and it is sold as such. The crystals and crystal aggregates are often found coating other minerals, especially sphalerite, or lining cavities. Both smithsonite and calamine are generally limited to the zone of weathering. It has been stated that these minerals are not usually found in paying quantity below a depth of 50 feet, and on account of their solubility in surface waters are seldom found at depths less than 15 or 20 feet.[a]

CALAMINE OR ZINC SILICATE.

Composition.—Zinc silicate (H_2ZnSiO_5)=Silica, 25.0 per cent; zinc oxide, 67.5 per cent; water, 7.5. When pure, calamine contains 54.2 per cent metallic zinc.

Physical properties.—Specific gravity, 3.4 to 3.5. Hardness, 4.5 to 5. Luster, vitreous. Color, white; sometimes with a bluish or greenish shade; also, yellowish to brown. Transparent to translucent. Streak, white. Fracture uneven to subconchoidal. Brittle.

Calamine, commonly known as "silicate" in the district, is found in the zone of oxidation, usually associated with smithsonite. "Tallow clays" often contain more or less basic zinc silicate that has not crystallized. These clays are fine grained and rather soft, having a conchoidal fracture. They are yellowish brown to gray in color, and in the mines are soft and plastic.

GOSLARITE.

Composition.—Hydrous sulphate of zinc ($ZnSO_4+7H_2O$).

Physical properties.[b]—Specific gravity, 1.95 to 2.04. Hardness, 2 to 2.5. Luster, vitreous. Color, white, yellowish.

Goslarite is formed by the decomposition of sphalerite, but is a rather rare mineral on account of its solubility in water. It has been found in old drifts, usually where the sulphate of iron has been found, and is known as ferrogoslarite.

GALENA.

Composition.—Lead sulphide (PbS)=Sulphur, 13.4 per cent; lead, 86.6 per cent.

Physical properties.—Specific gravity, 7.4 to 7.6. Hardness, 2.5 to 2.75. Luster, metallic. Color and streak, pure lead-gray. Opaque. Fracture, flat subconchoidal or even. Very brittle.

Galena, locally known as "lead," is the most important lead ore found in the district, but is of less commercial importance than sphalerite. It occurs in crystals and crystal aggregates in cavities; also in massive form, in seams of chert, and in chert breccia.

The galena of this district is unusually pure and is nonargentiferous, containing practically no silver. The lead concentrates produced average nearly 80 per cent lead. Galena is found in varying proportions associated with sphalerite, and when in large quan-

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[a] Siebenthal, C. E., and Smith, W. S. T., Joplin district folio (No. 148): Geol. atlas, U. S. Geol. Survey, 1907, p. 12.

[b] Brush, G. J., and Penfield, S. L., Manual of determinative mineralogy, 1906, p. 291.

tities it lowers the zinc recovery and the grade of the zinc concentrates produced from the mills. The galena, owing to its high specific gravity and mode of occurrence, is easily separated from other minerals by wet-milling methods.

CERRUSITE.

Composition.—Lead carbonate ($PbCO_3$)=Carbon dioxide, 16.5 per cent; lead oxide, 83.5 per cent; or lead, 77.5 per cent.

Physical properties.—Specific gravity, 6.46 to 6.57. Hardness, 3 to 3.5. Luster, adamantine, inclining to vitreous or pearly. Color, white, gray, grayish black. Transparent to subtranslucent. Streak, uncolored. Fracture, conchoidal. Very brittle.

Only small amounts of cerrusite are found at present in the mines of the district, and as a rule only near the surface. It occurs as crystal aggregates, but also in granular, massive, and compact forms.

PYRITE AND MARCASITE.

Pyrite.—Composition: Iron sulphide (FeS_2)=sulphur 53.4 per cent; iron=46.6 per cent. Physical properties: Specific gravity, 4.95 to 5.1; hardness, 6 to 6.5; luster, metallic; color, pale brass yellow; streak, greenish black to brownish black; opaque.

Marcasite.—Composition, same as pyrite. Physical properties: Specific gravity, 4.85 to 4.90; hardness, 6 to 6.5; luster, metallic; color, pale bronze-yellow, deepening on exposure; streak, grayish or brownish black; opaque; fracture uneven; brittle.

These two minerals, locally known as "mundic," are common throughout the ores of the district in small quantities. When present in more or less large amounts in the zinc ores, they are detrimental to the grade of concentrates produced. The difficulty in separating these sulphides of iron from blende is due to the fact that their specific gravities differ little from that of the sulphide of zinc.

The pyrite and the marcasite occur in crystals, crystal aggregates, and massive, and are found in rather large quantities closely associated with the lead and zinc sulphide ores, in the mines near Thom's Station and in the Miami, Okla., district. They are also found in small amounts in the country rock at much lower depths.

CHALCOPYRITE.

Composition.—Sulphide of copper and iron (Cu,FeS_2)=Sulphur, 35.0 per cent; copper, 34.5 per cent; iron, 30.5 per cent.

Physical properties.—Specific gravity, 4.1 to 4.3. Hardness, 3.5 to 4. Luster, metallic. Color, brass yellow; often tarnished or iridescent. Streak, greenish black. Opaque. Fracture, uneven. Brittle.

Chalcopyrite is present only in small amounts with the ores of the district and has not been found in commercial quantities. It occurs in small tetragonal sphenoids, and when these small crystals are on sphalerite, one finds parallel orientation of the crystal faces.

CALCITE.

Composition.—Calcium carbonate ($CaCO_3$)=Carbon dioxide, 44.0 per cent; lime' 56.0 per cent. Small quantities of magnesium, iron, manganese, zinc, and lead may replace the calcium.

Physical properties.—Specific gravity, 2.714 in pure crystals, varying somewhat widely in impure forms. Hardness, 3. Cleavage, highly perfect. Luster, vitreous to subvitreous, to earthy. Color, white or colorless; pale shades of gray and yellow; also brown and black when impure. Transparent to opaque. Streak, white or grayish. Fracture, conchoidal, obtained with difficulty.

Calcite, locally known as "tiff," is one of the most common minerals associated with the ores. It is the essential constituent of limestone, and occurs in crystals or crystal aggregates, and in granular form as a cement in chert breccia. In crystal form it is most frequently found lining cavities.

GYPSUM.

Composition.—Hydrous calcium sulphate ($CaSO_4+2\,H_2O$)=Sulphur trioxide, 46.6 per cent; lime, 32.5 per cent; water, 20.9 per cent.

Physical properties.—Specific gravity, 2.314 to 2.328 when in pure crystals. Hardness, 1.5 to 2. Luster, pearly to subvitreous; massive varieties often glistening sometimes dull earthy. Color usually white; sometimes gray, or brownish yellow impure varieties often black, brown, or reddish brown. Transparent to opaque Streak white.

The gypsum is a product of the weathering of calcium carbonate and is seldom found in the ores of the district, except in small quantities.

DOLOMITE.

Composition.—Carbonate of calcium and magnesium [(Ca, Mg) CO_3]=Carbon dioxide, 47.9 per cent; lime, 30.4 per cent; magnesia, 21.7 per cent; or calcium carbonate, 54.35 per cent; magnesium carbonate,, 45.65 per cent.

Physical properties.—Specific gravity, 2.8 to 2.9. Hardness, 3.4 to 4. Luster, vitreous, inclining to pearly in some varieties. Color, white, reddish, or greenish white, light gray or pale buff. Transparent to translucent. Fracture, subconchoidal. Brittle.

Dolomite occurs in granular massive form, and also in crystal aggregates lining cavities. As a rule, it is closely associated with or adjacent to the ore deposits. It is commonly known as "spar" or "pink spar" depending upon the shade of color. When this mineral is associated in considerable quantity with the ore, it is often difficult to make a high-grade zinc concentrate, as the specific gravity of dolomite is somewhat higher than that of flint, and small amounts will be found with the zinc concentrates.

BARITE.

Composition.—Barium sulphate ($BaSO_4$)=Sulphur trioxide, 34.3 per cent; baryta, 65.7 per cent.

Physical properties.—Specific gravity, 4.3 to 4.6. Hardness, 2.5 to 3.5. Luster, vitreous, inclining to resinous. Color, white; also inclining to yellow, gray, or brown. Transparent to translucent, to opaque. Streak, white. Fracture uneven. Brittle.

Barium sulphate is not common in this district, and is found in small scattered quantities. It is more abundant, however, in the Central Missouri district, where it is found with sphalerite and is practically impossible to separate from the zinc ore by the usual wet methods, although it is believed that flotation could be used to good advantage. When it occurs with the ore in the Missouri-Kansas-Oklahoma region it is concentrated with the zinc and is found in the zinc concentrates.

QUARTZ (CHERT).

Composition.—Silica (SiO_2)=Oxygen, 53.3 per cent, silicon, 46.7 per cent.

Physical properties.—Specific gravity, 2.65 to 2.66. Hardness, 7. Luster, vitreous, splendent to nearly dull. Colorless when pure; often various shades of yellow, brown, green, gray, blue, black. Streak, white. Transparent to opaque. Fracture conchoidal to subconchoidal in crystallized forms, uneven to splintery in some massive kinds. Brittle to tough.

In this district quartz occurs massive in the cryptocrystalline form known as chert and in granular aggregates in jasperoid, also known as secondary chert. It also is found as minute crystals coating siliceous surfaces, either on chert or jasperoid. As chert it is one of the most common minerals of the district, and locally is best known as flint.

JASPEROID.

Jasperoid is a dark siliceous rock resembling chert and is often called secondary chert. When found it always accompanies the ores in the sheet-ground mines and is the matrix of the chert breccia in mines other than those in sheet-ground.

HYDROCARBON COMPOUNDS.

The hydrocarbon compounds sometimes found with the ores of this district are generally not definite minerals, but are mixtures, which by the action of solvents or by fractional distillation can be separated into two or more components. These hydrocarbons are mostly bituminous coal, and bitumen, and are found in small quantities in many of the "shale patches." When the bitumen, locally known as "tar," "coal tar," or "asphalt" by the miners, is viscous, it often interferes with the milling of the ore by sticking to the rolls. Bituminous material has been especially noted in the mines near Miami, Okla., although it has also been found in the mines near Chitwood, Mo., and in other parts of the Missouri-Kansas-Oklahoma region.

VALUATION OF MINE AND ORE DEPOSITS.

VALUATION OF MINES IN RELATION TO PROSPECTING.

From the uncertain extent and character of the ore bodies in the Joplin district and the lack of prospecting or development work, other than churn drilling, before mining starts, it would seem that the opening of a mine represents a hazardous investment. Fortunately, the sum of money needed to open a mine in this field is much less than in some mining districts where larger enterprises are undertaken, so that the loss from the failure of any one property is not great.

Many failures in mining enterprises in the Joplin field are due to overvaluation based on insufficient prospecting and development and to the payment of excessive royalties. There are other causes, but these will not be discussed here.

The value of a prospective mine in this district is usually based on the zinc and lead content of the churn-drill cuttings. Unless the drill cuttings are fairly rich in zinc, they are described by the drillers simply as "shines" or "fair zinc." When the ore seems to be in fairly payable quantities, the drill cuttings are assayed for zinc, and also for lead if the latter is present. Even with the assay results, however, it is hardly possible to ascertain the value of an ore body.

Several times shafts have been sunk and mills erected on the basis of only three or four drill holes that gave cuttings fairly rich in metal content. A few of these mines proved to be of value, whereas others were worthless.

Such instances cause some investors in the Joplin district, as well as in other districts, to regard mining as a purely speculative and hazardous enterprise, especially those who do not realize that mining enterprises should be based on business principles, and are not familiar with mining conditions.

As the value of a mine is determined by its ore reserves and the quantity of available ore blocked out, sufficient prospecting and development work previous to beginning mining and milling operations is not only important but essential. The amount of such work necessary will depend on the richness of the ore, the size and shape of the ore body, and its depth below the surface, the location of the property with regard to other mines in the vicinity, and the area controlled by the company. Another important detail that bears a close relation to development work is the size of the concentrating plant to be built for treating the ore.

The design and capacity of the concentrating plant will depend on the character and the richness of the ore, the size of the ore body, and the tonnage to be handled. In other words, the capacity of the mill is more or less dependent on the life of the mine, and on whether the profits would be greater by treating the ore with a larger plant in a short time, or with a smaller plant during a longer period of time. The building of small and inexpensive plants in the Joplin district is largely a result of the ore bodies being generally small, irregular, and uncertain.

As the loss of time from lack of a continuous supply of ore is more costly with a large plant than with a small one, large central mills have not been considered feasible in this district. On the other hand, the writer believes that at many properties considerably more prospecting and development work would have shown the advisability of erecting a larger mill or one better equipped for saving more of the lead and zinc, especially in the treatment of the fine material.

In determining the value of a property many difficulties that the investor or shareholder does not always realize, or knows nothing about, are encountered. A few of these difficulties are the irregularity of the ore bodies, uncertainty in the estimates of the quantity of ore available, overvaluation of the ore body through insufficient prospecting and development work, and the changes in the market prices for the metals. In order to obviate these difficulties as far as possible, the investor or operating company must determine, as conditions permit, the following points: The extent of the ore deposit; the amount of prospecting and development necessary to delimit the ore; the quantity of ore available; the size and equipment of the mill, with reference to the character of the ore; the conditions to be met in actual practice on a commercial scale; and, at all times, safety in the capital invested.

PRESENT SYSTEM OF ROYALTIES AND EFFECT ON MINING AND MILLING PRACTICE.

Frequently a mine operator has two or more adjacent holdings leased from different landowners and has proved by development that the ore body is large enough to last several years. He is, however, not allowed to treat the ore mined on one lease in a mill on another lease, but is obliged to erect a mill on each property. It would therefore seem advisable to buy the land outright at a reasonable price, but as such purchase is not always possible, the operator is compelled to mine and mill the ore from each lease separately.

As the land owner seldom mines his own property but leases it for royalties of 10 to 15 per cent of the gross mineral output, and the property is again subleased one or more times, thus increasing the royalty to 20 or even 30 per cent, the operating company is compelled to use those milling methods by which the largest capacity possible from a given plant can be obtained. Consequently, in order to make any profit, milling efficiency is sacrificed and the mills are often greatly overcrowded.

Some of the operators and miners try to mine only the higher grade ore, leaving much of the lower grade ore that by itself would be unprofitable to mine. In other words, when the royalties are excessive, it is often the practice to gouge the ore. In consequence much lean ore is left underground where it represents a loss or total waste of mineral resources.

Some landowners and lessors do not seem to realize that the high royalties obtained through the subleasing practice are detrimental to their interests, as their financial returns are less in the end, and many of the gouged workings are left in such condition as to hinder seriously further mining and prospecting should the operating company be obliged to suspend work. Another result is that drifts are often left in such condition that the lives of miners are endangered. High royalties also tend to reduce wages, because lessees are compelled to work the mine as cheaply as possible, and consequently the labor is apt to be less efficient.

On the other hand, the present system of leasing has an advantage in that the land is constantly being prospected for rich ore bodies, and when a rich showing is struck on a lease, other persons seek to obtain a lease on adjacent land in order to find the continuation of the ore body.

In view of the above conditions, a system of royalties based on the net profits, or of reduced royalties under the present system, especially in subleasing, or a combination of the two, would seem feasible, and at many mines would result in more efficient mining and milling, thus improving the general conditions and placing the district on a firmer and more reliable basis.

PRODUCTION.

The Joplin district is one of the main ore-producing areas of the Ozark region, a dissected plateau occupying most of the southern half of Missouri and parts of Kansas, Oklahoma, and Arkansas; in this region and around its border are many important deposits of lead and zinc ore. Not only is the Joplin district the principal zinc-producing area of this region but its ores yield more spelter than is produced by any other district in the United States. A comparison of the total mine production of zinc in the United States for the years 1906 to 1916, by States, is given in Table 1 following.

TABLE 1.—*Mine production of zinc in the United States, 1907–1916, in short tons.*[a]

State.	1907	1908	1909	1910	1911	1912	1913	1914	1915	1916
Arizona	114	339	2,989	2,742	2,281	4,379	4,714	4,896	9,110	9,839
Arkansas	474	605	510	994	664	748	478	608	3,209	6,815
California	116	....	....	....	1,404	2,173	529	195	6,547	7,628
Colorado	26,394	15,065	25,605	38,545	47,304	66,111	59,673	48,387	52,297	67,143
Idaho	3,493	19	676	2,802	4,170	6,953	11,587	21,006	35,077	43,253
Illinois	737	1,717	2,163	3,549	4,219	4,065	2,236	4,811	5,534	3,404
Iowa	145	395	35	96	....	....	....	....	....	21
Kansas	17,772	14,119	11,445	13,229	10,272	10,633	10,088	11,284	14,365	12,448
Kentucky	....	....	56	6	158	491	327	230	764	1,096
Missouri	116,752	107,404	130,162	128,589	122,515	136,551	124,963	105,994	136,300	155,960
Montana	122	820	4,680	15,819	21,905	13,459	44,337	55,790	93,573	114,630
Nevada	1,084	558	1,507	1,354	1,774	6,661	7,210	6,490	12,188	16,222
New Hampshire	....	....	....	....	....	....	15	6	24	36
New Jersey	69,369	67,165	82,510	68,678	77,445	69,755	72,156	74,253	116,618	110,698
New Mexico	375	1,788	6,543	9,044	5,119	6,783	8,262	9,202	12,702	18,285
New York	....	....	....	....	81	....	....	....	2,455	4,507
North Carolina	....	....	....	....	....	142	10	....	....	....
Oklahoma	1,657	4,529	7,806	6,394	5,150	5,769	11,664	13,992	14,314	28,754
Tennessee	109	344	596	966	1,117	2,191	5,583	10,425	16,461	26,428
Texas	22	18	....	....	....	119	326	108	15	116
Utah	2,726	730	4,930	8,184	8,920	8,534	9,429	7,995	12,146	14,786
Virginia	....	705	58	794	1,032	249	2,719	174	1,267	2,276
Washington	....	....	....	....	10	....	....	....	122	847
Wisconsin	18,490	18,206	23,152	25,927	29,720	33,050	30,110	31,113	41,403	56,803
Total	259,951	234,526	305,423	327,712	345,260	378,816	406,416	406,959	586,491	701,995

[a] Figures for 1907–1915 from Mineral Resources U. S. for 1915: U. S. Geol. Survey, 1916, pt. 2, p. 859. Figures for 1916 supplied by U. S. Geol. Survey.

The rank of the Joplin district as a zinc producer is shown by Table 2. This table shows the production of spelter from the zinc ores in Missouri and also the production for the Joplin district, which includes practically the total zinc production from Missouri, Kansas, and Oklahoma. The percentage of total production in the United States represented by Missouri alone and by the Joplin district is also given for the different years.

TABLE 2.—*Primary spelter production from zinc ores of the Missouri-Kansas-Oklahoma districts.*

Source.	1906	1907	1908	1909	1910	1911	1912	1913	1914	1915[a]	1916[a]
Kansas[b]...short tons	3,902	13,850	8,628	9,185	10,220	6,843	5,668	9,956	10,634	14,365	12,448
Missouri[b].......do	130,348	141,824	123,655	140,676	140,652	127,540	149,557	129,018	114,019	136,300	155,960
Oklahoma[b].....do	....	719	2,235	3,008	2,297	2,247	2,041	6,397	9,449	14,314	28,754
Total	134,250	156,393	134,518	152,869	153,169	136,630	157,266	145,371	134,102	164,979	197,162
Missouri, percentage of United States production	65.3	63.4	64.8	61.0	55.7	47.0	46.2	38.2	33.2	23.2	22.2
Joplin district, percentage of United States production	67.2	70.0	70.5	66.4	60.7	50.3	48.5	43.1	39.0	28.1	28.1

[a] Figures are for mine production of zinc.
[b] Figures for 1908–1915 from annual volumes of Mineral Resources of United States, U. S. Geol. Survey; figures for 1916 supplied by U. S. Geol. Survey.

The production of spelter from the Joplin district ores has varied more or less during the past several years. The production from the Miami, (Okla.) camp has been on the increase since 1912, and it is believed that before long it will rival both the Joplin and Webb City camps in production.

The tenor of crude lead and zinc ore and concentrates produced in the Missouri-Kansas-Oklahoma zinc district for the years 1908 to 1916 is given in Tables 3 to 6.

TABLE 3.—*Tenor of crude lead and zinc ore and concentrates produced in southwestern Missouri, 1908–1916.*[a]

	1908	1909	1910	1911	1912	1913	1914	1915	1916
SOFT GROUND.[b]									
Total crude ore....short tons..	2,584,990	3,096,789	3,275,705	3,217,166	3,856,045	3,745,400	3,179,830	4,004,900	4,711,700
Total concentrates in crude ore....per cent..	5.2	4.6	4.4	4.5	4.21	4.56	4.61	3.88	3.38
Lead....do....	.5	.4	.4	.4	.34	.44	.48	.32	.28
Zinc....do....	4.7	4.2	4.0	4.1	3.87	4.12	4.13	3.56	3.10
Metal content of crude ore....do....	3.0	2.6	2.5	2.6	2.41	2.58	2.64	2.26	1.92
Lead....do....	.4	.3	.3	.3	.27	.34	.38	.25	.22
Zinc....do....	2.6	2.3	2.2	2.3	2.14	2.24	2.26	2.01	1.70
Average lead content of galena concentrates....do....	79.3	79.7	79.2	79.9	79.5	78.6	78.6	79.2	78.0
Average lead content of lead carbonate concentrates....do....	41.8	43.6	47.6	54.7	54.8	62.2	51.3	58.1	55.5
Average zinc content of sphalerite concentrates....do....	58.1	59.3	59.1	58.9	58.3	57.5	57.9	57.7	58.1
Average zinc content of silicates and carbonates....do....	42.0	38.5	35.9	40.4	38.8	38.8	38.6	39.3	39.4
Average value per ton:									
Galena concentrates....dollars..	52.50	52.37	51.06	54.41	54.43	51.51	46.16	48.19	82.09
Lead carbonate concentrates....do....	27.44	28.89	32.41	35.72	35.52	32.66	24.19	32.56	61.48
Sphalerite concentrates....do....	34.48	40.85	39.40	40.80	49.99	39.93	38.65	72.68	77.94
Zinc silicates and carbonates....do....	20.90	23.74	23.58	23.76	28.95	22.45	21.13	45.00	50.21
SHEET GROUND.									
Total crude ore....short tons..	3,422,976	4,994,123	5,779,192	4,944,910	5,465,100	4,303,900	3,594,170	6,501,000	8,484,700
Total concentrates in crude ore....per cent..	3.4	3.1	2.6	2.6	2.62	2.63	2.69	2.25	2.19
Lead....do....	.6	.5	.4	.5	.46	.47	.44	.34	.33
Zinc....do....	2.8	2.6	2.2	2.5	2.16	2.16	2.23	1.91	1.86
Metal content of crude ore....do....	2.2	2.0	1.6	1.64	1.62	1.63	1.65	1.38	1.35
Lead....do....	.5	.4	.3	.4	.36	.37	.35	.25	.25
Zinc....do....	1.7	1.6	1.3	1.24	1.26	1.26	1.30	1.13	1.10
Average lead content of galena concentrates....do....	80.0	8.03	79.9	79.9	79.6	79.0	79.3	76.4	76.5
Average zinc content of sphalerite concentrates....do....	57.6	59.5	59.3	58.6	58.6	58.3	58.4	59.1	59.2
Average value per ton:									
Galena concentrates....dollars..	53.70	54.41	52.01	56.50	55.83	51.43	47.38	54.63	85.62
Sphalerite concentrates....do....	32.59	41.55	40.81	39.79	50.84	41.69	38.86	78.79	86.02

[a] Figures for 1908 from Mineral Resources of United States, 1909, U. S. Geol. Survey, 1909, Pt. I, pp. 518–19; figures for 1909–10 from Mineral Resources of United States for 1910, U. S. Geol. Survey, 1911, p. 644; figures for 1911–1915 from Mineral Resources of United States for 1915, U. S. Geol. Survey, 1916, p. 881; figures for 1916 from U. S. Geol. Survey Press Bull. 320, June, 1917, p. 7.

[b] Includes also the small production of lead and zinc ores in central Missouri.

TABLE 4.—*Tenor of crude lead and zinc ore and concentrates produced in Kansas, 1908–1916.*[a]

	1908	1909	1910	1911	1912	1913	1914	1915	1916
Total crude ore....short tons..	677,722	595,786	704,046	761,465	744,254	590,300	540,500	709,400	774,400
Total concentrates in crude ore....per cent..	4.7	4.2	4.3	3.2	3.2	4.0	4.37	....	....
Lead....do....	.5	.3	.4	.5	.4	.5	.33	.23	.29
Zinc....do....	4.2	3.9	3.9	2.7	2.8	3.5	4.04	3.92	3.15
Metal content of crude ore....do....	2.8	2.4	2.5	1.9	1.9	2.3	2.58	....	....
Lead....do....	.4	.2	.3	.4	.3	.4	.26	.17	.20
Zinc....do....	2.4	2.2	2.2	1.5	1.6	1.9	2.32	2.25	1.78
Average lead content of galena concentrates....do....	78.6	77.4	79.8	78.9	78.3	78.2	77.9	74.9	74.3
Average lead content of lead carbonate concentrates....do....	....	....	62.3	....	54.5	....	....	....	....
Average zinc content of sphalerite concentrates....do....	57.7	56.1	55.3	56.0	55.4	55.7	57.4	57.4	57.0
Average zinc content of silicates and carbonates....do....	40.0	38.0	34.0	38.5	39.2	35.9	34.2	38.0	38.0
Average value per ton:									
Galena concentrates....dollars..	53.11	48.65	48.53	54.64	52.88	50.94	42.02	52.79	73.12
Lead carbonate concentrates....do....	....	....	40.92	....	40.45	....	....	....	....
Sphalerite concentrates....do....	32.70	38.72	36.79	36.19	47.30	39.58	38 06	68.96	75.13
Zinc silicates and carbonates....do....	11.17	16.33	13.70	23.90	25.30	16.13	16.40	35.52	39.20

[a] Figures for 1908 from Mineral Resources, United States, 1909, U. S. Geol. Survey, 1909, Pt. I, p. 510; figures for 1909 and 1910 from Mineral Resources, United States, 1910, U. S. Geol. Survey, 1911, p. 623; figures for 1911 to 1915, from Mineral Resources, United States, 1915, U. S. Geol. Survey, 1916, p. 884; figures for 1916 from U. S. Geol. Survey Press Bull. 320, June, 1917, p. 4.

TABLE 5.—*Tenor of lead and zinc ore and concentrates produced in Oklahoma, 1908–1916.*[a]

	1908	1909	1910	1911	1912	1913	1914	1915	1916
Total crude ore .... short tons..	225,184	317,574	223,212	224,450	293,500	581,100	793,920	891,800	1,406,900
Miami district.									
Total concentrates in crude ore .... per cent..	10.6	9.2	11.8	7.6	7.2	5.9	5.08	....	....
Lead .... do....	2.2	2.4	2.9	2.0	2.1	1.5	1.24	1.10	1.15
Zinc .... do....	8.4	6.8	8.9	5.6	5.1	4.4	3.84	3.28	4.03
Metal content of crude ore .... do....	5.9	5.3	6.7	4.3	4.3	3.6	3.03	....	....
Lead .... do....	1.8	1.9	2.4	1.5	1.7	1.2	1.00	.86	.91
Zinc .... do....	4.1	3.4	4.3	2.8	2.6	2.4	2.03	1.82	2.33
Average lead content of galena concentrates .... do....	79.8	79.6	79.5	78.3	79.6	79.8	80.00	78.9	79.0
Average zinc content of sphalerite concentrates .... do....	48.9	50.3	48.7	50.4	51.8	53.6	54.8	55.7	57.77
Average value per ton:									
Galena concentrates .... dollars..	53.28	51.92	51.76	53.55	54.42	52.91	47.19	54.60	83.94
Sphalerite concentrates .... do....	21.56	31.21	29.32	29.50	39.80	31.50	32.66	66.48	74.80
Quapaw district.									
Total concentrates in crude ore .... per cent..	2.8	3.7	4.0	3.4	....	2.1	....	....	....
Lead .... do....	.3	.2	.3	.4	....	.2	....	.12	.07
Zinc .... do....	2.5	3.5	3.7	3.0	1.9	1.9	3.82	2.14	2.42
Metal content of crude ore .... do....	1.7	2.2	2.4	2.0	....	1.23	....	....	....
Lead .... do....	.3	.2	.2	.3	....	.13	....	.09	.06
Zinc .... do....	1.4	2.0	2.2	1.7	1.1	1.1	2.14	1.23	1.31
Average lead content of galena concentrates .... do....	79.0	79.3	78.2	79.3	79.5	78.4	79.0	76.8	77.0
Average zinc content of sphalerite concentrates .... do....	56.8	57.9	57.9	57.0	57.8	55.8	56.2	57.5	54.2
Average value per ton:									
Galena concentrates .... dollars..	51.74	51.41	50.84	55.32	50.00	51.55	50.00	53.74	71.79
Sphalerite concentrates .... do....	31.02	41.39	38.81	36.78	48.36	38.03	34.87	80.97	75.30

[a] Figures for 1908 from Mineral Resources, United States, 1909, U. S. Geol. Survey, 1909, Pt. I, pp. 526–527; figures for 1909–10 from Mineral Resources, United States, 1910, U. S. Geol. Survey, 1911, p. 857; figures for 1911–1915 from Mineral Resources, 1915, U. S. Geol. Survey, 1916, p. 665; figures for 1916 from U. S. Geol. Survey Press Bull. 320, June, 1917, p. 8.

GENERAL NATURE OF MINING AND MILLING IN THE DISTRICT

The methods used in mining and milling the ores in this district may seem crude and wasteful to one accustomed to the more elaborate methods employed in other mining districts, but careful study shows that they are well adapted to the existing conditions. The ore bodies vary widely in shape, richness, and extent, and it is difficult to calculate the probable yield of an ore body from drill-hole records, especially in other than blanket or sheet-ground deposits. Frequently, however, failure or loss on investment by the operating company has been due more to insufficient drilling and development work previous to erecting the concentrating plant than to variations of the ore body or to poor management.

The generally small size and irregular shape of the ore bodies have not encouraged the building of large, expensive plants. Economy in first cost rather than a higher saving by more elaborate and efficient equipment has been the prevailing consideration. Increased milling capacity at small cost is sought, a necessary course where the ore is so low in mineral content and the tonnage controlled by the operating company so small as at most mines in this district. Also the high royalties demanded have naturally led to small rather than large investments.

Although the controlling conditions mentioned persist with little change, both the mining and the milling practice have greatly improved during the past few years, and conditions at the mines are being bettered steadily. This result has largely come through the higher prices received for zinc and lead ores, making profitable the working of leaner deposits, and through a closer saving in mills by the use of more efficient methods. A few years ago ores with less than 4 per cent of recoverable mineral content were considered unworkable, but now many deposits of sheet-ground ore with a content of less than 3 per cent are being worked with a good margin of profit.

One of the most noticeable improvements throughout the district is the better saving of zinc and lead in the fine material. Where formerly only three or four concentrating tables were used in a mill, four or five times as many can now be found in later mills of the same capacity. The commercial saving possible by treating the finest material, or slimes, a product that for many years was wasted, is being realized more and more. The average recovery of zinc is about 65 per cent, and a few of the better equipped mills are making a recovery of about 70 per cent based on the zinc content of the ore and as high as 75 or 80 per cent when the lead recovery is included. This is good practice in view of the equipment of the mills and the tonnage that passes through them. However, there is still plenty of room for improvement in many of the mills, and it is to be hoped that the percentage of saving will show an even more marked increase during the next few years.

MINING METHODS.

UNDERGROUND WORK.

The mining methods employed vary with the nature of the deposits and have been developed to suit the conditions in the various mines. As the ore deposits are relatively flat and lie at shallow depths, they are reached by vertical shafts sunk at frequent intervals. The depth of shafts usually varies from 100 to 300 feet, although the depth is sometimes less than 100 feet in soft-ground deposits and nearly 400 feet at some mines in the Miami district, Okla. In general, the ore bodies are worked from one level, as the ore-bearing deposits are

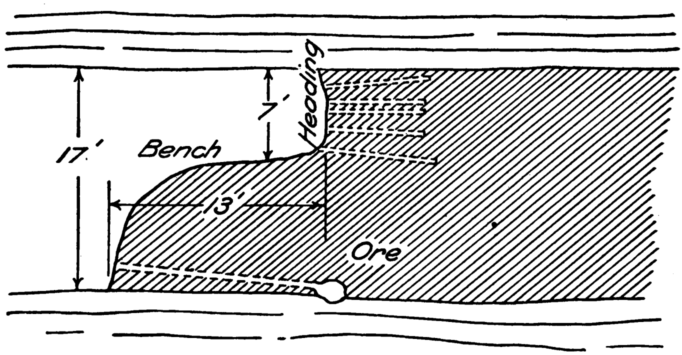


FIGURE 2.—Cross section showing underhand stoping as practiced in Joplin district.

rarely more than 30 feet thick. Where there are two distinct runs of ore, one above the other, separated by a relatively thick stratum of barren rock, the two runs are usually mined independently or separately, at different times (see Plate V).

Development work underground is usually limited, there being no marked distinction between it and the mining proper. The development work consists chiefly in sinking shafts to ore bodies that have been located by drill holes or by driving lateral drifts, and in some places raises or winzes are started in search of ore bodies indicated by mineral-bearing fissures or pockets. As a rule, the development work is not satisfactory, owing to the irregularity of the ore bodies and the lack of well-defined mineral-bearing zones.

Where possible, a system of underhand stoping (figs. 2 and 3) is practiced, especially in the "hard-ground" or "sheet-ground" mines, provided the working face is high enough to warrant stoping. Pillars are left at frequent intervals, the distance between centers and the thickness of the pillars depending upon the character of the ore body and the height of face mined. The methods of mining used may be divided into "hard-ground" and "soft-ground" mining.

In hard-ground or "sheet-ground" mining, where the height of face warrants underhand stoping (see Pl. VI), a machine drill is usually set on a 7-foot column placed at the upper part of the working face, near the roof, in order to advance a heading about 6 to 8 feet high. When the heading or breast has been advanced 15 to 25 feet the underlying bench is drilled and blasted. This bench is usually 9 to 12 feet

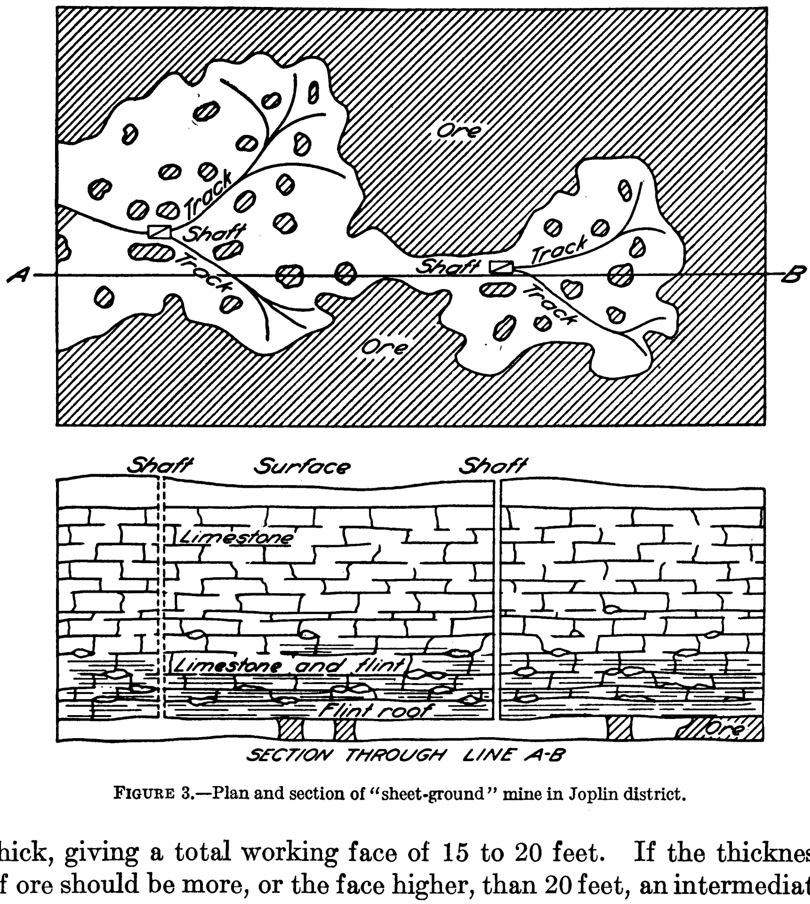
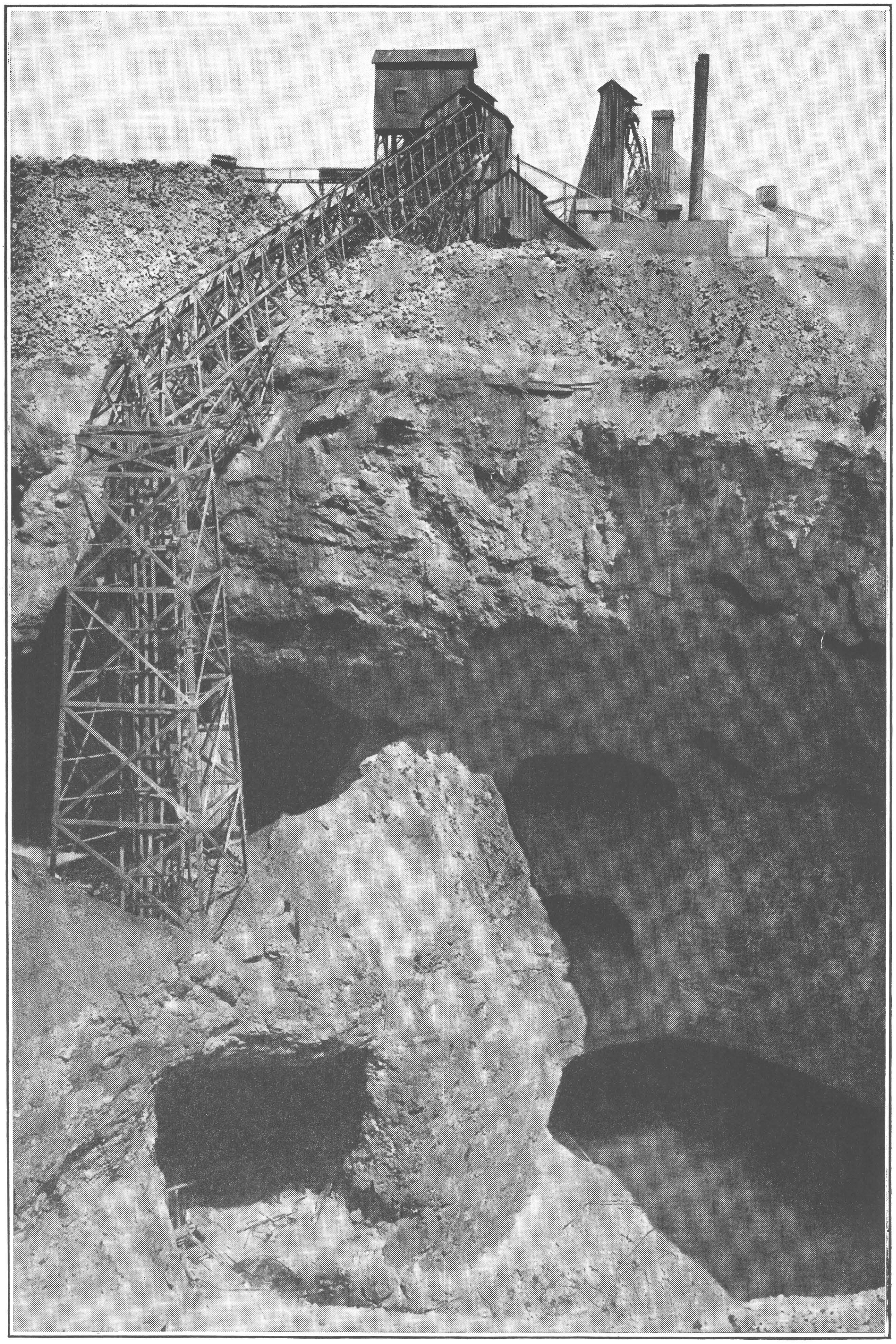
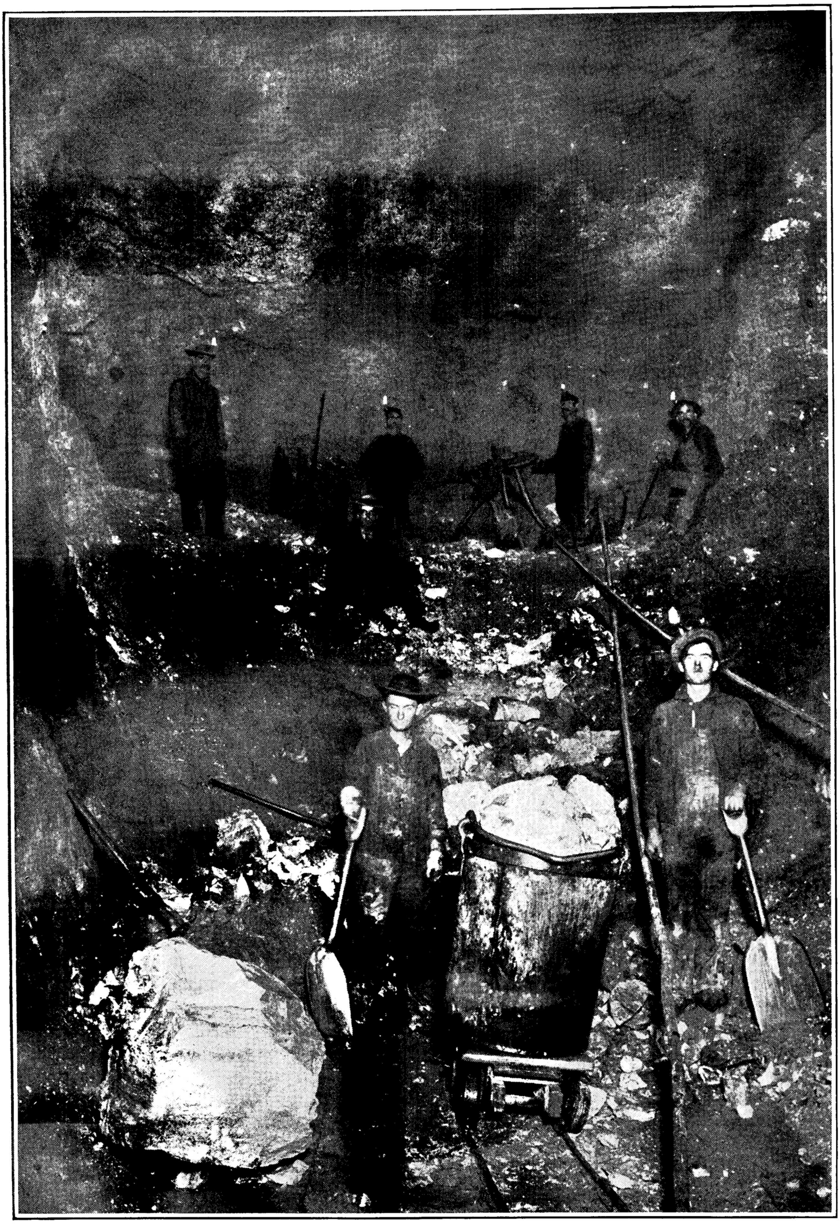


FIGURE 3.—Plan and section of "sheet-ground" mine in Joplin district.

thick, giving a total working face of 15 to 20 feet. If the thickness of ore should be more, or the face higher, than 20 feet, an intermediate bench is worked between the heading and underlying bench. In some of the hard-ground mines only the upper part of the ore-bearing deposit can be worked at a profit and in others only the lower part. This condition is chiefly due to the presence of an intermediate layer of practically barren rock, varying in thickness from 2 to 6 feet, which would have to be mined with the ore if the two ore-bearing strata were mined together. Where stoping is not carried on, or where the deposit is not more than 8 to 12 feet thick (Pl. VII), the



OLD WORKINGS. SHOWS UPPER AND LOWER WORKING LEVELS.



MINING BY BENCH SYSTEM.

ore is so mined as to leave an irregular face to the best advantage for subsequent drilling and blasting.

Where the ore deposits are "runs" or large and irregular the ore bodies are often large enough to be worked from more than one level. In a few mines this method has been used, but in many places the ore has been followed by raises from the bottom level, at which the mine was opened, to its upper limits, the broken ore being allowed to fall down a slope or chute whence it could be shoveled into cars. The method of mining used in such a case has been to follow the ore with a heading and break the underlying parts by the usual benching system. This method results in drifts or cuts 40 to 80 feet high and 20 to 40 feet wide.

Little timbering (see Pl. VIII, *A*) is done, except occasional props or posts placed wherever necessary to support slabs and to prevent possible scaling of the roof. However, pillars (see Pl. VIII, *B*) are left at intervals of 30 to 60 feet, center to center. They vary in thickness from 12 to 30 feet, depending on the nature of the deposit, the height of face, and the character and firmness of the roof and overlying strata. The average thickness of the pillars left to protect the shafts is about 50 feet. Wherever possible, the pillars are of the relatively leaner ore. The proportion of ground left as pillars is 15 to 30 per cent, although the pillars are usually trimmed or in some mines pulled after the mine has been worked out.

In the sheet-ground mines the usual roof is a layer of hard flint, 3 to 5 feet thick, which serves as a good support for the overlying beds and as a good bedding plane for the miner to work against. Where this layer is thin or lacking, more or less trimming of the roof is required after each round of shots. In a few of the mines a thin layer of white "cotton rock," or decomposed chert, 2 inches to 1 foot thick, is found in the roof. As this rock in places is relatively soft and can not be supported by posts, it has to be taken down and shoveled out with the ore, thus increasing the amount of "dead rock" to be handled and decreasing the relative richness of the ore treated.

"Soft-ground" deposits are mined by driving lateral drifts through the ore body, the walls and the roof being supported by timbering or by a combination of pillars and timbering. Usually the fore-poling system of timbering is used where the drifts are narrow and the face is relatively low, whereas square sets are used where the drifts are fairly wide and the face is relatively high. Lagging is used to prevent small pieces of rock falling into the drifts.

SHAFTS.

As the lead and zinc deposits mined lie at comparatively shallow depths, the ore bodies are reached by vertical shafts, either single or double compartment. The dimensions of the one-compartment shafts vary usually from 4 by 5 feet to 5 by 7 feet in the clear, whereas those of two-compartment shafts vary from 5 by 10 feet to 7 by 12 feet. Recently a few three-compartment shafts have been sunk.

The sinking of relatively shallow shafts, 40 to 100 feet deep, is usually a simple matter, but in the deeper shafts, particularly when a large flow of water is tapped, more or less trouble may arise. Difficulties are especially common in the Miami district when sinking to the lower ore-bearing levels, where the underground flow is considerable, and large pumps are necessary. It is believed, however, that as new shafts are sunk and new mines opened such trouble from underground water will be reduced to a minimum.

The shafts are usually sunk according to the American center-cut system and are 6 to 12 feet deeper than the mine level, to allow room for a drainage sump from which the mine water is pumped to the surface.

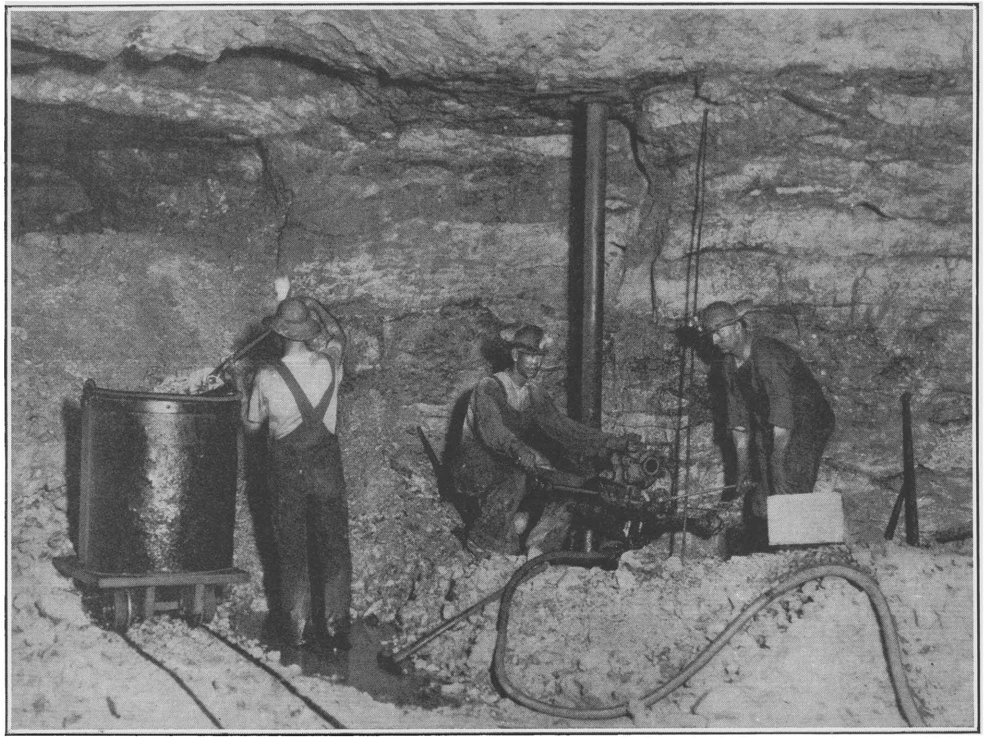
The amount of cribbing used in the shaft depends upon the character of the ground penetrated and the amount of water entering the shaft from lateral water courses. The shafts in soft-ground mines are usually lined from the collar to the mine level with close cribbing to prevent pieces of rock falling down the shaft and to keep out as much water as possible. Where the ground is more or less hard and the walls of the shaft are fairly firm, the timbers are placed one above the other, an open space alternating with each timber. The cut timbers are usually 2 by 4 inches or 2 by 6 inches in size. The lining is usually "laced," which helps to support it, keeps the shaft free from water and small pieces of rock, and gives the sides a smooth surface.

In the sheet-ground mines the shaft lining usually extends only from the collar of the shaft through the soil and less firm ground near the surface to the "hard ground" beds of rock.

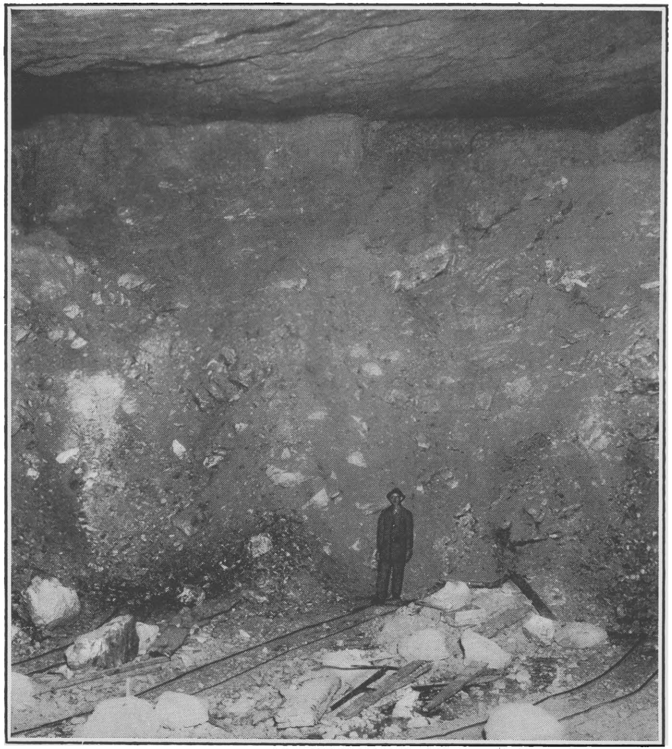
Usually the main shaft is connected directly with the main hopper of the mill. When two or more shafts are used the distance between them is about 300 feet.

METHODS OF RAISING AND SINKING SHAFTS.

At a few mines raising a shaft has been found practicable. A cable attached to a small headframe at the surface, passed through a drill hole to the underground workings, supports a platform. This platform is raised and lowered by means of the cable, and is used to support the air drill and the drill man. After a round of holes has been drilled, the holes are charged and blasted, the platform being removed before the charge explodes. By this method the face is always clear of broken rock. It has been found that the cost of raising a shaft in sheet ground is less than that of sinking to the same depth.



MINING "SHEET-GROUND" ORE WITH LOW FACE. SHOWS IRREGULARITY OF FACE, ALSO BANDING OF ORE.



A. HIGH WORKING FACE IN HARD GROUND. NOTE ABSENCE OF TIMBERING.

B. PILLAR LEFT TO SUPPORT ROOF OF RELATIVELY HIGH WORKING FACE.

The cost of sinking shafts in this district varies according to the nature of the ground, the amount of water encountered, and the size and depth of the shaft. The average cost of sinking to about 200 feet can be figured at $10 to $30 per foot.

The actual cost of raising a shaft in sheet ground 210 feet, the size of shaft being 6 by 10 feet in the clear, is shown in Table 6.

TABLE 6.—*Cost of raising shaft in sheet ground.*

[Size of shaft 6 by 10 feet in the clear; depth 210 feet.]

Small headframe for raising and lowering cable:		
Supplies	$51.96	
Labor	58.51	
		$110.47
Power machinery:		
Hoist	45.00	
Cable	33.14	
Supplies	65.33	
		143.47
Drills:		
Drills	274.94	
Steel	218.87	
		493.81
Explosives:		
Powder	454.68	
Exploders	53.59	
Fuses	39.82	
		548.09
Labor raising:		
Top	222.85	
Drilling	431.25	
Shoveling	62.97	
		717.07
Labor sinking:		
Top	126.62	
Drilling	255.25	
		381.87
Timbering:		
Cribbing	457.13	
Labor	59.01	
		516.14
Compressed air		350.00
Total cost		3,260.92
Cost per foot		15.53

In contrast to the figures above, a shaft was previously sunk in similar ground at the same mine to a depth of 240 feet at a total cost of about $5,000, or $20.83 per foot. The difference in cost per foot between the two shafts is the more marked because the deeper shaft was the smaller, measuring 5 by 10 feet in the clear, and at the time this shaft was sunk the average wage was about 50 cents lower per shift than at the time the other shaft was raised.

DRILLING.

As the chief object in mining is to excavate the most ore at the least cost, certain conditions in drilling the ore must be considered in order to obtain the best effect in subsequent blasting. First among these are the character of the rock in which the ore bodies lie—whether hard, soft, brittle, or tough—and the form of the ore deposit, whether massive or in stratified layers. Some rocks are creviced and others contain large cavities, variations that make it necessary for the drill man to use his best judgment in placing the holes so that the effect of the explosive in breaking ground will be greatest. Too much emphasis, therefore, can not be placed upon the need of ascertaining in advance, as nearly as possible, the nature of the rock to be mined. Failure to ascertain it is apt to make drilling and blasting costs unnecessarily high.

TYPES OF DRILLS USED.

Several makes of machine drills are used in this district, but practically all are percussion drills driven by compressed air. The air leaves the compressor at a pressure of 90 to 100 pounds, the pressure at the rock drill being about 5 to 10 per cent less because of leakage and other losses in transmission. At some of the mines the air compressors are overtaxed in order that as many drills as possible may be used, thus reducing still more the effective air pressure.

CHARACTER OF ROCK.

The rock drilled, chiefly flint, is hard and brittle and rapidly wears the cutting edge of the bit, but because of its brittleness it is more easily fractured by the concussion of the drill. In some of the mines the gangue rock is tough, so that the footage per shift is less, whereas in other deposits it is more or less soft and the footage for the same amount of time consumed in drilling is much greater than the average. The average footage for drilling in sheet ground is 20 to 50 feet per 8-hour shift for each drill, varying with the character of the ground and the skill of the drill man. Owing to the abrasive action of the flint on the cutting edges of bits, changes every 2 feet are often necessary, and therefore the proper tempering of the steel when bits are sharpened is of much importance.

DEPTH OF HOLES.

The length of the holes drilled varies with the height of the working face and the method of mining. Where the face is about 10 feet high, and no stoping is done, the holes are 8 to 10 feet long. When underhand stoping or benching is used the holes for advancing the heading are usually 6 to 7 feet long, whereas those drilled horizontally in the underlying bench, which is called the "stope," are usually 12 to 14 and sometimes 16 to 18 feet long. The average diameter of the drill holes is about 3 to 3½ inches at the mouth and 1¼ to 1½ inches at the bottom, depending on the length of the hole. Each drill is usually run by a drill man, known as a machine man, and one helper.

ROCK DUST AND METHODS OF ABATEMENT.

In drilling horizontal holes in dry ground, especially in sheet ground, a large proportion of the cuttings and finely powdered dust gradually passes out of the hole at the mouth. Before any "squibbing" or blasting is done, these cuttings must necessarily be removed. Until recently this was generally accomplished by inserting a long iron pipe about half an inch in diameter, connected with the air hose, into the drill hole. When the air under pressure was turned on, this fine flinty dust was blown from the hole into the drift, where the miners inhaled it. Drilling and blowing dry holes has been one of the chief causes of lung diseases among the miners of the Joplin district. It is believed that much of what is known as "miners' consumption" is caused by sharp particles of siliceous dust, especially flint, being embedded in the lung tissue, forming small receptacles, and that the lungs when affected in this way are much more susceptible to the germs of tuberculosis and other diseases. In view of the particularly harmful effect of the rock dust in the sheet-ground mines, a cooperative investigation was carried on in the Joplin district by the Federal Bureau of Mines, the United States Public Health Service, the State mine inspectors, and the mine operators.

The investigation was conducted by A. J. Lanza, passed assistant surgeon of the Bureau of Public Health Service, and Edwin Higgins, formerly of the Bureau of Mines, under the supervision of G. S. Rice, chief mining engineer of the Bureau of Mines. A survey was made of the conditions in the mines with special reference to rock dust produced in the sheet-ground mines by drilling, blasting, and shoveling. The results of these investigations were published shortly afterwards, in the form of a preliminary report.[a]

Following this investigation the State mine inspectors recommended certain regulations and laws to be passed for the abatement of unnecessary dust in the mines and of other insanitary conditions affecting the health of the miners. A final report of the investigation has since been published by the Bureau of Mines as Bulletin 132.[b]

Owing to the more stringent regulations adopted and the careful and efficient inspection of the mines by the State officials, the drill

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[a] Lanza, A, J., and Higgins, Edwin, Pulmonary disease among miners in the Joplin district, Missouri, a preliminary report: Technical Paper 105, Bureau of Mines, 1915, 48 pp.

[b] Higgins, Edwin, Lanza, A. J., Laney, F. B., and Rice, G. S., Siliceous dust in relation to pulmonary disease among miners in the Joplin district, Missouri: Bull. 132, Bureau of Mines, 1917, 112 pp.

cuttings are now removed from the holes by the use of a water hose, which is also used for spraying down the faces and broken ground, thus reducing to a minimum the rock dust suspended in the air. The State mine inspectors at the time of this investigation were Ira L. Burch, Walter W. Holmes, and C. M. Harlan.

PLACING OF HOLES.

To insure efficiency in blasting careful attention is necessary in drilling and placing the holes. The miner or machine man should be familiar with the nature of the ground, in order to arrange his holes to assure the most effective results. He should ascertain the direction of the line of least resistance and drill the holes in such a direction that the charge will break most effectively with the minimum amount of powder that part of the face which he wishes to remove. Other important factors that should be considered in breaking ground are the irregularities of the face and the form in which it will be left after the round of shots is fired. The mine foreman should constantly try to foresee the condition of the face after shooting, and should have the machine men follow, as far as practicable, a system that leaves the face most accessible for subsequent drilling and blasting. As the use of explosives is one of the most important items of cost in hard-ground mining in this district, their use to the best advantage should always be considered. These considerations are followed in many of the mines throughout the district.

BLASTING.

The effectiveness or efficiency of an explosive is usually measured by the quantity of rock broken and the quality of the product. The gases generated by the shot exert pressure in every direction, and the result will depend on the character of the rock in which the explosive is used, the kind of explosive, the manner in which it is used, and the method by which it is exploded.

CHARACTER OF EXPLOSIVES USED.

The strength of the explosive most used in this region is equivalent to that of a 35 to 40 per cent "straight" nitroglycerin dynamite. The "straight" nitroglycerin dynamites are quick acting and are said to have a greater disruptive force than other commercial types of explosives.[a]

In wet holes gelatin dynamite is used, as it is relatively impervious to water, whereas for dry holes the low-freezing ammonia dynamites are much used. The ammonia dynamites have a lower rate of deto-

[a] Munroe, C. E., and Hall, Clarence, A primer on explosives for metal miners and quarrymen: Bull. 80, Bureau of Mines, 1915, p. 22.

nation and their action is more a pushing and less a shattering. They have the disadvantage, however, of taking up moisture, and consequently must be kept in a dry place.

STORAGE OF EXPLOSIVES.

Explosives reach the mines in 50-pound boxes containing 80 to 140 cartridges, according to the kind of explosive used. The usual supply kept at the mines is 15 to 40 boxes; these are stored on the surface in a magazine situated about 300 feet from the mill and other buildings. The magazines are generally built of wood with double walls, the space between the walls being filled with tailings or manure, and many are surrounded on three sides by tailings. These buildings are used not only as magazines, but as thawing houses, and are equipped with steam pipes, or other means of heating, to keep the dynamite at a temperature above its freezing point. Some of the mines have small magazines underground but these are used only for the storage of fuse and detonators and for the day's supply of explosive. These are situated away from the main drifts and working faces.

LOADING THE HOLES.

The cartridges are prepared and are loaded into the holes by a "powder man," who usually has a helper. Each stick of dynamite is sliced down the side and is loaded separately into the hole with a pointed wooden bar. Chambering of holes or "squibbing," preliminary to inserting the final charge, and the firing of one or two sticks in "blocked" holes have in the past usually been done by the machine men at noon or during the day, but new regulations require this to be done when the men come off shift, in order that the amount of dust thrown into the mine air while miners are at work may be minimized.

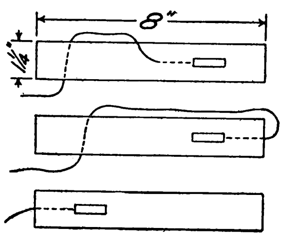

FIGURE 4.—Three ways of inserting detonator in primer as practiced in Joplin district.

After a sufficient charge has been loaded into the hole, the primer or cartridge containing the cap (detonator) and fuse is inserted. Three different ways in which the cap is inserted in the primer are shown in figure 4.[a] The primer is usually followed by one or two sticks of dynamite which are tamped with a wooden bar. At a few of the mines specially prepared sticks of stemming[b] material are used, whereas, at others no stemming is used. The men as a

[a] The Bureau of Mines recommends that the detonator be inserted in the end of the primer as in the bottom diagram in figure 4. To lace fuse through a cartridge of high explosive is dangerous because the explosive, ignited by side spitting of the fuse, may burn before exploding, with the result that poisonous gases will be produced. Detonator should not be inserted so far in the cartridge as is indicated in figure 4.

[b] In the publications of the Bureau of Mines the word "stemming" is used to designate the material placed on a charge of explosive in a drill hole, and the word "tamping" to designate the act of placing and ramming the material.

rule do not care to use specially prepared stemming, claiming that its use produces no better results and involves a loss of time. This opinion seems hardly based on proof, in view of the limited amount of experimenting done with explosives in this district; for this reason some of the results of tests made at the Pittsburgh station of the Bureau of Mines [a] to determine the advantages from the use of stemming are given below.

ADVANTAGES OF USING STEMMING.

In carrying out these tests, the Trauzl lead block—a lead cylinder 20 cm. (7⅞ inches) high and 20 cm. (7⅞ inches) in diameter, with a bore hole 2.5 cm. (1 inch) in diameter and 12.5 cm. (5 inches) long—was used, as it furnishes one of the simplest means for measuring the relative strength of high explosives.

A known amount of the explosive under test is placed in the bore hole and then fired. The pressure of the gases evolved by the explosive expands the hole into a pear-shaped cavity. The original volume of the bore hole is determined by measuring the amount of water required to fill it exactly; the volume, after a test, is determined in the same way. The difference between the volume of the hole after the firing of the charge and the volume before firing is taken as a measure of the force exerted by the explosive.

The different kinds of stemming used in the tests were dry sand, both tamped and untamped; dry fire clay, tamped and untamped; moist sand, tamped; and moist fire clay, tamped. The table below shows the effect of using different weights and kinds of stemming.

TABLE 7.—*Expansion of bore hole in lead blocks by 20 grams of explosive with different kinds and weights of stemming.*

FORTY PER CENT STRENGTH AMMONIA DYNAMITE.

Kind of stemming.	Weight of stemming, grams.						
	0	6.25	12.5	25	50	100	200
Untamped dry sand .... expansion of bore hole, c. c..	230	304	359	408	430	434	446
Untamped dry fire clay .... do....	230	288	347	385	387	381	378
Tamped dry sand .... do....	230	......	355	379	393	397	417
Tamped dry fire clay .... do....	230	285	339	383	389	379	381
Tamped moist sand .... do....	230	265	293	375	389	419	470
Tamped moist fire clay .... do....	230	280	341	382	407	462	487

FORTY PER CENT "STRAIGHT" NITROGLYCERIN DYNAMITE.

Kind of stemming.	0	6.25	12.5	25	50	100	200
Untamped dry sand .... expansion of bore hole, c. c..	367	462	504	569	587	636	642
Untamped dry fire clay .... do....	367	460	496	574	578	590	589
Tamped dry sand .... do....	367	454	503	591	592	615	612
Tamped dry fire clay .... do....	367	385	504	601	602	629	635
Tamped moist sand .... do....	367	465	525	603	645	697	696
Tamped moist fire clay .... do....	367	507	547	623	643	696	710

[a] Snelling, W. O., and Hall, Clarence, The effect of stemming on the efficiency of explosives: Tech. Paper 17, Bureau of Mines, 1915, p. 20.

According to Snelling and Hall[a] the results obtained from the experiments prove definitely that confinement by the use of stemming greatly increases the efficiency of a charge of explosive.

As the expansion of the bore hole of the lead blocks in the experiments is a good measure of the pressure that is exerted on the wall of a drill hole by the same explosives in actual mining operations, it is reasonable to assume that the useful work done by the explosives would be in the same ratio. The results of the tests show that with quick-acting explosives, such as "straight" dynamite, a small quantity of stemming greatly increases the efficiency, whereas a large quantity is required with slow-burning explosives, such as black blasting powder, for effective results. In addition, the use of stemming reduces the evolution of poisonous gases to a minimum.

FIRING OF SHOTS.

In general, the holes are drilled and "squibbed" on one day, and the loading and blasting are done on the next day. The machine men, as a rule, light their own shots after all other men are out of the mine. The time of shooting is about 4 p. m., or between 12 and 1 o'clock at night, if there are two underground shifts.

The quantity of explosive used per ton of ore broken varies from 1 to 1.5 pounds in hard ground and from 0.2 to 0.6 pound in soft ground, depending on the nature of the ground. To reduce misfires and, consequently, to lessen the quantity of powder used per ton of rock broken, high-grade detonators are recommended. With stronger detonators and lower-grade dynamite it is possible to obtain, in most instances, as good results as with weaker detonators and higher-grade dynamite. As the lower-grade powders are cheaper, their use with the more efficient detonators should reduce the cost per ton of rock broken. The lower-grade dynamite also has a less shattering effect, thus reducing the amount of fines produced before the ore reaches the mill and making possible a saving of mineral that might otherwise be lost as slimes.

The actual quantities of dynamite used per ton of ore broken in three different mines are shown in Table 9.

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[a] Snelling, W. O., and Hall, Clarence, The effect of stemming on the efficiency of explosives: Tech. Paper 17, Bureau of Mines, 1915, p. 15.

TABLE 8.—*Actual quantities of dynamite used per ton of ore broken in three mines.*

Year	Quantity of dynamite used per ton of ore. 1914	1915	1916	Remarks.
	Pounds.	*Pounds.*	*Pounds.*	
Mine 1	1.417	1.395	1.326	Sheet-ground, hard, tough, face 12 to 15 feet high.
Mine 2	1.655	.784	.874	Sheet-ground, hard; 1914 included heading, 1915 and 1916 stope only (taking up floor of mine).
Mine 3	1.056	1.326	1.261	Hard ground, Miami district, near Commerce, high face.

SHOVELING AND UNDERGROUND TRAMMING.

After the ground has been broken the ore is shoveled into cars or cylindrical tubs which are trammed by the shovelers or are hauled on a small truck by mules from a lay-by (siding) to the shaft. The tubs, locally known as "cans," usually measure 30 by 30 inches in diameter and hold about 1,000 pounds of broken ore. A piece of sheet iron or a small floor of wooden planks is laid so as to provide a flat and smooth surface from which to shovel. The bowlders and larger pieces of rock are lifted into the tub, those too large for one man to handle being broken with sledges or blasted. The blasting of bowlders is known as "bowlder popping." The shovelers are good workmen, as a rule, averaging about 20 tons per man per shift. They are usually paid by the car or tub. This system incites the men to work hard and tends to increase the tonnage, but has the serious disadvantage that the men are apt not to fill the tubs full in order to get a greater number to their credit; this drawback can be overcome only by careful supervision. As a result of this tendency of the men and of the fact that the cars or tubs are rarely weighed, the reported tonnage of ore hoisted is apt to be somewhat higher than the actual quantity. If the ore hoisted were weighed, the men could be paid by the ton, or could receive a daily wage combined with a bonus for extra tonnage above a required amount.

Besides the usual underground haulage system already mentioned, both gasoline and electric motors are used in some of the larger mines with satisfactory results. (See Pl. IX, *A*.) At one or two mines a system of rope haulage, driven by electric motors, is used to good advantage. (See Pl. IX, *B*.) Traction motors, where used, necessitate a wider track gage and heavier rails.

TIMBERING.

The main object of timbering in mines is not to support the total weight of the overlying strata but to keep the roof in place and prevent falls of rock and ore. (See Pl. X, *A*.) Cracking or gradual breaking of timbers warns the workmen to place additional timbering or to escape before a fall occurs. In the best practice the timbers are so placed that the pressure is evenly distributed and in line with the thrust that they are to resist. The joints are so made that the pressure, up to the crushing strength of the timber, tends to hold the structure together rather than to weaken it.

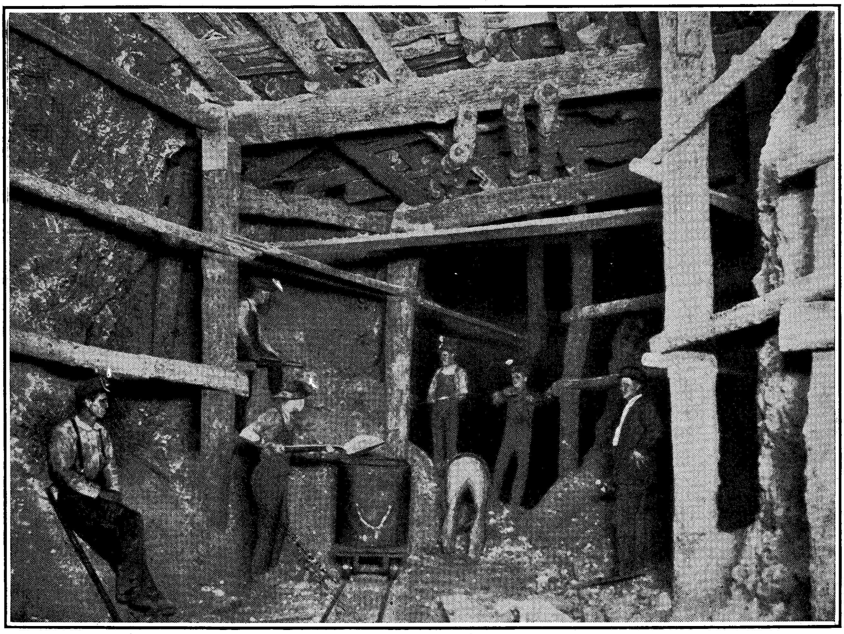
In the mines of the Joplin district, timbering is rarely used in either the sheet-ground or the hard-ground deposits, for the roof is generally firm and pillars are left at intervals of 25 to 60 feet, center to center, according to the nature of the ground and the height of the roof above the floor. Posts are used to support dangerous slabs of rock, and in some of the mines cribs filled with waste rock are used as roof supports.

In the soft-ground deposits, however, timbering is largely employed. These deposits, as already mentioned, are mined by lateral drifts in the ore body, the roof and walls being supported by timbering or by a combination of pillars and timbering. Both fore-poling and square-set systems are used. Where the drifts are narrow and the height of the face is relatively low, fore-poling is used to good advantage, the square-set system being more applicable to fairly wide drifts and relatively high faces of ore. The drifts are seldom timbered more than two square sets high. In a few mines reinforced concrete pillars (Pl. X, *B*) have been used as supports, especially at the shaft, with more or less satisfactory results. The concrete mixture is poured through a drill hole from the surface into a form placed in the underground workings just below the drill hole.

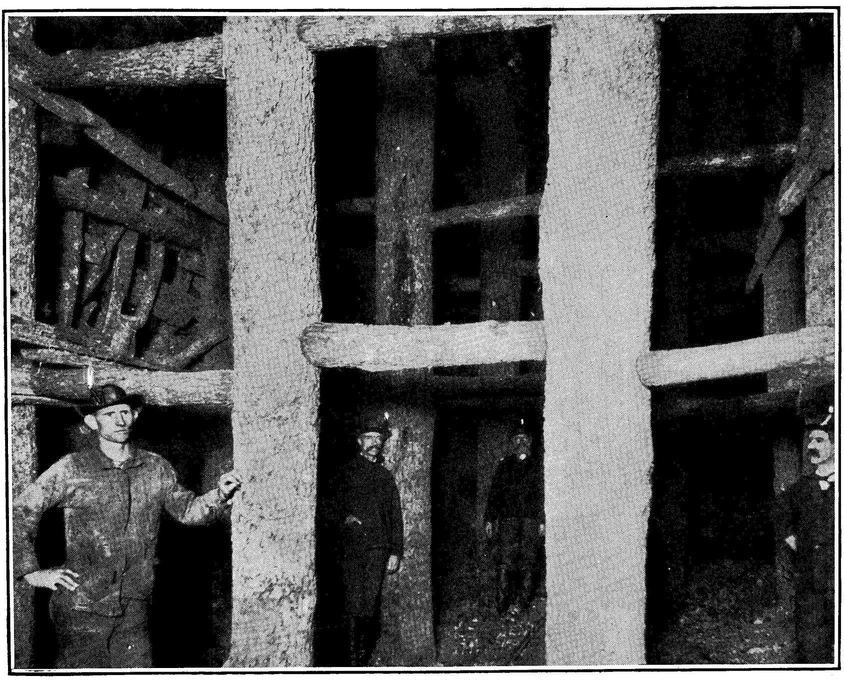
Plate XI shows the general methods of timbering used in the district.

PUMPING.

The quantity of water pumped from the mines in the Joplin district varies from about 200 to 2,000 gallons a minute. The working levels of the sheet-ground mines are as a rule fairly dry, whereas in many of the soft-ground mines much water is encountered. The water usually enters the mine through watercourses or by intermittent seepage, and unless the workings have a natural drainage outlet, it has to be pumped out. Seepage in the mines is rather general throughout the district, as the ore deposits are flat and relatively shallow. At a few of the mines underground watercourses have necessitated heavy pumping, especially in the Miami field, and in several instances have caused much difficulty in the sinking of shafts. The usual practice is to drain the underground water to a sump at the foot of a shaft, either by means of ditches along the drifts or by some natural drainage course, and then pump it to the surface.



A. FOREPOLING IN SOFT GROUND.



B. METHOD OF TIMBERING IN SOFT-GROUND MINING.

Mine pumps commonly used in this district include the usual piston types driven by steam and electrically driven centrifugal pumps, their size being governed by the flow of water to be handled. In a few places in the district, pumping is on a cooperative basis, the water from several mines being handled by one pumping station and each mine paying an apportioned sum to the company doing the pumping.

Because of the mines being shallow, and the topography having slight relief, a heavy rainfall greatly increases the amount of water that enters the workings; and if the pumping equipment at such times is insufficient the mine is shut down temporarily, causing loss of both time and money.

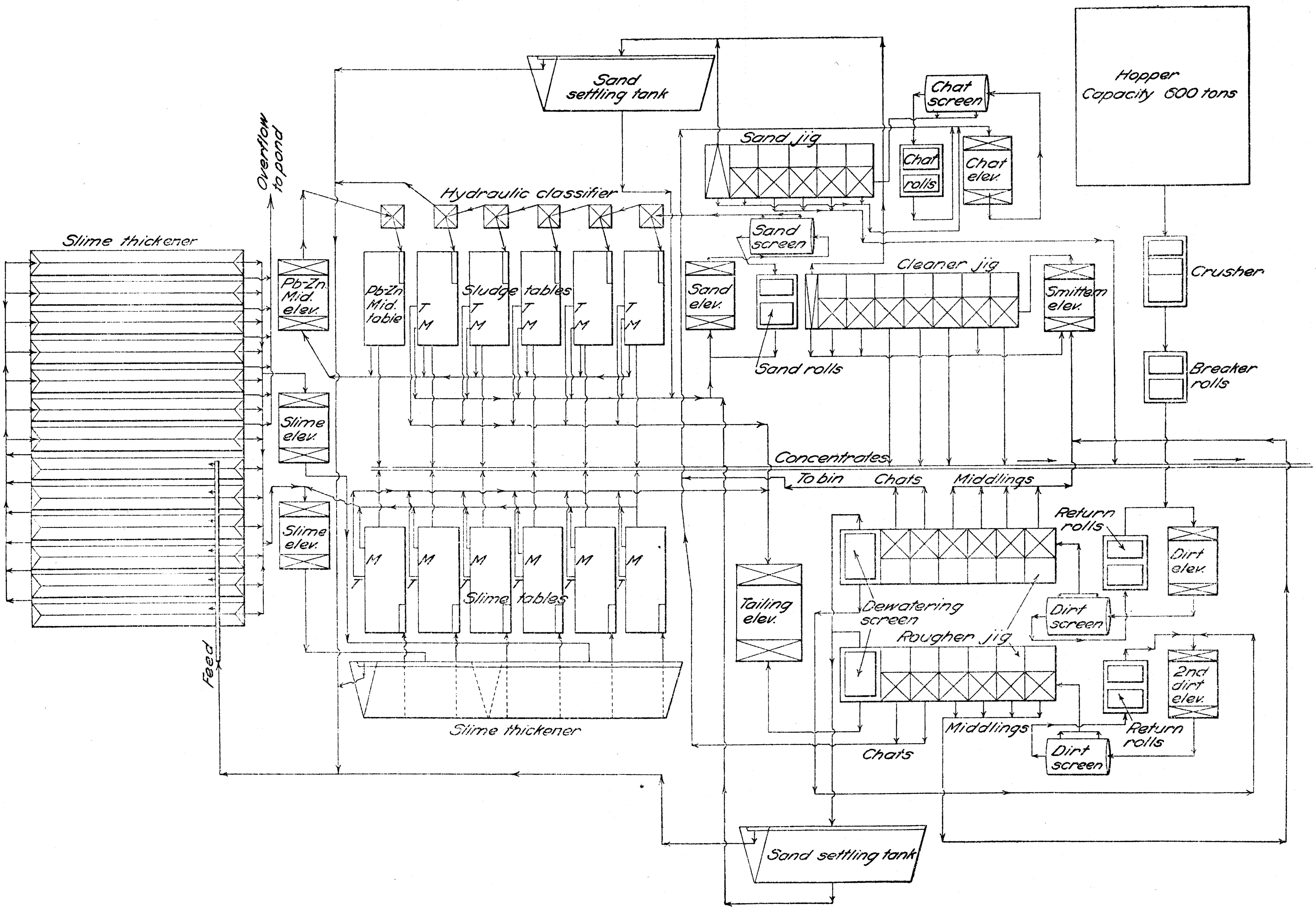
HOISTING AND SURFACE TRAMMING.

Direct winding without any attempt to balance the load is the usual practice, the tub or skip being hoisted by power and lowered by gravity. The hoisting engine is coupled or geared direct to the shaft of the drum, which is provided with friction devices or positive clutches or brakes. Steam is the usual power, though electric hoists are in use at many mines. Cars, if used, are placed in cages and hoisted in the usual way. When a tub has been hoisted to the top of the headframe the hoistman covers the opening with a lid to prevent any material from falling down the shaft, and then dumps the load on a chute leading directly to the grizzly over the hopper. Cages and self-dumping skips of 1 to 3 tons capacity, provided with shaft guides and safety catches, are used at a few mines.

Connected with each shaft is a storage bin or hopper, and with few exceptions the mill hopper is connected directly with the main shaft. The ore hoppers have a capacity of 100 to 500 tons each, although at some mines the capacity of the mill hopper is more than 500 tons in order to hold ore enough to supply the mill during an extra shift. If hoisting is done from more than one shaft, the ore is trammed to the mill hopper over inclined or horizontal tramways, or, where the distance is too great, by aerial tramways or surface motor haulage. The capacity of the surface tram cars is 1 to 3 tons.

POWER.

Electricity, natural gas, coal, and oil are all used in the district as sources of power. Natural gas seems to be most favored, owing to its cheapness; coal and oil the least. However, during the winter months when the gas pressure becomes low, coal is used to a large extent. At many of the properties the mill is driven by electric motors or gas engines and the hoisting and compressor engines are driven by steam, whereas at others electric power is used throughout.



FLOW SHEET OF MILL IN PROSPERITY DISTRICT, MISSOURI.

ORE-DRESSING PRACTICE.

In general, the average lead and zinc content of the ores mined and milled is low as compared with that of other lead and zinc fields throughout the United States. Owing to this fact and to the high grade of concentrate produced, the ratio of concentration is unusually high.

GRADE OF CONCENTRATE AND PERCENTAGE OF RECOVERY.

RESULTS OF MILL TESTS.

The object of most of the operating companies has been to produce as high a grade of concentrate as possible in order to receive the highest base price for their product, a practice that has resulted in lowering the percentage of zinc recovered. The average percentage of zinc recovery is 60 to 70 per cent; or, in other words, for every 2 tons of zinc concentrate produced 1 ton is lost. The percentages of recoveries from actual mill tests are shown in Tables 9 to 11, on pages 44 to 52.

Concentration of the ore is commonly effected by crushing to ½-inch size and roughing and cleaning the crushed material over two or more large Cooley jigs of the Harz type, and running the finer sands over the usual types of sand and slime tables. The treatment of slimes by flotation is being tried at several of the larger mills. The various steps in the concentration of the ores in this district are shown more fully by the typical flow sheets in figures 5 to 8 and Plate XII. The significance of the numbers on each figure are shown by the tabulation opposite the figure.

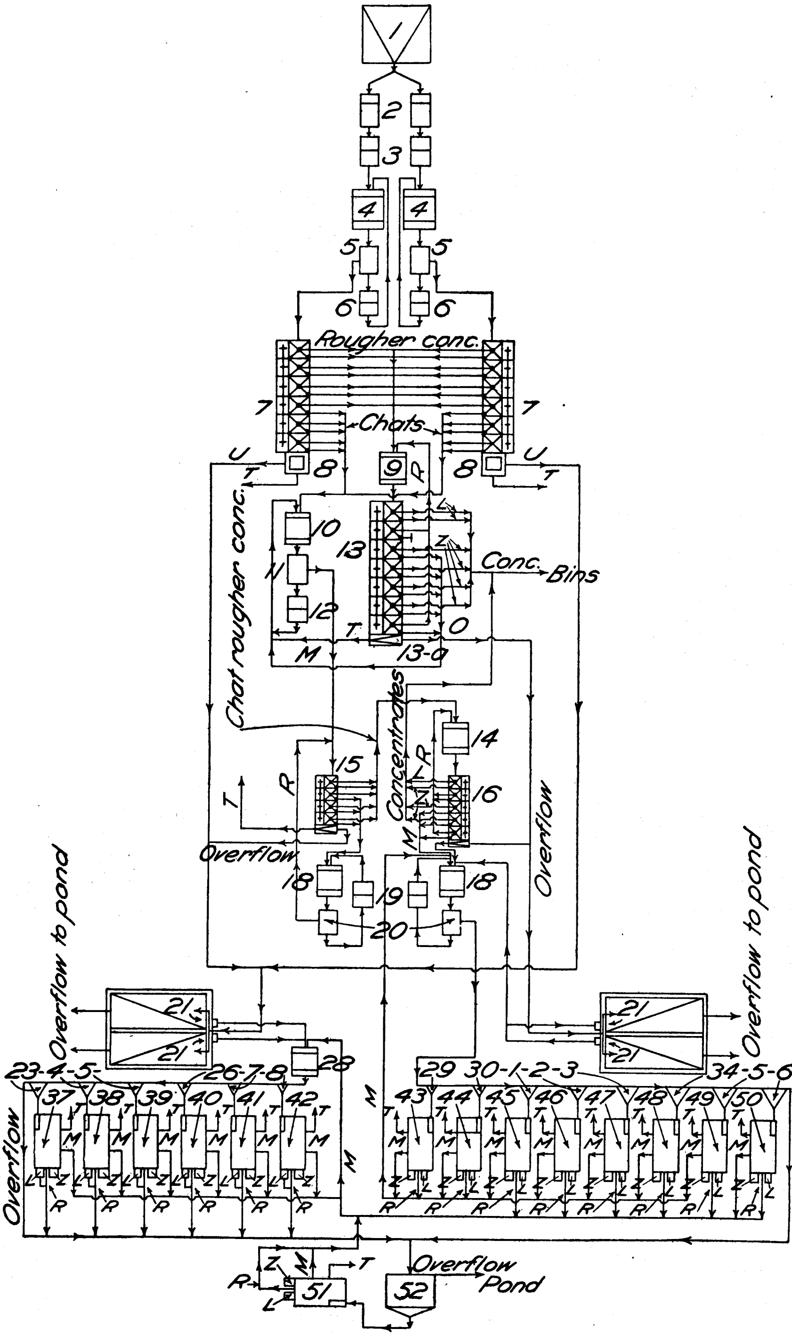


FIGURE 5.—Flow sheet of mill in Oronogo district, Missouri. *L*, lead concentrate; *Z*, zinc concentrate; *M*, middling; *O*, overflow; *R*, return; *T*, tailing. For significance of numbers see table on opposite page.

Data on parts shown in figure 5.

1. Hopper, 400-ton capacity.
2. Crushers (two), 18 by 12 inches.
3. Rolls (two), 36 by 14 inches.
4. Elevators (two).
5. Trommel screens (two), 8 by 4 feet, ⅜-inch openings.
6. Rolls (two), 30 inches in diameter.
7. Rougher jigs (two), cells 36 by 48 inches.
8. Dewatering screens (two).
9, *10*. Elevators.
11. Trommel screen, 8 by 4 feet, ¼-inch openings.
12. Rolls, 36 inches in diameter.
13. Cleaner jig, cells 30 by 42 inches.
13a. Dewatering box.
14. Elevator.
15. Rougher chat jig, cells 30 by 42 inches, with dewatering box.
16. Cleaner chat jig, cells 26 by 36 inches, with dewatering box.
18. Elevators (two).
19. Rolls (two sets), 24 inches in diameter.
20. Trommel screens (two), 36 by 72 inches, size of openings 3 mm.
21. Settling tanks.
22. Elevator.
23-36. Hydraulic classifiers.
37-51. Tables.
52. Tank.

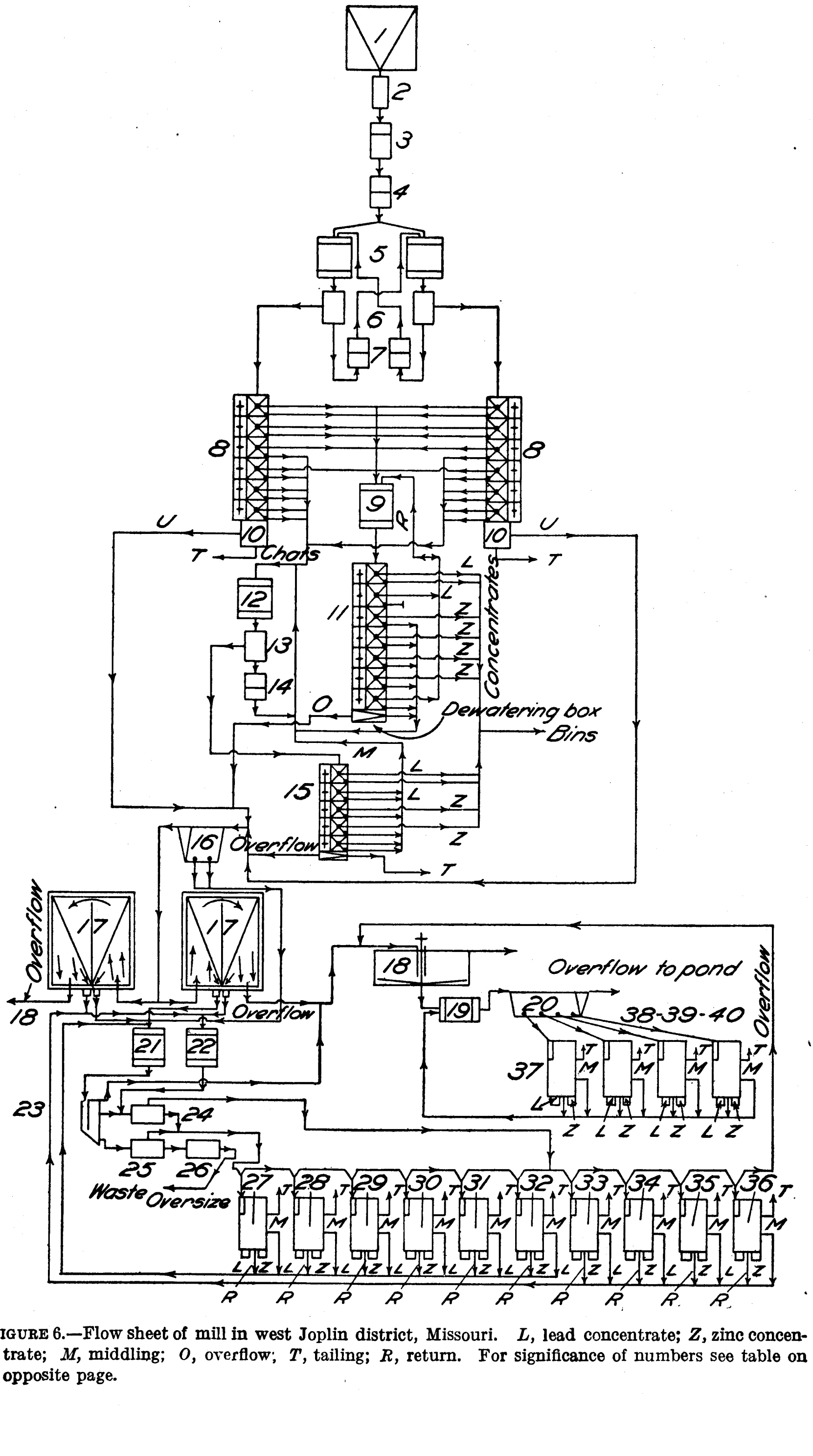


FIGURE 6.—Flow sheet of mill in west Joplin district, Missouri. *L*, lead concentrate; *Z*, zinc concentrate; *M*, middling; *O*, overflow; *T*, tailing; *R*, return. For significance of numbers see table on opposite page.

Data on parts shown in figure 6.

1. Hopper, capacity 400 tons.
2. Shaking screen, 20 inches by 4 feet 6 inches, with ¾-inch openings.
3. Crusher, 18 by 12 inches, speed 390 r. p. m.
4. Rolls, 42 by 16 inches, speed 27 r. p. m.
5. Elevators (two), 20 inches, buckets spaced 10 inches, speed 275 feet per minute.
6. Trommels (two), 4 by 12 feet, ½-inch openings, speed 25 r. p. m.
7. Return rolls (two), 36 by 14 inches, speed 30 r. p. m.
8. Rougher jigs (two), Cooley type, 6 cells, 36 by 48 inches, speed 110 strokes per minute.
9. "Smittem" elevator, 16-inch, 320 feet per minute.
10. Dewatering trommels (two), 5 by 5 feet, 2-mm. screen openings, speed 1½ r. p. m.
11. Cleaner jig, Cooley type, 7 cells 32 by 42 inches, speed 180 strokes per minute.
12. Chat elevator, 16-inch, speed 300 feet per minute.
13. Trommel, 4 by 8 feet, screen opening ¼-inch, speed 26 r. p. m.
14. Chat rolls, 30 by 14 inches, speed 56 r. p. m. (spaced rolls).
15. Chat jig, Cooley type, 5 cells 32 by 42 inches, speed 180 strokes per minute.
16. V tank, used to eliminate large proportion from feed to settling tanks.
17. Settling tanks (two), double tanks, 30 by 40 feet (plan view), 8 feet deep at discharge, 2 feet deep at back end near partition.
18. Dorr thickener, 40 by 8 feet, speed 10 revolutions per hour.
19. Slime elevator, 14-inch, 350 feet per minute.
20. V tank, discharge to slime tables, overflow to pond.
21. Coarse-sand elevator, receiving sand from first half of settling tank, 14-inch, speed 350 feet per minute.
22. Fine-sand elevator, receiving sand from second half of settling bank, 14-inch, speed 350 feet per minute.
23. Bird classifier, discharge products to trommels, overflow to Dorr thickener.
24. Trommel, 3 by 4 feet, speed 14 r. p. m., ¾-mm. screen opening, undersize to classifier above table *33* (seventh table).
25. Trommel, 4 by 8 feet, speed 14 r. p. m., 1¼-mm. screen opening, receiving oversize from trommel *24* and coarse discharge from classifier *23*. Undersize to tables.
26. Trommel, same as No. *25*, receives oversize from trommel *25*. Oversize goes to waste as tailing, undersize to table feed.
27–32. Concentrating tables, feed consisting of undersize from trommels *25* and *26* and oversize from trommel *24*.
33–36. Concentrating tables, feed consisting of undersize from trommel *24* and overflow from classifier to table *33*. Overflow from classifier to table *36* goes to Dorr thickener.
37–40. Concentrating tables, receiving spigot discharge from V tank *20*.

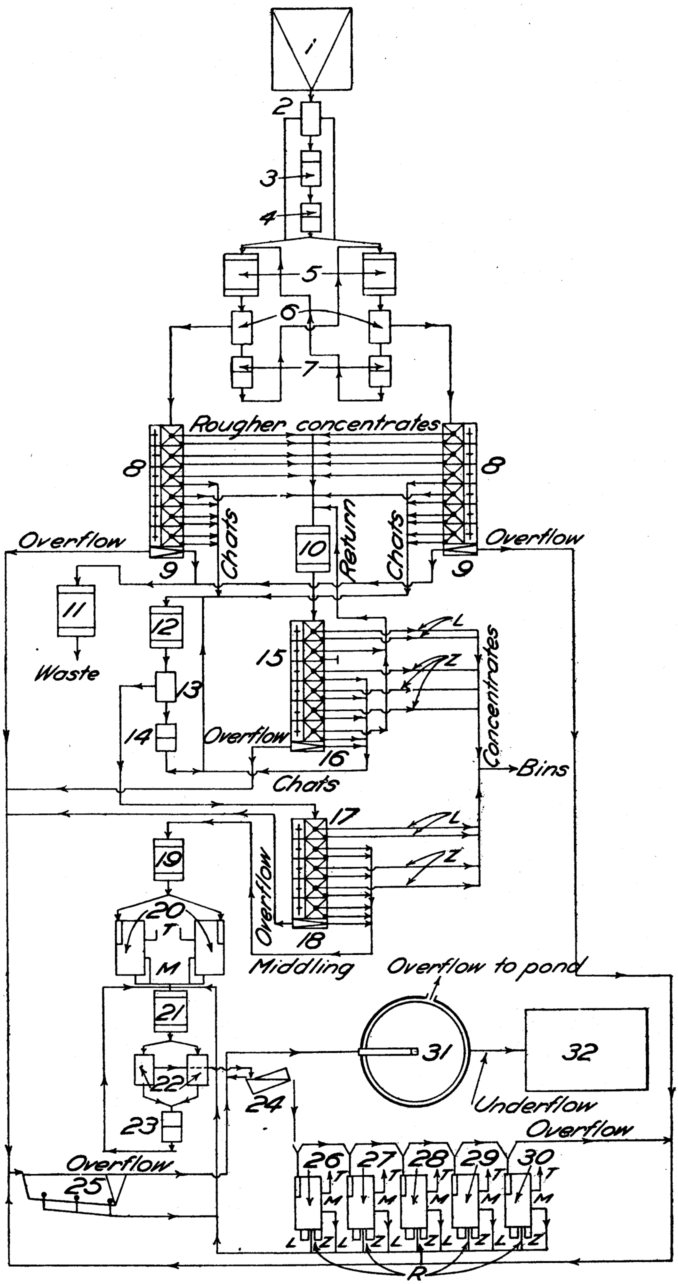


FIGURE 7. Flow sheet of mill in Granby district, Missouri. *L*, lead concentrates; *Z*, zinc concentrates; *M*, middling; *T*, tailing; *R*, return. For significance of numbers see table on opposite page.

Data on parts shown in figure 7.

1. Mill storage bin, 400-ton capacity, all ore through 6-inch grizzly.
2. Shaker screen, $\frac{1}{8}$-inch plate with $1\frac{1}{8}$-inch openings, stroke $1\frac{1}{4}$ inches, speed 125 strokes per minute.
3. Crusher, Blake type, size 24 by 12 inches, speed 350 r. p. m., set for $1\frac{1}{2}$-inch opening.
4. Cornish rolls, size 42 by 16 inches, geared, speed 30 r. p. m.
5. Feed elevators (two), 20-inch belt, buckets spaced 20 inches apart, belt speed 300 feet per minute.
6. Trommels (two), size 48 by 96 inches, slope 1 inch per foot, speed 28 r. p. m. Screen is $\frac{1}{8}$-inch plate with $\frac{3}{8}$-inch round, punched holes.
7. Cornish rolls (two), size 36 by 14 inches, geared, speed 36 r. p. m.
8. Cooley jigs (two), Harz type, six cells each 36 by 48 inches in size. Screens (grates) are punched plate with slotted holes. Sizes of screen openings: Cells 1 to 3, $\frac{3}{8}$-inch, cells 4 and 5, $\frac{1}{8}$-inch, cell 6, $\frac{1}{12}$-inch. Lengths of plunger stroke for cells 1 to 6 are $1\frac{1}{4}$, $1\frac{1}{4}$, $1\frac{5}{8}$, $1\frac{1}{2}$, $1\frac{3}{8}$, $1\frac{5}{16}$ inches, respectively. Speed 110 strokes per minute.
9. Dewatering box.
10. Elevator ("smittem"), 16-inch belt, speed 400 feet per minute.
11. Elevator (tailings), 20-inch belt, belt speed 420 feet per minute.
12. Elevator (chats), 16-inch belt, belt speed 400 feet per minute.
13. Trommel, size 48 by 72 inches, $\frac{3}{16}$-inch round holes in screen, speed 28 r. p. m.
14. Rolls (chats), Cornish type, geared, size 30 by 14 inches.
15. Cooley jig, Harz type, with six cells each 36 by 42 inches in size. Screens (grates) are punched plate with slotted holes. Sizes of screen openings: Cell 1, $\frac{3}{16}$ inch, cells 2 to 5, $\frac{1}{8}$ inch, cell 6, $\frac{1}{12}$ inch. Lengths of plunger stroke for cells 1 to 6 are $\frac{11}{16}$, $\frac{11}{16}$, $\frac{5}{8}$, $\frac{5}{8}$, $\frac{5}{8}$, $\frac{5}{8}$ inch, respectively. Speed 165 strokes per minute.
16. Dewatering box.
17. Cooley jig, Harz type, with five cells each 36 by 42 inches in size. Screens (grates) are punched plate with slotted holes. Sizes of screen openings: Cells 1 to 4, $\frac{1}{12}$-inch, cell 5, $\frac{1}{8}$-inch. Lengths of plunger stroke for cells 1 to 5 are $\frac{13}{16}$, $\frac{13}{16}$, $\frac{5}{8}$, $\frac{5}{8}$, and $\frac{11}{16}$ inch, respectively. Speed 180 strokes per minute.
18. Dewatering box.
19. Elevator (for "rougher" tables), 16-inch belt, speed 440 feet per minute.
20. Rougher tables (two).
21. Elevator (sand), 20-inch belt, belt speed 440 feet per minute.
22. Trommels (two), size 48 by 96 inches, $1\frac{1}{2}$-inch round holes in screen. Speed 24 r. p. m.
23. Rolls, size 36 by 14 inches, belted, speed 95 r. p. m., steel shells.
24. Akins classifier, 54 inches in diameter, speed 5 r. p. m., slope $2\frac{1}{2}$ inches per foot.
25. V tanks (three), height 12 feet, length 30 feet, width 12 feet.
26, 27, 28. Sand tables, speeds 238 strokes per minute.
29, 30. Sand tables, speeds 267 strokes per minute.
31. Dorr thickener, 40 feet in diameter, speed 8.6 revolutions per hour.
32. Flotation plant.

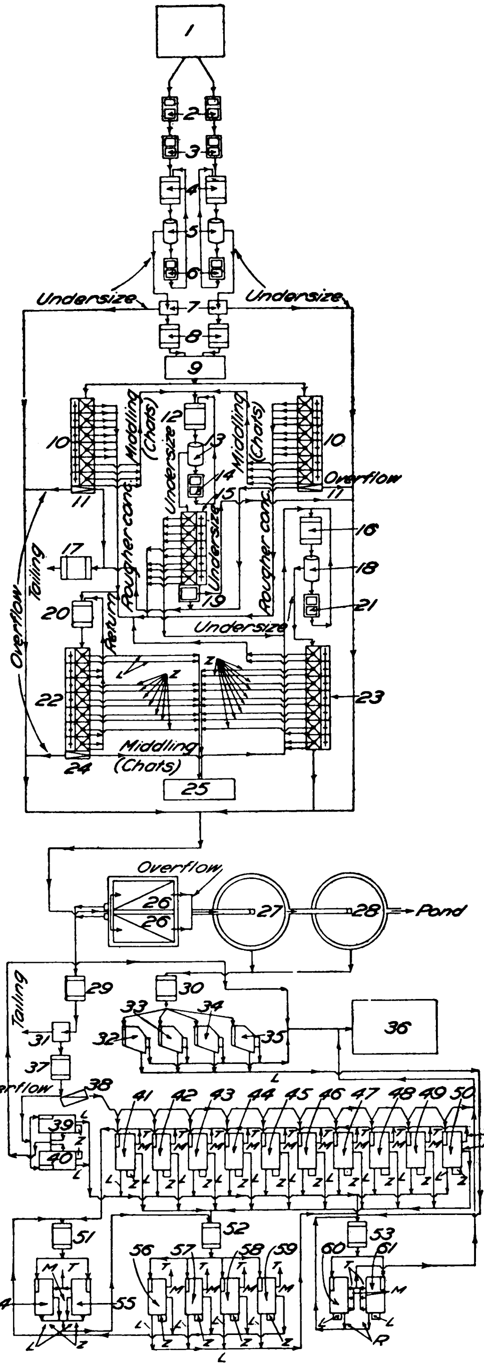


FIGURE 8. Flow sheet of mill in Picher district, Okla. *C*, concentrate; *M*, middling, *T*, tailing; *L*, lead concentrate; *Z*, zinc concentrate; *R*, return. For significance of numbers, see table on opposite page.

Data on parts shown in figure 8.

1. Hopper.
2. Crushers (two), size 18 by 12 inches, speed 354 r. p. m.
3. Rolls (two sets), Cornish type, size 42 by 16 inches, speed 24.8 r. p. m.
4. Elevators (two), size 24 inches, speed 372 feet per minute.
5. Trommels (two), size 48 by 96 inches, ⅜-inch openings, speed 21.2 r. p. m.
6. Rolls (two sets), Cornish type, size 36 by 14 inches, speed 23.6 r. p. m.
7. Dewatering screens (two), 48 by 48 inches, 1½-inch openings, speed 4 r. p. m.
8. Elevators (two), size 18 inches, speed 310 feet per minute.
9. Hopper, 300-ton capacity.
10. Cooley rougher jigs (two), Harz type, with six cells, each 42 by 48 inches in size, speed 91 strokes per minute.
11. Dewatering boxes (two).
12. Elevator, 16-inch, speed 376 feet per minute.
13. Trommel, size 36 by 72 inches, ¼-inch openings, speed 24 r. p. m.
14. Rolls, size 36 by 14 inches, speed 35 r. p. m.
15. Cooley chat jig, Harz type, five cells, each 36 by 42 inches in size, speed 114 strokes per minute.
16. Elevator, 16-inch, speed 384 feet per minute.
17. Elevator, 24-inch, speed 392 feet per minute.
18. Trommel, size 36 by 72 inches, 2-mm. openings, speed 24.6 r. p. m.
19. Dewatering screen, size 48 by 48 inches, 1½-inch openings, speed 4 r. p. m.
20. Elevator, 18-inch, speed 373 feet per minute.
21. Rolls, size 36 by 14 inches, speed 93 r. p. m.
22. Cooley cleaner jig, Harz type, 7 cells, size of each cell 36 by 42 inches, speed 181 strokes per minute.
23. Cooley sand jig, Harz type, 7 cells, each 30 by 36 inches in size, speed 187 strokes per minute.
24. Dewatering box.
25. Concentrate bins.
26. Settling tanks (two), size 30 by 22 feet.
27, *28*. Dorr thickeners (two), 36 feet in diameter, speed 10 r. p. m.
29, *30*. Elevators.
31. Henry screen, 1½-inch openings.
32 to 35. Slime tables (four).
36. Flotation plant.
37. Elevator.
38. Akins classifier, 54 inches in diameter, speed 6 r. p. m., slope 2½ inches per foot.
39, *40*. Slime tables (two).
41 to 50. Sand tables.
51, *52*, *53*. Elevators.
54, *55*. Roughing tables.
56 to 59. Middling tables.
60, *61*. Lead tables.

TABLE 9.—*Results of concentration tests with mills treating sheet-ground ore.*

MILL IN WEBB CITY DISTRICT.

Screen analysis of mill tailings.					Zinc (Zn) content.		
Size of mesh.	Size of openings. Mm.	Size of openings. Inches.	Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
			Per cent.	*Per cent.*	*Per cent.*		
3	6.680	0.263	29.47	....	0.53	20.0	....
6	3.327	.131	28.38	57.85	.68	24.7	44.7
10	1.651	.065	15.48	73.33	.80	15.8	60.5
20	.833	.0328	16.10	89.43	.45	9.3	69.8
35	.417	.0164	5.50	94.93	.42	2.9	72.7
65	.208	.0082	1.37	96.30	.96	1.7	74.4
100	.147	.0058	.63	96.93	1.10	.9	75.3
150	.104	.0041	.40	97.33	1.25	.6	75.9
200	.074	.0029	.32	97.65	1.58	.6	76.5
<200	.074	.0029	2.35	100.00	7.90	23.5	100.0
Total or average			100.00	....	.78	100.0	....
Check sample			....	....	.67	....	....

		Tons.
Ore treated		1,960
Lead and zinc concentrates produced		57.43
Total tailings		1,902.57
Solids in overflow from settling tanks		40.22
Tailings, not including overflow		1,862.35
Zinc concentrates		42.95
Zinc (Zn) in—	Per cent.	
Zinc concentrates	55.0	23.623
Lead concentrates	1.2	.174
Tailings without overflow	.70	12.478
Overflow	7.04	2.831
Total zinc in ore treated		39.106
Percentage of zinc recovery	60.4	

TABLE 9.—*Results of concentration tests with mills treating sheet-ground ore*—Contd.

MILL IN CARTERVILLE DISTRICT.

Screen analysis of mill tailings not including overflow.					Zinc (Zn).		
Size of mesh.	Size of openings. Mm.	Size of openings. Inches.	Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
			Per cent.	*Per cent.*	*Per cent.*		
3	6.680	0.263	33.58	....	0.31	24.3	....
6	3.327	.131	30.75	64.33	.44	31.6	55.9
10	1.651	.065	16.84	81.17	.44	17.3	73.2
20	.833	.0328	8.78	89.95	.31	6.35	79.55
35	.417	.0164	4.60	94.55	.31	3.3	82.85
65	.208	.0082	2.57	97.12	.55	3.3	86.15
100	.147	.0058	.98	98.10	.57	1.3	87.45
150	.104	.0041	.80	98.90	1.04	1.95	89.40
200	.074	.0029	.21	99.11	1.56	.8	90.2
<200	.074	.0029	.89	100.00	4.68	9.8	100.00
Total or average			100.00	....	.43	100.00	....
Check sample			....	....	.49	....	....

		Tons.
Ore treated		3,456
Concentrates produced (lead and zinc)		65.53
Total tailings from mill		3,390.47
Solids in overflow from settling tanks		52.66
Tailings, not including overflow		3,337.81
Zinc concentrates		62.765
Lead concentrates		2.765
Zinc (Zn) in—	Per cent.	
Zinc concentrates	61.88	38.839
Lead concentrates	15.00	.415
Mill tailings without overflow	.49	16.355
Solids from overflow	6.38	3.360
Total zinc in ore treated		58.969
Percentage of zinc recovery	65.86	

TABLE 9.—*Results of concentration tests with mills treating sheet-ground ore*—Contd.

MILL IN PROSPERITY DISTRICT.

Screen analysis of mill tailings not including overflow.					Zinc (Zn).		
Size of mesh.	Size of openings. Mm.	Size of openings. Inches.	Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
			Per cent.	*Per cent.*	*Per cent.*		
3	6.680	0.263	9.90	....	0.53	12.3	....
6	3.327	.131	38.82	48.72	.37	33.8	46.1
10	1.651	.065	23.76	72.48	.29	16.2	62.3
20	.833	.0328	11.43	83.91	.19	5.1	67.4
35	.417	.0164	6.24	90.15	.25	3.7	71.1
65	.208	.0082	4.12	94.27	.26	2.55	73.65
100	.147	.0058	1.94	96.21	.42	1.9	75.55
150	.104	.0041	1.45	97.66	.96	3.3	78.85
200	.074	.0029	.19	97.85	1.16	.5	79.35
<200	.074	.0029	2.15	100.00	4.07	20.65	100.00
Total or average			100.00	....	.42	100.00	....
Check sample			....	....	.45	....	....

		Tons.
Ore treated		470
Concentrates produced (zinc concentrates)		8.545
Tailings		461.455
Solids in overflow from settlings		12.457
Tailings, not including overflow		448.998
Zinc (Zn) in—	Per cent.	
Concentrates	60.18	5.142
Tailings without overflow	.45	2.020
Overflow	5.25	.654
Total zinc in ore treated		7.816
Percentage of zinc recovery	65.79	

TABLE 9.—*Results of concentration tests with mills treating sheet-ground ore*—Contd.

MILL IN PORTO RICO DISTRICT.

Screen analysis of mill tailings not including overflow.					Zinc (Zn) content.		
Size of mesh.	Size of openings. Mm.	Size of openings. Inches.	Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
			Per cent.	*Per cent.*	*Per cent.*		
	13.33	0.525	6.89	....	0.30	5.6	....
3	6.680	.263	32.08	38.97	.22	19.3	24.9
6	3.327	.131	26.75	65.72	.20	14.6	39.5
10	1.651	.065	18.40	84.12	.19	9.5	49.0
20	.833	.0328	9.07	93.19	.30	7.4	56.4
35	.417	.0164	2.70	95.89	.56	4.1	60.5
65	.208	.0082	1.80	97.69	1.27	6.3	66.8
100	.147	.0058	.44	98.13	2.93	3.5	70.3
150	.104	.0041	.39	98.52	3.35	3.6	73.9
200	.074	.0029	.21	98.73	5.70	3.3	77.2
<200	.074	.0029	1.27	100.00	6.55	22.8	100.0
Total or average			100.00	....	.37	100.0	....
Check sample			....	....	.41	....	....

	Per cent.	Tons.
Ore treated		1,994
Concentrates produced		35.5
Tailings		1,958.50
Solids in overflow from settling tanks		34.73
Tailings, not including overflow		1,923.77
Zinc (Zn) in—		
Concentrates	59.4	21.087
Tailings without overflow	.41	7.887
Overflow	5.01	1.740
Total zinc in ore treated		30.714
Percentage of zinc recovery	68.6	

TABLE 9.—*Results of concentration tests with mills treating sheet-ground ore*—Contd.

MILL IN DUENWEG DISTRICT.

Screen analysis of mill tailings not including overflow.					Zinc (Zn) content.		
Size of mesh.	Size of openings. Mm.	Size of openings. Inches.	Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
			Per cent.	*Per cent.*	*Per cent.*		
	13.33	0.525	18.50	....	0.60	21.05	....
3	6.680	.263	35.93	54.43	.46	31.35	52.40
6	3.327	.131	19.48	73.91	.42	15.5	67.90
10	1.651	.065	12.59	86.50*	.36	8.6	76.5
20	.833	.0328	7.29	93.79	.38	5.3	81.8
35	.417	.0164	3.17	96.96	.36	2.15	83.95
65	.208	.0082	.65	97.61	.74	.9	84.85
100	.147	.0058	.54	98.15	.95	.95	85.80
150	.104	.0041	.50	98.65	.84	.8	86.60
200	.074	.0029	.30	98.95	1.41	.8	87.4
<200	.074	.0029	1.05	100.00	6.30	12.6	100.00
Total or average			100.00	....	.53	100.00	....
Check sample			....	....	.60	....	....

		Tons.
Ore treated		938
Concentrates produced (lead and zinc)		20.53
Tailings		917.47
Solids in overflow from settling tanks		42.52
Tailings not including overflow		874.95
Zinc concentrates		17.03
Lead concentrates		3.50
Zinc (Zn) in—	Per cent.	
Zinc concentrates	61.47	10.468
Tailings without overflow	.60	5.250
Overflow	4.21	1.790
Lead concentrates	1.00	0.035
Total zinc in ore treated		17.543
Percentage of zinc recovery	59.67	

TABLE 10. *Results of concentration test with mills treating hard-ground ore other than sheet-ground.*

MILL IN CARL JUNCTION DISTRICT.

Screen analysis of mill tailings not including overflow.					Zinc (Zn) content.		
Mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
			Per cent.	*Per cent.*	*Per cent.*		
....	13.33	0.525	9.20	....	0.24	3.15	....
3	6.680	.263	31.97	41.17	.64	29.3	32.45
6	3.327	.131	24.01	65.18	.47	16.15	48.60
10	1.651	.065	15.99	81.17	.38	8.7	57.30
20	.833	.0328	8.90	90.07	.42	5.35	62.65
35	.417	.0164	4.70	94.77	.77	5.2	67.85
65	.208	.0082	1.62	96.39	1.55	3.6	71.45
100	.147	.0058	.89	97.28	1.72	2.2	73.65
150	.104	.0041	.39	97.67	2.27	1.3	74.95
200	.074	.0029	.22	97.89	3.69	1.15	76.10
<200	.074	.0029	2.11	100.00	7.90	23.90	100.0
Total or average..............			100.00	....	.70	100.00	....
Check sample..............			....	....	.76	....	....

		Tons.
Ore treated..		1,975
Zinc concentrates produced..		67.45
Tailings..		1,907.55
Solids in overflow from settling tanks..		65.09
Tailings without overflow..		1,842.46
Zinc (Zn) in—	Per cent.	
Concentrates..	60.1	40.537
Tailings without overflow..	.76	14.003
Overflow..	6.95	4.524
Total zinc in ore treated..		59.064
Percentage of zinc recovery..	68.63	

TABLE 10.—*Results of concentration test with mills treating hard-ground ore other than sheet-ground*—Continued.

MILL IN COMMERCE, OKLA., DISTRICT.

Screen analysis of mill tailings, not including overflow.					Zinc (Zn) content.		
Size of mesh.	Size of openings. Mm.	Size of openings. Inches.	Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
			Per cent.	*Per cent.*	*Per cent.*		
3	6.680	0.263	1.12	....	0.60	0.8	....
6	3.327	.131	37.60	38.72	.70	30.4	31.2
10	1.651	.065	29.21	67.93	.80	26.95	58.15
20	.833	.0328	10.27	78.20	.90	10.65	68.80
35	.417	.0164	5.10	83.30	1.15	6.8	75.60
65	.208	.0082	3.42	86.72	1.13	4.45	80.05
100	.147	.0058	2.72	89.44	.75	2.35	82.40
150	.104	.0041	2.38	91.82	.75	2.1	84.50
200	.074	.0029	1.54	93.36	.95	1.7	86.20
<200	.074	.0029	6.64	100.00	1.80	13.8	100.00
Total or average			100.00	....	0.86	100.00	....
Check sample			....	....	0.85	....	....

		Tons.
Ore treated		1,504
Concentrates produced		100.46
Tailings		1,403.54
Solids in overflow from settling tanks		122.20
Mill tailings without overflow		1,281.34
Zinc concentrates		39.65
Lead concentrates		60.81
Zinc (Zn) in—	Per cent.	
Zinc concentrates	47.03	18.647
Lead concentrates	2.09	1.271
Tailings without overflow	.85	10.891
Solids from overflow	1.13	1.381
Total zinc in ore treated		32.190
Percentage of zinc recovery	57.93	
Lead (Pb) in—		
Lead concentrates	80.93	49.214
Zinc concentrates	2.44	0.967
Tailings without overflow	0.20	2.563
Overflow	0.14	0.171
Total lead in ore treated		52.915
Percentage of lead recovery	93.06	
Quantity of lead and zinc (metallic) in ore treated		85.105
Quantity of available lead and zinc in concentrates		67.861
Percentage of lead and zinc recovery combined	79.74	

TABLE 10.—*Results of concentration test with mills treating hard-ground ore other than sheet-ground*—Continued.

MILL IN COMMERCE, OKLA., DISTRICT.

Screen analysis of mill tailings, not including overflow.					Zinc (Zn) content.		
Size of mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
			Per cent.	*Per cent.*	*Per cent.*		
3	6.680	0.263	4.99	....	0.68	3.0	....
6	3.327	.131	29.72	34.71	1.08	28.45	31.45
10	.651	.065	22.98	57.69	1.12	22.85	54.30
20	.833	.0328	14.57	72.26	.85	11.0	65.30
35	.417	.0164	12.50	84.76	.87	9.65	74.95
65	.208	.0082	2.02	86.78	.79	1.4	76.35
100	.147	.0058	1.01	87.79	.79	.7	77.05
150	.104	.0041	.70	88.49	.97	.6	77.65
200	.074	.0029	.50	88.99	1.13	.5	78.15
<200	.074	.0029	11.01	100.00	2.24	21.85	100.00
Total or average			100.00	....	1.13	100.00	....
Check sample			....	....	1.1	....	....

	Per cent.	Tons.
Ore treated		1,498
Zinc concentrates		51.05
Lead concentrates		45.50
Concentrates produced (lead and zinc)		96.55
Tailings		1,401.45
Solids in overflow from settling tanks		84.13
Tailings without overflow		1,317.32
Zinc (Zn) in—		
Zinc concentrates	42.80	21.849
Lead concentrates	2.50	1.137
Tailings without overflow	1.10	14.490
Overflow	3.28	2.759
Total zinc in ore treated		40.235
Percentage of zinc recovery	54.33	
Lead (Pb) in—		
Lead concentrates	76.0	34.580
Zinc concentrates	2.7	1.378
Tailings without overflow	.2	2.635
Overflow	1.92	1.615
Total lead in ore treated		40.208
Percentage of lead recovery	86.00	
Quantity of lead and zinc in ore treated		80.443
Quantity of lead and zinc available in concentrates		56.429
Percentage of lead and zinc recovery	70.15	

TABLE 11.—*Results of concentration test with mill treating soft-ground ore.*

Screen analysis of mill tailings, including overflow.					Zinc (Zn) content.		
Mesh.	Size of opening.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
			Per cent.	*Per cent.*	*Per cent.*		
3	6.680	0.263	16.23	....	0.43	4.9	....
6	3.327	.131	30.20	46.43	.73	15.5	20.4
10	1.651	.065	17.50	63.93	.89	10.95	31.35
20	.833	.0328	11.03	74.96	1.34	10.4	41.75
35	.417	.0164	6.81	81.77	1.82	8.7	50.45
65	.208	.0082	1.86	83.63	2.90	3.8	54.25
100	.147	.0058	1.22	84.85	3.07	2.65	56.90
150	.104	.0041	.88	85.73	3.59	2.2	59.10
200	.074	.0029	.68	86.41	3.70	1.75	60.85
200	.074	.0029	13.59	100.00	4.10	39.15	100.00
Total or average			100.00	....	1.42	100.00	....
Check sample			....	....	1.55	....	....

		Tons.
Ore treated		642
Concentrates (zinc) produced		40.32
Tailings		601.68
Zinc (Zn) in—	Per cent.	
Concentrates	58.60	23.627
Tailings	1.55	9.326
Total zinc (metallic) in ore treated		32.953
Percentage of zinc recovery	71.70	

IMPORTANCE OF MILL TESTS.

When an ore is to be mined and treated, it is not always apparent, even to the most experienced metallurgist or millman, what method of procedure will give the highest recovery of the valuable minerals. After a concentrating plant or mill has been constructed and put in operation the mill superintendent or millman is apt to find that certain changes in construction are necessary to effect the best saving. However, tests should be made to determihe what kind of work the mill, or a certain part of the concentration equipment, is doing, both before and after any changes are made. The millman in charge, however, should not discontinue the practice of making tests when he believes the most profitable recovery with the ore in question is being obtained, but should continue to make tests at frequent intervals throughout the life of the plant, because of the variations in the richness and character of the ore and in the quantity treated. The millman is obliged to consider what degree of recovery will yield the greatest profit; therefore the best technical results do not correspond to the best commercial results. Frequent testing, however, will indicate where improvements can be made for increasing the saving of mineral; also the point of highest profits relative to the highest degree of efficiency practicable is brought out more definitely.

The concentration methods used in the Joplin district may seem crude to outsiders, but local practices adapted to local conditions should not be condemned without careful consideration. The penalties the smelters impose upon impure products and the conditions governing local prices must be taken into account. From these conditions and the usually low mineral content of the ore mined, it is obvious that the operator is guided more by the relative market values of his product than by the increase in saving that he might be able to effect through more elaborate methods or equipment.

VALUE OF SCREEN ANALYSES.

Except at a few of the mills in the district, little attempt is made at systematic mill tests to determine the percentage of recovery made in the mill or in the different steps of the milling process. This in spite of the fact that sampling tests should be made at every mill to aid the millman in detecting and rectifying unnecessary losses.

Screen analyses of the tailings should be made to determine in what sizes the mineral losses are greatest, so that a more definite idea can be obtained as to where further saving is possible. It may be found that the material coarser than 20-mesh does not contain mineral that would repay recovery, whereas that passing through a 20-mesh screen does. Consequently minor changes could be made to collect the material finer than 20-mesh for further treatment, or to improve that part of the mill that is losing the greater proportion of the ore particles finer than 20-mesh. If such screen analyses are made and each screen product is assayed, the results will indicate the metal content of the different sizes, and from the weights the percentage of loss in these sizes can be calculated.

In any screen analysis, the millman wishes to know what values are contained in the oversize or undersize of a screen used in the mill and having a certain size of screen openings. The selection of a set of screens that would be suited to every mill and every purpose would be difficult, if not impossible, as the perforated screens used are of various size openings, according to the local practice or the practice at any one mill. For making a screen test the millman can use whatever size of screen openings that, in his judgment, is best suited to the test he wishes to make. However, if he should wish to make an analysis of several sizes, especially of those intermediate of the sizes of screens used in the mill, the different size openings of the screens used should have a definite relation to each other. Many different makes of screens have the same mesh, but owing to the different diameters of the wire used by the different manufacturers, the openings are not the same. The mesh, which is the number of openings per linear inch, means nothing if the size or diameter of the opening is not given.

Also, screen analyses should be made to determine the crushing efficiency of rolls and the screening efficiency of screens.

For all screen tests made, as given herein, a set of Tyler standard-screen-scale sieves was used, the openings of which are as follows:

TABLE 12.—*Sizes of Tyler standard-screen-scale sieves.*

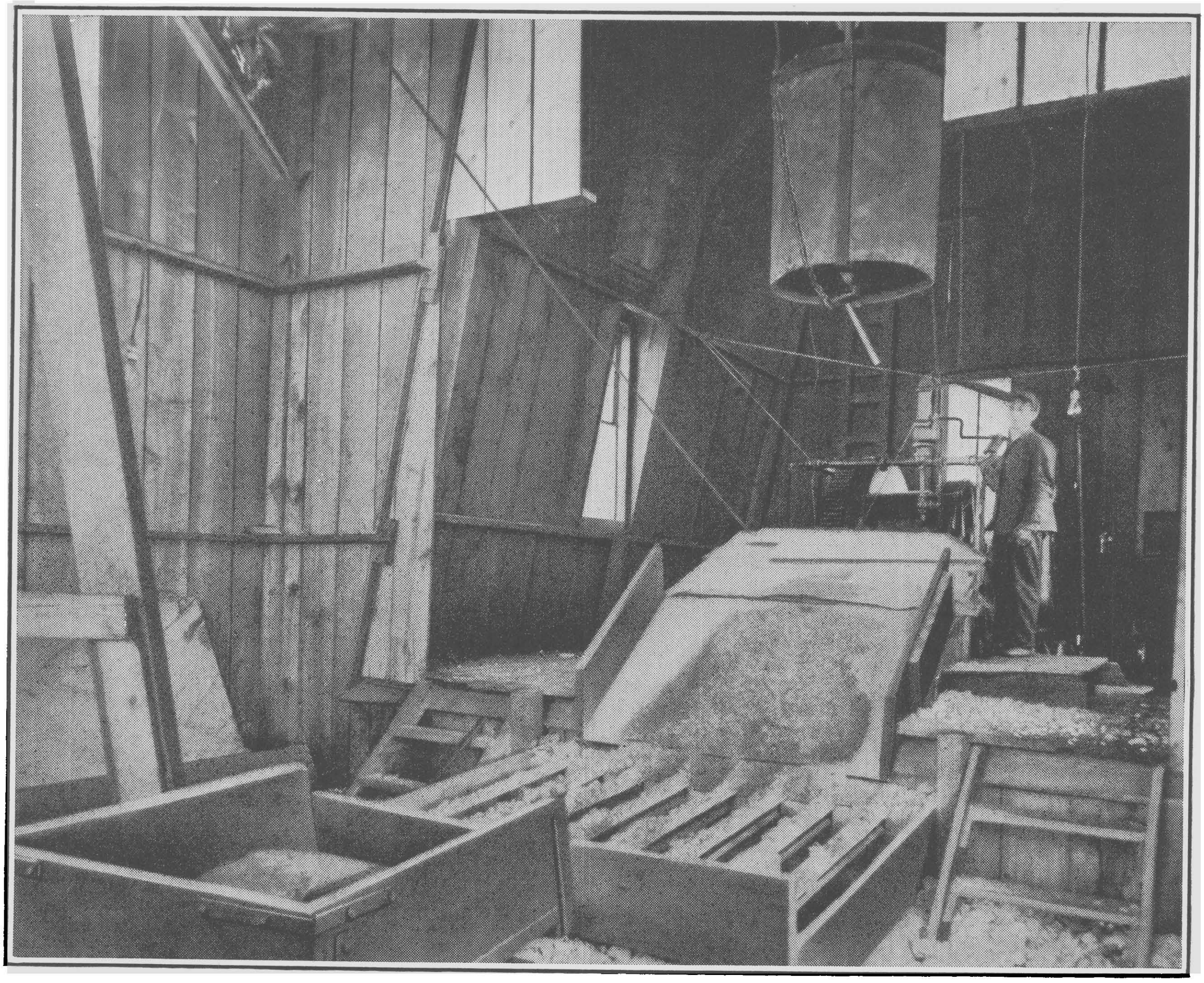
Size of openings.		Mesh.	Diameter of wire.
Inches (ratio, $\sqrt{2}$ or 1.414).	Milli-meters.		
			Inch.
1.050	26.67	....	0.149
.742	18.85	....	.135
.525	13.33	....	.105
.371	9.423	....	.092
.263	6.680	3	.070
.185	4.699	4	.065
.131	3.327	6	.036
.093	2.362	8	.032
.065	1.651	10	.035
.046	1.168	14	.025
.0328	.833	20	.0172
.0232	.589	28	.0125
.0164	.417	35	.0122
.0116	.295	48	.0092
.0082	.208	65	.0072
.0058	.147	100	.0042
.0041	.104	150	.0026
.0029	.074	200	.0021

COARSE CONCENTRATION.

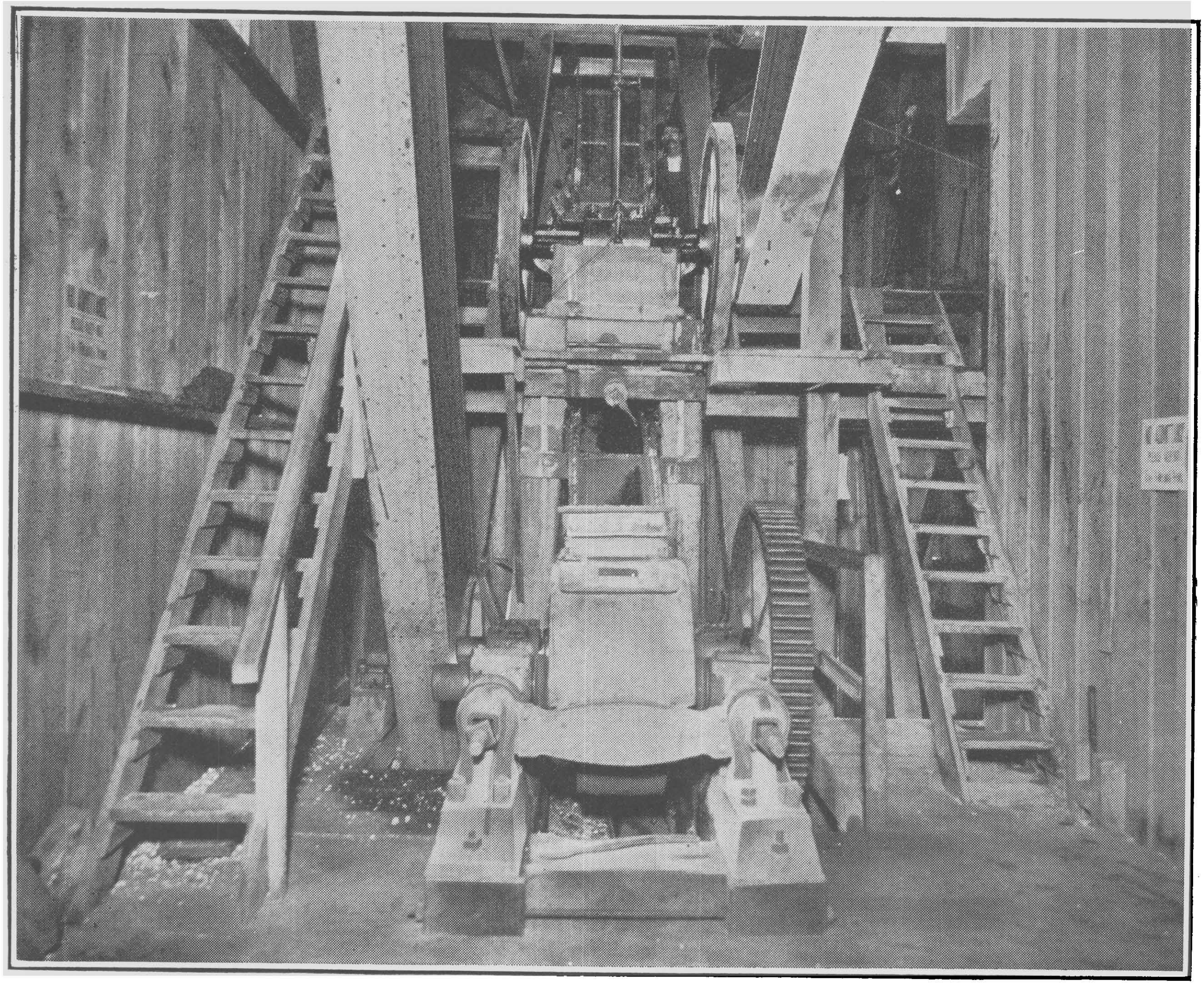
SORTING.

When the ore has been hoisted from the shaft it is delivered on to a chute leading to a grizzly. This grizzly consists of iron bars, usually heavy rails, spaced 4 to 5 inches apart as shown in Plate XIII, *A*. The mineralized lumps in the oversize caught on the grizzly are broken with sledge hammers into pieces small enough to pass through; the bowlders that are practically barren are sorted by hand and trammed to the waste-rock pile. The proportion of waste rock thus sorted out varies from a fraction of 1 per cent to 15 per cent, according to the character of the ore. The sorting is done by screen-men, better known locally as "cull men."

Sorting other than that by the screenmen is rarely practiced at the mines of the Joplin district. It is believed that sorting out the greater proportion of the large bowlders of waste rock from the ore is of much importance if properly done. The reasons for this belief are perfectly clear. Eliminating a certain amount of waste rock increases the richness of the material retained for further treatment



A. METHOD OF DUMPING TUB ON GRIZZLY ABOVE HOPPER.



B. BLAKE CRUSHER, MOUNTED ABOVE PRIMARY ROLL

and also the relative capacity of the mill; makes possible a higher recovery; lessens the wear and tear on rolls, elevators, screens, and other equipment, and saves the power that would be consumed in treating waste rock. At one of the sheet-ground mines hand-picking the ore just before it entered the crusher was tried to good advantage, and a considerable saving in both mineral and costs per ton was said to have been made. If a system of sorting underground, especially in the sheet-ground mines, could be developed, or even a large proportion of waste rock sorted out by the screenmen at the grizzly, the writer believes that an appreciable saving would be effected.

CRUSHING AND SCREENING.

The undersize material, 5 inches or less in diameter, passes through the openings of the grizzly into the mill hopper below. From the hopper the ore is fed by means of an incline chute into a jaw crusher of the Blake type (Pl. XIII, *B*), where the larger size material is reduced to 2 inches or less in diameter, depending on the setting of the jaws of the crusher at the discharge end.

USE OF SHAKING SCREENS AHEAD OF CRUSHER.

At a few mills it was found that instead of an incline a perforated shaking screen having about $\frac{3}{8}$ to $\frac{3}{4}$ inch holes was used, a stream of water being played upon the material as it passed over the screen. Much of the finer material was washed through, thus eliminating unnecessary crushing of fines and increasing to some extent the relative capacity of the crusher and the rolls, the undersize being fed directly to the feed or "dirt" elevator. This feature is a useful one and has been effectively applied in some of the larger mills in other mining districts.

In the few mills in this district where this device has been tested there has been some difficulty in getting the undersize through the openings of the screen, chiefly because the narrow width of the screen causes the ore to bed too heavily and because a good shaking motion is lacking. In those districts where shaking screens are used at the head of the stamp or the crusher to by-pass a certain undersize, the screen is 4 to 5 feet wide at the feed end and tapers to 1 to $1\frac{1}{2}$ feet in width at the discharge end. The shaking motion is effected by an eccentric. In the mills of the Joplin district, where such screens are used, the writer believes that the screening efficiency could be increased by using wider screens, thus permitting the ore to spread into a thinner bed, and by effecting a better shaking motion. A simpler device that might be used to by-pass the undersize into the feed elevator is an inclined grizzly with the bars set about three-fourths of an inch apart.

SCREEN TEST OF PRODUCT FROM SHAKING SCREEN.

To give some idea of the product from a shaking screen with ¾-inch perforations, as used at the head of the crusher in one of the mills, the following screen analysis is given:

TABLE 13.—*Results of screen analysis of undersize from shaking screen used ahead of crusher.*

Size of screen openings (mesh).	Weight of screen product.	Cumulative weight.	Size of screen openings (mesh).	Weight of screen product.	Cumulative weight.
	Per cent.	*Per cent.*		*Per cent.*	*Per cent.*
[a] 0.525	4.96	....	100	.52	98.32
3	43.21	48.17	150	.52	98.84
6	27.25	75.42	200	.62	99.46
10	11.18	86.60	<200	.54	100.00
20	5.37	91.97			
35	3.79	95.76	Total......	100.00	....
65	2.04	97.80			

[a] Inch.

The table shows that 75 per cent of the undersize product from the shaking screen was oversize on the 6-mesh screen, and that only a small percentage of material larger than 0.525 inch, or approximately ½ inch, passed through the ¾-inch round openings in the shaking screen.

CRUSHERS, ROLLS, AND TROMMEL CIRCUIT.

As stated previously, the crushers used in the mills of this district are of the Blake type, and vary in size from 8 by 14 inches to 12 by 24 inches. More than one crusher is seldom used with the exception that if the mill is divided into two sections or units, as in a few of the larger plants, two crushers are employed. The usual running speed of crushers is 300 to 400 revolutions per minute.

The discharge from the crusher is fed directly to a set of rolls (see Pl. XIII, *B*) and is largely reduced to ¾-inch size or less. As a rule only one set of rolls, better known as "first" rolls or primary rolls, is used for crushing to this size, but a few of the larger mills use two sets. Primary rolls generally measure 36 by 14 inches, although rolls measuring 30 by 14 and 42 by 14 inches are often used.

The roll shells are cast locally and consist of ordinary hard iron. Some of the mills are now using steel shells for recrushing middlings with satisfactory results. Closed rolls (faces touching) are generally used, although a few mills are equipped with spaced rolls which are run at a higher speed than the closed type. The advantage of spaced rolls is the production of less slimes.

The crushed ore from the primary rolls is elevated to a trommel, usually with ½-inch round-hole perforations; the undersize from the trommel goes to the roughing jigs and the oversize returns to one or two secondary rolls, or return rolls, for further crushing. The product from the secondary rolls is fed back to the feed or "dirt" elevator and is returned to the trommel again. Thus the recrushing equipment forms a closed circuit so that all the material from the primary rolls must pass through the ½-inch openings of the trommel.

SCREEN TESTS OF PRODUCTS.

Some screen tests of the products from crushers, primary rolls, and return rolls are shown in Table 14 following.

TABLE 14.—*Results of screen tests of feed and discharge of primary rolls in mill treating "sheet-ground" ore.*

FIRST TEST.

Size of screen openings (mesh).	Feed to primary rolls (crusher discharge). Weight of screen products.	Feed to primary rolls (crusher discharge). Cumulative weight.	Discharge from primary rolls. Weight of screen products.	Discharge from primary rolls. Cumulative weight.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
[a]0.5	73.83	....	32.78	....
3	8.78	82.61	30.08	62.86
6	8.08	90.69	17.90	80.76
10	4.48	95.17	8.65	89.41
20	1.97	97.14	4.12	93.53
35	1.15	98.29	2.86	96.39
65	.68	98.97	1.77	98.16
100	.26	99.23	.56	98.72
150	.23	99.46	.47	99.19
200	.10	99.56	.21	99.40
<200	.44	100.00	.60	100.00
	100.00		100 00	

SECOND TEST.

[a]0.5	78.69	....	38.26	....
3	6.13	84.82	27.13	65.39
6	6.55	91.37	13.44	78.83
10	3.71	95.08	6.93	85.76
20	1.79	96.87	4.42	90.18
35	1.19	98.06	4.35	94.73
65	.75	98.81	2.79	97.52
100	.31	99.12	.86	98.38
150	.27	99.39	.62	99.00
200	.15	99.54	.28	99.28
<200	.46	100.00	.70	99.98
	100.00		99.98	

[a] Inch.

The table shows that more than 70 per cent of the material discharged from the crusher would remain on a ½-inch mesh screen, and that by passing this material through the first rolls the proportion of material coarser than ½ inch is reduced about one half. The other screen products less than ½ inch in size indicate how that part of the crusher discharge larger than ½-inch size, is distributed through the different sizes below ½ inch.

The oversize from the trommels handling the material from the feed or "dirt" elevator is crushed by one or two sets of return rolls, usually 30-inch. As previously mentioned, the crushing is in closed circuit, all the material being crushed to pass the openings of the trommel before going to the jigs.

The results of screen analyses of the feeds to return rolls and their products are given in Table 15, followiug.

TABLE 15.—*Results of screen analyses of feed and product of return rolls in mill treating "hard-ground" ore.*

Screen.			First series of tests.				Second series of tests.			
			Feed to return rolls.		Product from return rolls.		Feed to return rolls.		Product from return rolls.	
Size of mesh.	Size of opening.		Weight of screen products.	Cumulative weight	Weight of screen products.	Cumulative weight.	Weight of screen products.	Cumulative weight.	Weight of screen products.	Cumulative weight.
	Mm.	Inches.								
			Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
....	....	0.525	31.39	....	3.00	....	30.82	....	2.15	....
3	6.680	.263	64.23	95.62	43.53	46.53	59.23	90.05	44.06	46.21
6	3.327	.131	3.75	99.37	26.70	73.23	8.03	98.08	29.56	75.77
10	1.651	.065	.27	99.64	11.94	85.17	.79	98.87	10.59	86.36
20	.833	.0328	.08	99.72	5.94	91.11	.36	99.23	5.38	91.74
35	.417	.0164	.08	99.80	3.55	94.66	.28	99.51	3.49	95.23
65	.208	.0082	.20	100.00	1.93	96.59	.20	99.71	1.81	97.04
100	.147	.0058	....	....	.88	97.47	.11	99.82	.81	97.85
150	.104	.0041	....	....	.19	97.66	.04	99.86	.13	97.98
200	.074	.0029	....	....	.65	98.31	.04	99.90	.70	98.68
<200	.074	.0029	....	....	1.69	100.00	.10	100.00	1.32	100.00
Total................			100.00	....	100.00	....	100.00	....	100.00	....

In the two examples of screen tests given above, the rolls were doing good work, better than the average done in the mills throughout the district. However, the relatively more efficient crushing is partly the result of efficient screening by the screen trommels.

CRUSHING OF "CHATS."

The next step in crushing for coarse concentration is to crush the middlings from the different jigs. This material, which needs further grinding and concentration, is locally known as "chats," the set of rolls crushing it is known as "chat" rolls, and the jigs that treat the crushed product separately are called "chat" jigs.

It is in crushing chats that the least effective crushing is done in the mills of the district. As an example of rather poor results in crushing chats, screen analyses of the feed and the product from a set of 24-inch rolls are shown in Table 16 herewith.

TABLE 16.—*Results of screen analyses of feed and product of "chat" rolls in mill treating "hard-ground" ore.*

Sieve.			Feed to chat rolls.		Product from chat rolls.	
Size of mesh.	Size of openings. Mm.	Size of openings. Inches.	Weight of screen products.	Cumulative weight.	Weight of screen products.	Cumulative weight.
			Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
3	6.680	0.263	16.16	....	10.48	....
6	3.327	.131	69.02	85.18	68.98	79.46
10	1.651	.065	11.96	97.14	16.10	95.56
20	.833	.0328	1.86	99.00	1.91	97.47
35	.417	.0164	.78	99.78	1.49	98.96
65	.208	.0082	.11	99.89	.44	99.40
100	.147	.0058	.04	99.93	.15	99.55
150	.104	.0041	.01	99.94	.07	99.62
200	.074	.0029	.02	99.96	.07	99.69
<200	.074	.0029	.04	100.00	.31	100.00
Total........................			100.00	....	100.00	....

Uneven wear of the shells, of the usual cast-iron type used so extensively throughout the district, is responsible for the seemingly poor work of these rolls. For material of this size steel shells are much more efficient and should be used. Returning so much oversize to the trommel reduces its screening efficiency.

COARSE SCREENING.

Cylindrical trommels are the screens most commonly used throughout the district for coarse work. Both trommel and shaking screens are used for fine screening although the former finds more favor among the mill men.

The screening efficiency of a trommel depends on the size of the trommel, both its diameter and length, its speed, its slope, the size of screen opening, and the quantity and rate of feed. In the mills of the Joplin district the trommels are run at speeds ranging from 15 to 28 revolutions per minute, and have slopes of five-eighths of an inch to 2 inches per foot. For coarse screening the trommels usually are 36 by 96 inches, 48 by 96 inches, and 48 by 144 inches, with three-eighths to five-eighths inch openings at the head of "rougher" jigs and one-fourth to one-eighth inch openings at the head of sand or "chat" jigs.

It has been stated that the discharge from the feed or "dirt" elevator is fed to a trommel; the undersize from the trommel passes to the rougher jigs, while the oversize is crushed through the "return" rolls and returned to the feed elevator. Where the entire feed passes eventually through the openings of the trommel, the trommel being in closed circuit with the return rolls, the millman is likely to be less careful in determining the best conditions for most efficient and economical screening and capacity, as long as the main purpose of the screen is accomplished. The writer has frequently found trommels running too fast for efficient screening. Increasing the speed beyond certain limits, lessens the screening efficiency, increases the quantity of oversize, adds to the wear on the rolls from grinding additional oversize, and the undersize, which should have passed through the screen openings, is reduced still finer by the rolls in closed circuit.

In the Joplin district the ore is screened wet. If the trommel has not sufficient slope with reference to the speed and the diameter, the feed is liable to be too heavy and prevent the undersize from working freely through the screen. This is especially true of the finer sized material when not enough water is used. The mill man should determine as nearly as possible the slope and speed best suited to the trommel for the given quantity of feed and size of material.

EFFICIENCY OF TROMMELS.

To show the screening efficiency of trommels in some of the Joplin mills the results of a few tests are given below.

TEST 1.—*Screening efficiency of trommel with ½-inch openings.*

[Trommel size, 48 by 144 inches; speed, 25 r. p. m.; slope, 1½ inches per foot.]

$$\text{Efficiency}=\frac{\text{Weight of undersize passing through the screen.}}{\text{Total weight of undersize in the feed.}}$$

Product.	Trommel feed.	Trommel oversize.
Oversize on ½-inch screen, per cent	17. 2	44. 7
Undersize through ½-inch screen, per cent	82. 8	55. 3
	100. 0	100. 0

$$\frac{17.2\times 55.3}{44.7}=21.28;\ \text{efficiency}=\frac{(82.8-21.28)\times 100}{82.8}=74.06 \text{ per cent.}$$

TEST 2.—*Screening efficiency of trommel with ½-inch openings.*

[Trommel size, 48 by 144 inches; speed, 25 r. p. m.; slope, 1½ inches per foot.]

Product.	Trommel feed.	Trommel oversize.
Oversize on ½-inch screen, per cent	21. 4	42. 1
Undersize on ½-inch screen, per cent	78. 6	57. 9
	100. 0	100. 0

$$\frac{21.4\times 57.9}{42.1}=29.43;\ \text{efficiency}=\frac{(78.6-29.43)\times 100}{78.6}=62.55 \text{ per cent.}$$

TEST 3.—*Screening efficiency of trommel with $\frac{7}{16}$-inch openings.*

[Trommel size, 48 by 144 inches; speed, 21 r. p. m.]

Product.	Trommel feed.	Trommel oversize.
Oversize on $\frac{7}{16}$-inch screen, per cent	23. 1	64. 5
Undersize through $\frac{7}{16}$-inch screen, per cent	76. 9	35. 5
	100. 0	100. 0

$$\frac{23.1\times 35.5}{64.5}=12.71;\ \text{efficiency}=\frac{(76.9-12.71)\times 100}{76.9}=83.47 \text{ per cent.}$$

TEST 4.—*Screening efficiency of trommel with ½-inch openings.*

[Trommel size, 48 by 144 inches; speed, 21 r. p. m.; slope, 1½ inches per foot.]

Product.	Trommel feed.	Trommel oversize.
Oversize on ½-inch screen, per cent	13.3	40.3
Undersize through ½-inch screen, per cent	86.7	59.7
	100.0	100.0

$\frac{13.3\times59.7}{40.3}=19.7$; efficiency$=\frac{(86.7-19.7)\times100}{86.7}=77.28$ per cent.

TEST 5.—*Screening efficiency of trommel with ¼-inch openings.*

[Trommel size, 48 by 96 inches; speed, 26 r. p. m.; slope, 1½ inches per foot.]

Product.	Trommel feed.	Trommel oversize.
Oversize on ¼-inch screen, per cent	15.3	29.7
Undersize through ¼-inch screen, per cent	84.7	70.3
	100.0	100.0

$\frac{15.3\times70.3}{29.7}=36.22$; efficiency$=\frac{(84.7-36.22)\times100}{84.7}=57.24$ per cent.

In most mills the poor screening efficiency of trommels results from too high speed. Usually the best speed for a trommel, under proper conditions, is 15 to 21 revolutions per minute, and the trommel should not, as a rule, be run much faster or slower than these limits.

JIGGING.

The Cooley jig (Pl. XIV and fig. 9), similar in principle to the Harz jig, is the most common type of jig used in this district. It is of the fixed-sieve type, the water being forced up and down through the stationary screens or grates by the action of plungers in adjacent compartments. The number and the size of the compartments for each jig depend on the size and character of the ore treated.

OUTLINE OF CONCENTRATION SYSTEM.

In general, a system of "roughing" and "cleaning" is followed. The feed is given a preliminary cleaning on one or two "rougher" jigs that eliminates the greater proportion of waste material. Then the enriched product, which will assay 10 to 25 per cent zinc, is treated on a "cleaner" jig, bringing the zinc tenor up to 50 to 60 per cent. The "chats" or middlings from the rougher and the cleaner jigs, which together will assay 4 to 8 per cent zinc, are recrushed and either returned to the rougher-jig feed or treated separately over a "chat" jig. The tailings from the rougher jig are usually dewatered by means of a dewatering trommel with 1½-mm. to 2-mm. openings, over the outside of which the tailings pass as the trommel slowly revolves. The undersize from this trommel flows to the settling tanks and the oversize to the tailings elevator as waste. The overflow from the

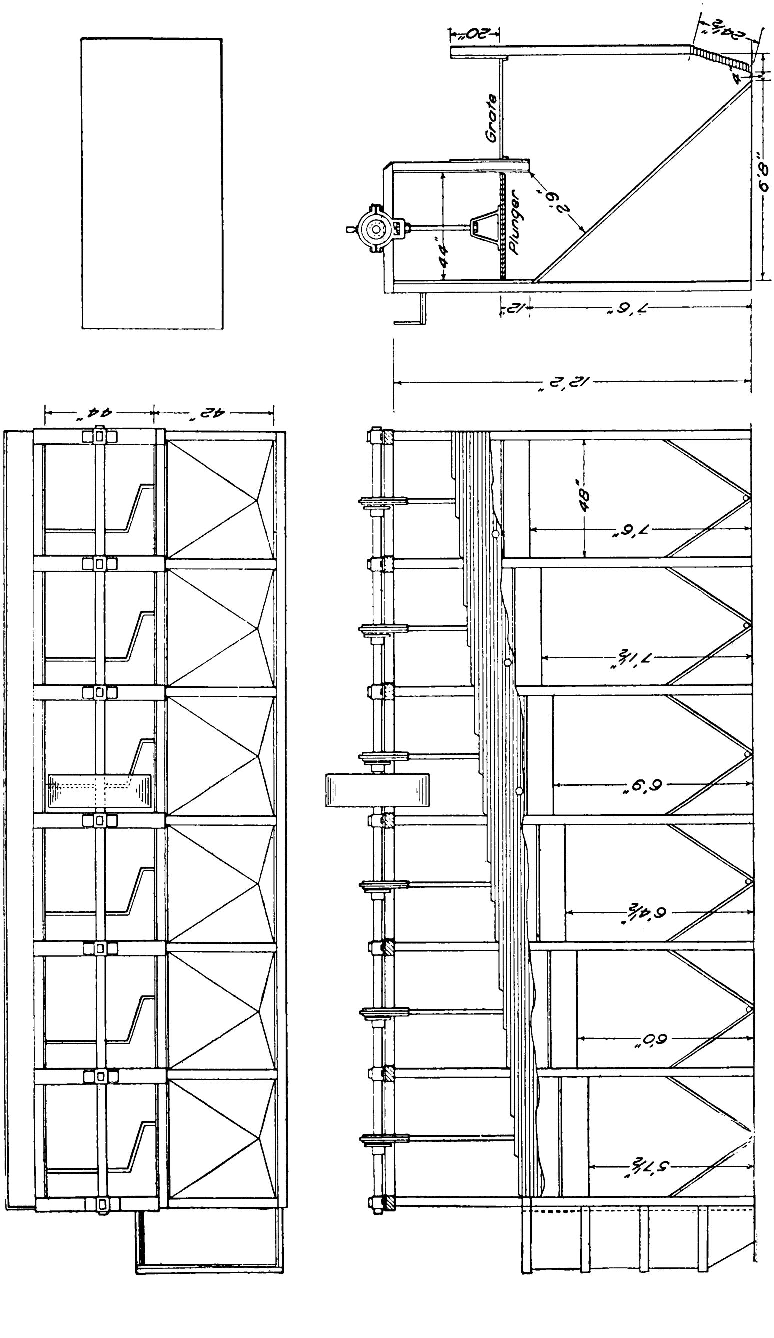
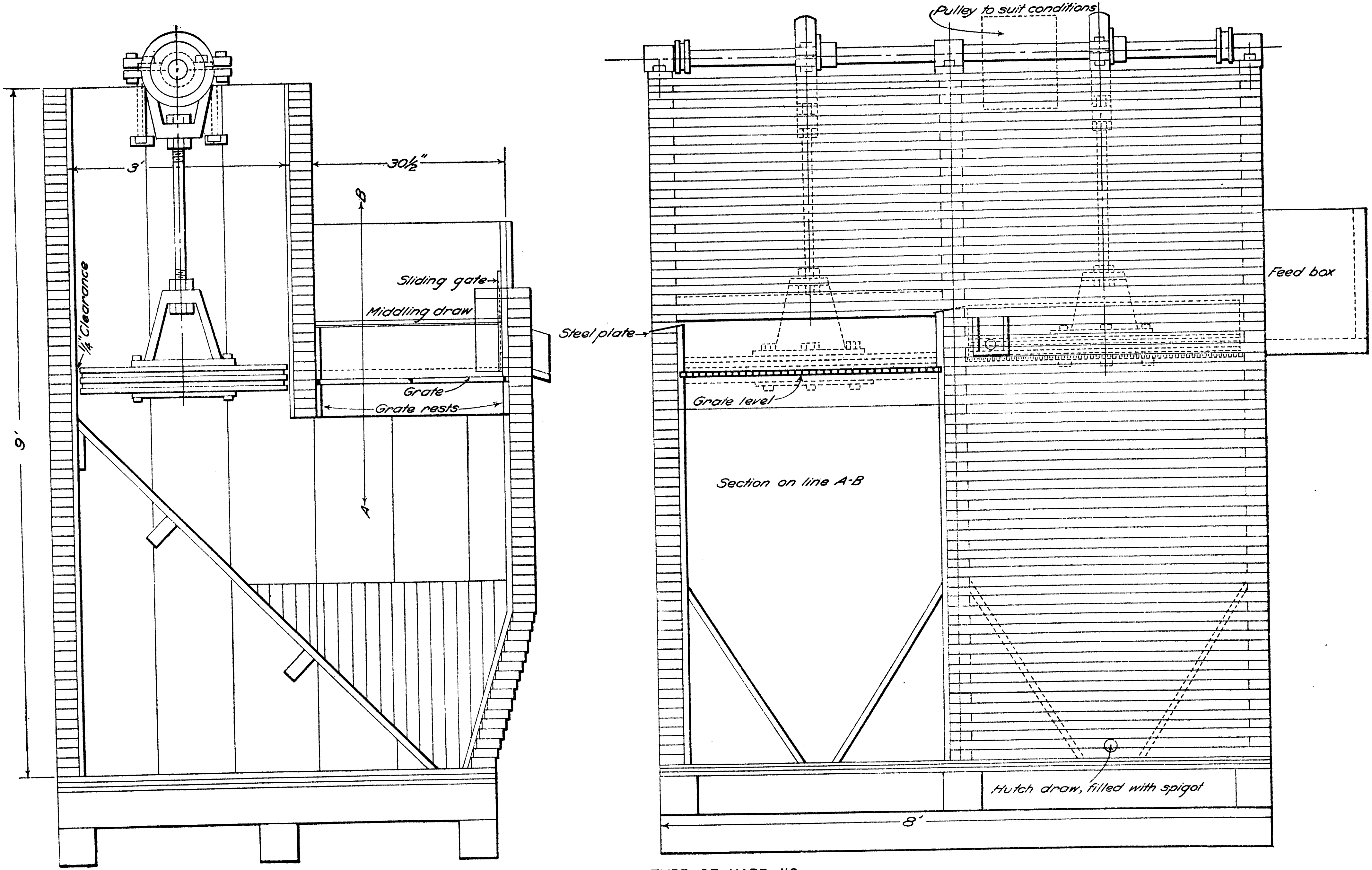


FIGURE 9.—A recent design of Cooley jig.



COOLEY TYPE OF HARZ JIG.

tailing end of the cleaner jig, and other jigs if used, also passes to the settling tanks for subsequent treatment.

The rougher jigs usually have 5 or 6 cells with a screening or grate area ranging from 30 by 42 inches to 36 by 48 inches. The speed of the shaft connecting the plungers and the eccentrics varies from 90 to 120 revolutions per minute; the length of stroke of the plungers ranges from $\frac{5}{8}$ inch to $1\frac{1}{4}$ inches. The cleaner jigs each have 6 or 7 cells with a grate area somewhat smaller than that of the rougher jigs. The speed of the shaft connecting the plungers and the eccentrics of cleaner jigs is 160 to 200 revolutions per minute; the length of stroke ranges from $\frac{3}{8}$ to $\frac{3}{4}$ inch. The chat or sand jigs, which are not commonly used, are smaller in size and usually have 4 or 5 cells each, and are operated at a higher speed and shorter stroke.

The material fed to the first cell of the rougher jig is, as a rule, not graded or classified. A bed 5 to 6 inches deep is formed as a result of the pulsating action of the water, the lighter gangue material, such as flint, rises to the top of the bed, the heavier free grains of lead and zinc minerals working down to the bottom. The downward suction stroke draws the finer grains of mineral through the screen or grate openings into the hutch of each compartment. The strength of the suction is increased by leaving the gates of the hutch partly open. The material that accumulates in the hutch is known as "smittem," and is further treated on the cleaner jig. The bed products of the first two or three cells are also drawn off and pass with the hutch product to the cleaner jig. The chats or particles containing included lead and zinc mineral, usually from the last hutch and the last two or three beds, are drawn off and reground before further treatment.

Various tests conducted by the writer at different mills throughout the district indicate that a few minor changes in the flow sheet or treatment of the ore would at many of the mills effect more efficient jigging and a greater saving. To publish the data on each test would require too much space, but the results of some of the tests are given to bring out the important points under discussion.

EFFICENCY OF JIGS.

A few results of efficiency tests made on rougher jigs are given to show the average percentage of efficiency of rougher jigs on zinc-blende ore.

The treatment of rougher jigs can be considered a closed circuit in each example given, because the chats or middlings were being reground and returned to the feed of the rougher jig. On this basis, the percentage of efficiency or zinc recovery can readily be calculated by the formula

$$R=\frac{100\times c(h-t)}{h(c-t)}$$

where R represents percentage of efficiency, h represents the zinc content of the head or feed to the rougher jig, c the zinc content of the concentrate product from the jig, and t the zinc content of the tailing as it leaves the jig. In the first example the feed (h) assayed 2.85 per cent zinc, the concentrate (c) 25.5 per cent zinc, and the tailing (t) 1.10 per cent zinc. Substituting these figures in the formula and solving show that the efficiency or percentage of zinc recovery from the rougher jig was 64.17 per cent. In other words this jig saved only 64.17 per cent of the total zinc content in the feed. However, in this mill the tailing, discharged from the last cell, was dewatered over a trommel before leaving the mill as waste. Dewatering eliminated a considerable proportion of slime from the tailing, and as the slime, as a rule, assays considerably higher than the coarser material, the tailing assay was lowered from 1.10 to 0.70 per cent zinc.

RESULTS OF EFFICIENCY TESTS OF ROUGHER JIGS.

EXAMPLE 1. SHEET-GROUND ORE.

	Zinc content, per cent.
Feed or head (h)	2.85
Concentrate (c)	25.5
Tailing (t) discharge from last cell	1.10
Dewatered tailing	.70

$$R=\frac{100\times c\ (h-t)}{h\ (c-t)}=\frac{100\times 25.5\ (2.85-1.10)}{2.85\ (25.5-1.80)}=64.17 \text{ per cent.}$$

EXAMPLE 2. SHEET-GROUND ORE.

Feed or head (h)	1.13
Concentrate (c)	15.0
Tailing (t) discharge from last cell	.50
Dewatered tailing	.38

$$R=\frac{100\times c\ (h-t)}{h\ (c-t)}=\frac{100\times 15.0\ (1.13-0.50)}{1.13\ (15.0-0.50)}=57.67 \text{ per cent.}$$

EXAMPLE 3. HARD-GROUND ORE OTHER THAN SHEET-GROUND.

Feed or head (h)	1.20
Concentrate (c)	9.57
Tailing (t) discharge from last cell	.55
Dewatered tailing	.34

$$R=\frac{100\times c\ (h-t)}{h\ (c-t)}=\frac{100\times 9.57\ (1.20-0.55)}{1.20\ (9.57-0.55)}=57.47 \text{ per cent.}$$

EXAMPLE 4. ORE IN MIAMI DISTRICT.

Feed or head (h)	1.99
Concentrate (c)	14.1
Tailing (t) discharge from last cell	.87
Dewatered tailing	.82

$$R=\frac{100\times c\ (h-t)}{h\ (c-t)}=\frac{100\times 14.1\ (1.99-0.87)}{1.99\ (14.1-0.87)}=59.98 \text{ per cent.}$$

From the results given, and others on hand, it can be stated that 20 to 30 per cent of the total zinc in the ore treated is lost in the first stage of concentration over rougher jigs. Hence, some tests are presented to show what the chief causes of this loss are and how it can be lessened.

LOSSES FROM JIGS.

The feed to the rougher jig, which in most mills is ungraded material, consists of both coarse and fine material. The results ot screen analyses of the feed, the total tailing from the rougher jig, and the dewatered tailing, and the assays of the various screen products of two mills are given in Tables 17 and 18 following. Table 17 represents the results of treating sheet-ground ore over a rougher jig. Table 18 represents the results of treating of a chatty ore in the Miami district over a rougher jig.

TABLE 17.—*Results of screen analyses in treatment of sheet-ground ore over rougher jig.*

A. FEED TO THE ROUGHER JIG.

Screen.			Weight of screen products.	Cumulative weight.	Zinc content by assay.	Percentage of total zinc.	Cumulative zinc content.
Size of mesh.	Size of opening.						
	Mm.	Inches.					
			Per cent.	*Per cent.*	*Per cent.*		*Per cent.*
3	6.680	0.263	9.26	9.26	0.55	4.53	4.53
6	3.327	.131	44.08	53.34	.65	26.18	30.71
10	1.651	.065	22.10	75.44	1.02	20.07	50.07
20	.833	.0328	10.14	85.58	1.16	10.46	61.24
35	.417	.0164	5.55	91.13	1.68	8.30	69.54
65	.208	.0082	3.16	94.29	2.15	6.04	75.58
100	.147	.0058	1.27	95.56	3.17	3.60	79.18
150	.104	.0041	.97	96.53	3.74	3.22	82.40
200	.074	.0029	.19	96.72	4.52	.79	83.19
<200	.074	.0029	3.28	100.00	5.77	16.81	100.00
Total or average			100.00	....	1.13	100.00	....

B. TAILING FROM ROUGHER JIG BEFORE PASSING DEWATERING SCREEN.

3	6.680	0.263	7.66	7.66	0.53	8.06	8.06
6	3.327	.131	46.80	54.46	.35	32.48	40.54
10	1.651	.065	23.83	78.29	.32	15.10	55.64
20	.833	.0328	9.57	87.86	.26	4.93	60.57
35	.417	.0164	4.25	92.11	.23	1.94	62.51
65	.208	.0082	2.08	94.19	.32	1.31	63.82
100	.147	.0058	.98	95.17	.51	.97	64.79
150	.104	.0041	1.06	96.23	1.14	2.40	67.19
200	.074	.0029	.15	96.38	1.85	.55	67.74
<200	.074	.0029	3.62	100.00	4.50	32.26	100.00
Total or average			100.00	....	.50	100.00	....

C. DEWATERED TAILING FROM ROUGHER JIG.

3	6.680	0.263	7.20	7.20	0.51	9.69	9.69
6	3.327	.131	50.49	57.69	.36	42.30	51.99
10	1.651	.064	27.81	85.50	.34	25.04	77.03
20	.833	.0328	8.72	94.22	.28	6.46	83.49
35	.417	.0164	2.84	97.06	.25	1.91	85.40
65	.208	.0082	.98	98.04	.27	.69	86.09
100	.147	.0058	.41	98.45	.43	.46	86.55
150	.104	.0041	.39	98.84	.96	.98	87.53
200	.074	.0029	.07	98.91	1.90	.35	87.88
<200	.074	.0029	1.09	100.00	4.20	12.12	100.00
Total or average			100.00	....	.38	100.00	....

TABLE 18.—*Results of screen analyses in treatment of chatty ore over rougher jig.*

[Miami district.]

A. FEED TO THE ROUGHER JIG.

Screen.			Weight of screen products.	Cumulative weight.	Zinc content by assay.	Percentage of total zinc.	Cumulative zinc content.
Size of mesh.	Size of opening.						
	Mm.	Inches.					
			Per cent.	*Per cent.*	*Per cent.*		*Per cent.*
3	6.680	0.263	2.00	2.00	1.20	1.21	1.21
6	3.327	.131	26.20	28.20	1.25	16.48	17.69
10	1.651	.065	21.26	49.46	1.32	14.13	31.82
20	.833	.0328	9.16	58.62	1.82	8.39	40.21
35	.417	.0164	6.32	64.94	3.20	10.17	50.38
65	.208	.0082	6.78	71.72	4.00	13.66	64.04
100	.147	.0058	5.15	76.87	2.90	7.52	71.56
150	.104	.0041	6.05	82.92	2.65	8.07	79.63
200	.074	.0029	4.65	87.57	2.67	6.24	85.87
<200	.074	.0029	12.43	100.00	2.25	14.13	100.00
Total or average			100.00	....	1.99	100.00	....

B. TAILING FROM ROUGHER JIG BEFORE PASSING DEWATERING SCREEN.

3	6.680	0.263	1.91	1.91	0.75	1.65	1.65
6	3.327	.131	48.43	50.34	.75	41.76	43.41
10	1.651	.065	21.87	72.21	.85	21.43	64.84
20	.833	.0328	5.92	78.13	.90	6.15	70.99
35	.417	.0164	2.67	80.80	.90	2.75	73.74
65	.208	.0082	2.20	83.00	.85	2.20	75.94
100	.147	.0058	2.10	85.10	.75	1.76	77.70
150	.104	.0041	2.67	87.77	.80	2.42	80.12
200	.074	.0029	2.20	89.97	1.25	3.18	83.30
<200	.074	.0029	10.03	100.00	1.45	16.70	100.00
Total or average			100.00	....	.87	100.00	....

C. DEWATERED TAILING FROM ROUGHER JIG.

3	6.680	0.263	1.14	1.14	0.75	1.05	1.05
6	3.327	.131	55.48	56.62	.75	50.93	51.98
10	1.651	.065	26.60	83.22	.85	27.70	79.68
20	.833	.0328	8.39	91.61	.85	8.71	88.39
35	.417	.0164	1.61	93.22	.95	1.86	90.25
65	.208	.0082	.90	94.12	.80	.87	91.12
100	.147	.0058	.81	94.93	.75	.75	91.87
150	.104	.0041	.81	95.74	.95	.93	92.80
200	.074	.0029	.47	96.21	1.20	.70	93.50
<200	.074	.0029	3.79	100.00	1.40	6.50	100.00
Total or average			100.00	....	.82	100.00	....

Part A of Table 17 shows that 3.28 per cent of the feed (crushed sheet-ground ore) to the rougher jig consisted of material finer than 200-mesh; its zinc content was 5.77 per cent, which represented 16.81 per cent of the total zinc in the feed. Part B of Table 17, which represents the screen analysis of the tailing before it was dewatered, shows that the assay value of the material finer than 200-mesh was 4.50 per cent zinc. The percentage of material finer than 200-mesh in the tailing was 3.62, which represented 32.26 per cent of the total zinc in the tailing. The screen analysis of the dewatered tailing, part C of Table 17, showed that there was still 1.09 per cent of the material finer than 200-mesh contained in the tailing, with an assay value of 4.20 per cent of zinc, representing 12.12 per cent of the total zinc contained in the dewatered tailing that was discarded.

DESLIMING AT HEAD OF ROUGHER JIGS ADVISABLE.

It is reasonable to believe, from these figures alone, that desliming at the head of the rougher jig would be advisable. As the results in Table 17 show the necessity of desliming at the head of the rougher jig when a sheet-ground ore is treated, the results in Table 18 show a much greater proportion of slimes in the feed to the rougher when the ore is "chatty" or more or less disseminated and is ground finer.

However, the importance of desliming is more strongly emphasized by the fact that the dewatering screens as used at the end of the jigs in this district are, as a rule, not very efficient, so that fine mineral is lost in the jig tailing. To show this more fully, the screening efficiency of the dewatering screens used at the end of the two rougher jigs under discussion was determined. It should be stated that the main object of the dewatering screens in this district is not only to eliminate the water from the tailing, but also to collect as much as possible of the fines from the tailing overflow for further treatment on tables.

EXAMPLE 1.—EFFICIENCY OF DEWATERING SCREEN WITH 1.5-MM. OPENINGS, AT END OF ROUGHER JIG TREATING SHEET-GROUND ORE.

Product, size of opening.	Feed, per cent.	Oversize, per cent.
Over 1.5 mm	80. 1	87. 5
Through 1.5 mm	19. 9	12. 5
	100. 0	100. 0

Weight of undersize in the oversize $=\frac{80.1\times12.5}{87.5}=11.44.$

$$\text{Screening efficiency}=\frac{\text{Weight of undersize passing through screen}}{\text{Total weight of undersize in feed to screen}}=$$
$$\frac{(19.9-11.44)\times100}{19.9}=42.51 \text{ per cent.}$$

The efficiency of a dewatering screen with 1.5-mm. openings used at the end of a rougher jig treating a chatty ore of the Miami district showed that 52.47 per cent of the undersize in the feed to the screen was screened through the 1.5-mm. openings. In this mill the dewatering screen used was an ordinary flat shaking screen, instead of the usual revolving trommel. The screening efficiency was as follows:

EXAMPLE 2.—EFFICIENCY OF FLAT DEWATERING SCREEN IN MILL TREATING CHATTY ORE.

Product, size of opening.	Feed, per cent.	Oversize, per cent.
Over 1.5 mm	73. 7	85. 5
Through 1.5 mm	26. 3	14. 5
	100. 0	100. 0

Weight of undersize in the oversize $=\frac{73.7\times14.5}{85.5}=12.5.$

$$\text{Screening efficiency}=\frac{\text{Weight of undersize passing through screen}}{\text{Total weight of undersize in feed to screen}}=$$
$$\frac{(26.3-12.5)\times100}{26.3}=52.47 \text{ per cent.}$$

To show more fully the screening efficiencies of dewatering screens in various mills, two more examples are given.

EXAMPLE 3.—EFFICIENCY OF DEWATERING SCREEN IN MILL TREATING HARD-GROUND ORE OTHER THAN SHEET-GROUND.

[Feed to rougher jig crushed to $\frac{7}{16}$ inch.]

Product, size of opening.	Feed, per cent.	Oversize, per cent.
Over 1.5 mm	81.6	91.0
Through 1.5 mm	18.4	9.0
	100.0	100.0

Weight of undersize in the oversize$=\frac{81.6\times 9}{91.0}=8.07$.

$$\text{Screening efficiency}=\frac{\text{Weight of undersize passing through screen}}{\text{Total weight of undersize in feed to screen}}=\frac{(18.4-8.07)\times 100}{18.4}=65.2 \text{ per cent.}$$

EXAMPLE 4.—EFFICIENCY OF DEWATERING SCREEN IN MILL TREATING SHEET-GROUND ORE.

[Feed to rougher jig crushed to $\frac{1}{2}$ inch.]

Product, size of opening.	Feed, per cent.	Oversize, per cent.
Over 2 mm	63.8	64.8
Through 2 mm	36.2	15.2
	100.0	100.0

Weight of undersize in the oversize$=\frac{63.8\times 15.2}{84.8}=11.43$.

$$\text{Screening efficiency}=\frac{\text{Weight of undersize passing through screen}}{\text{Total weight of undersize in feed to screen}}=\frac{(36.2-11.43)\times 100}{36.2}=68.37 \text{ per cent.}$$

The average efficiency of dewatering screens in the mills of the Joplin district can be given as 40 to 60 per cent. The results in example 4 show the highest screening efficiency obtained from a number of dewatering screens tested. The screen consisted of a 5-foot trommel running at a speed of $1\frac{1}{2}$ revolutions per minute. This relatively higher efficiency was attained, however, only after careful adjustment of the feed. As the efficiency of the trommel or the quantity of undersize passing through the openings depends on the flow of water used with the tailing feed to force the undersize through, it is essential that the tailing flow strike the trommel in such a way that most of the water will pass through the openings. In a few mills it was noticed that the line of flow was more or less tangent to the surface of the revolving screen so that a large part of the water and fines had no chance to pass through the openings.

In some mills no dewatering screens are used, but a rectangular box is added to the end of the jig. From the bottom of this box the coarse tailing is discharged through a spigot while the fines and water are allowed to overflow at the top. In a few mills water is discharged under pressure into the box to cause as much fines and slimes as possible to overflow with the water, thus eliminating them from the spigot discharge, but with rather unsatisfactory results on the whole.

Another point in favor of desliming at the head of the rougher jig is that the greater the content of slimes in the feed the more will the valuable or fine mineral particles be forced over the jig by the flow of top water, especially as the quantity of water added to any cell of the jig increases the top water of each succeeding cell. Also, better work can be done with the jig by having clear top water, which is another reason, when the material is not graded or sized, why the feed to the rougher should be deslimed. However, desliming the jig feed would not necessarily mean eliminating the dewatering screen at the end of the rougher jig.

SUGGESTED METHODS OF DESLIMING AHEAD OF ROUGHER JIG.

Several ways of desliming at the head of the rougher jig could be followed. A simple and cheap device would be a V-box with one or two discharge spigots attached to the bottom, the overflow of fines and slimes at the top being assisted by water under hydraulic pressure. A somewhat more efficient means would be the use of a trommel screen with 1½ or 2 mm. openings, although the upkeep would be higher. The feed would discharge into one end of the trommel on the inside, the undersize working through the openings in the usual way. This method has proved satisfactory in the desliming and retreatment of tailings.

Another effective method of desliming at the head of rougher jigs would be the use of a large size classifier similar to the Akins. The Akins is mentioned here as it is simple and would not require any extensive changes in the arrangement of the mill as far as headroom is concerned nor loss in drop from the "dirt" or feed trommel to the rougher jig. The cost of this desliming screen would be about $1,000 installed.

The Woodbury desliming jig might be used to good advantage. This jig would deslime the feed and also eliminate a large proportion of the lead mineral, thus permitting the rougher jig to do more efficient work on the remaining zinc ore. Such a jig would increase the efficiency of the roughing system, but the first cost would be higher than for any of the other devices mentioned, and in most mills more headroom would be needed and the feed trommels and feed elevators would have to be rearranged.

DESLIMING AT SECOND OR THIRD CELL OF ROUGHER JIG.

Instead of desliming at the head of the rougher jig a means has been devised for desliming the material after it has reached the second or third cell of the jig (see fig. 10). A wooden baffle is fastened to the ends of two strips of wood, the opposite ends of which are pivoted to the inner sides of a jig cell. This baffle is placed at right angles to the flow and adjusted to move up and down with the pulsating action of the plungers while resting on top of the bed, the water and slimes being removed just before they flow over the partition between the cells. The slimes and top water are diverted off the bed at the plunger side of the jig and by means of a launder conveyed to settling tanks; the gangue and remaining ore pass under the baffle into the next cell. Not only is a large proportion of slimes and excessive top water removed by having these baffles at one or more cells, but the speed of the surface currents or top water is reduced, thus providing a steadier discharge with less disturbance of the bed at the head of each succeeding cell. Such baffles, in the few mills where their use has been observed by the writer, eliminate sufficient slimes from the top water to give practically clear water in the last cell of the jig.

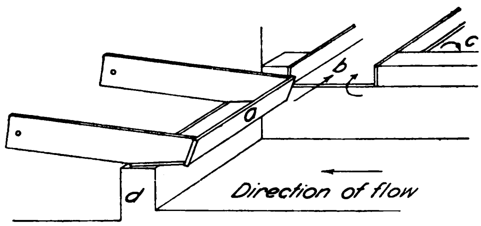


FIGURE 10.—Device used for desliming top water on jigs. *a*, Baffle; *b*, top water and slimes to settling tanks; *c*, plunger cells; *d*, partition between cells.

RETREATING RECRUSHED CHATS OVER SEPARATE JIGS AND TABLES.

The chats from the rougher jig are recrushed and treated over a chat jig or returned to the rougher jig. If treated over a chat or sand jig, the crushed material is usually cleaned on the jig direct, without a preliminary roughing. It would seem advisable, where the quantity of chats produced from both the rougher and cleaner jigs is relatively large, to treat the recrushed chats over a rougher chat jig and the roughed product over a cleaner chat jig. If only a moderate quantity of chats were produced, they could be crushed to a certain size, about one-eighth to three-sixteenths inch, and then treated over a rougher chat jig. The enriched product obtained could be given a final treatment over the main cleaner jig, and thus a cleaner chat jig would not be needed. The middling or chats from the rougher chat jig could be either returned to the chat rolls and retreated over the chat jig or sent to sand rolls and treated on tables.

Another method of treating the chats from rougher and cleaner jigs would be to crush to table size, give the crushed material a preliminary treatment on roughing tables to eliminate much of the waste sand, and run the product over the usual sand tables.

One of the chief reasons for treating the chats separately, and not returning them to the rougher jig feed, is that the recrushed chats is an enriched product, assaying 4 to 8 per cent zinc, whereas the feed to the rougher jig assays only about 1.5 to 3 per cent zinc. A higher recovery is possible with a richer feed than with a leaner one; hence a greater percentage of the zinc in the relatively richer chats could be saved by treating them separately than by returning them to the rougher jig feed and mixing them with the relatively much lower feed. Screen analyses and assays showing the differences in zinc content of chats from both rougher and cleaner jigs as compared to the feed to the rougher jigs are shown in Table 19 following.

TABLE 19.—*Results of screen analyses of feeds and products of rougher and cleaner jigs.*

FEED TO ROUGHER JIG NO. 1.

Screen.			Weight of screen products.	Cumulative weight.	Zinc content by assay.	Percentage of total zinc.	Cumulative zinc content.
Size of mesh.	Size of opening.						
	Mm.	Inches.					
			Per cent.	*Per cent.*	*Per cent.*		*Per cent.*
3	6.680	0.263	12.16	....	1.24	4.8	....
6	3.327	.131	33.71	45.87	1.96	20.9	25.7
10	1.651	.065	17.65	63.52	2.84	15.85	41.55
20	.833	.0328	12.01	75.53	3.25	12.4	53.95
35	.417	.0164	10.55	86.08	4.02	13.4	67.35
65	.208	.0082	5.67	91.75	6.55	11.75	79.10
100	.147	.0058	2.25	94.00	10.15	7.2	86.30
150	.104	.0041	.60	94.60	10.00	1.9	88.20
200	.074	.0029	2.37	96.97	8.10	6.1	94.30
<200	.074	.0029	3.03	100.00	5.93	5.7	100.00
Total or average............			100.00	....	3.16	100.00	....

FEED TO ROUGHER JIG NO. 2.

3	6.680	0.263	15.73	....	1.08	6.8	....
6	3.327	.131	34.50	50.23	1.76	24.4	31.2
10	1.651	.065	19.44	69.67	2.12	16.5	47.7
20	.833	.0328	10.00	79.67	2.58	10.3	58.0
35	.417	.0164	8.18	87.85	3.25	10.7	68.7
65	.208	.0082	4.56	92.41	5.58	10.2	78.9
100	.147	.0058	1.98	94.39	7.65	6.1	85.0
150	.104	.0041	1.17	95.56	8.71	4.1	89.1
200	.074	.0029	2.03	97.59	6.96	5.7	94.8
<200	.074	.0029	2.41	100.00	5.38	5.2	100.0
Total or average............			100.00	....	2.47	100.00	....

CHATS FROM ROUGHER JIG NO. 1.

3	6.680	0.263	7.79	....	4.00	7.37	....
6	3.327	.131	27.27	35.06	7.35	47.41	54.78
10	1.651	.065	27.18	62.24	3.45	22.18	76.96
20	.833	.0328	12.42	74.66	1.85	5.44	82.40
35	.417	.0164	15.24	89.90	1.40	5.05	87.45
65	.208	.0082	5.97	95.87	2.35	3.32	90.77
100	.147	.0058	1.60	97.47	6.30	2.39	93.16
150	.104	.0041	.56	98.03	11.40	1.52	94.68
200	.074	.0029	.46	98.49	12.45	1.27	95.95
<200	.074	.0029	1.51	100.00	11.30	4.05	100.00
Total or average...........			100.00	....	4.23	100.00	....

TABLE 19.—*Results of screen analyses of feeds and products of rougher and cleaner jigs*—Continued.

CHATS FROM ROUGHER JIG NO. 2.

Screen.			Weight of screen products.	Cumulative weight.	Zinc content by assay.	Percentage of total zinc.	Cumulative zinc content.
Size of mesh.	Size of opening.						
	Mm.	Inches.					
			Per cent.	*Per cent.*	*Per cent.*		*Per cent.*
3	6.680	0.263	5.98	....	1.19	2.8	....
6	3.327	.131	16.82	22.80	2.25	15.2	18.0
10	1.651	.065	29.15	51.95	2.85	33.4	51.4
20	.833	.0328	23.09	75.04	1.91	17.8	69.2
35	.417	.0164	15.85	90.89	1.60	10.2	79.4
65	.208	.0082	5.95	96.84	2.37	5.7	85.1
100	.147	.0058	1.34	98.18	9.21	5.0	90.1
150	.104	.0041	.13	98.31	9.50	.5	90.6
200	.074	.0029	.75	99.06	17.40	5.3	95.9
<200	.074	.0029	.94	100.00	10.80	4.1	100.0
Total or average..........			100.00	....	2.48	100.00	....

CHATS FROM CLEANER JIG.

6	3.327	0.131	8.82	....	28.80	19.98	....
10	1.651	.065	19.45	28.27	10.45	15.99	35.97
20	.833	.0328	20.38	48.65	7.70	12.35	48.32
35	.417	.0164	27.41	76.06	10.05	21.67	69.99
65	.208	.0082	17.05	93.11	12.80	17.17	87.16
100	.147	.0058	4.18	97.29	23.00	7.57	94.73
150	.104	.0041	.41	97.70	26.50	.86	95.59
200	.074	.0029	1.34	99.04	27.45	2.89	98.48
<200	.074	.0029	.96	100.00	20.10	1.52	100.00
Total or average..........			100.00	....	12.71	100.00	....

SPIGOT TAILING FROM CLEANER JIG.

6	3.327	0.131	1.62	....	12.40	2.0	....
10	1.651	.065	14.70	16.32	7.83	11.3	13.3
20	.833	.0328	21.84	38.16	6.75	14.5	27.8
35	.417	.0164	37.85	76.01	7.30	29.45	57.25
65	.208	.0082	16.63	92.64	14.05	23.0	80.25
100	.147	.0058	4.39	97.03	29.10	12.55	92.80
150	.104	.0041	.69	97.72	28.80	1.95	94.75
200	.074	.0029	1.40	99.12	27.30	3.75	98.50
<200	.074	.0029	.88	100.00	17.40	1.5	100.00
Total or average..........			100.00	....	10.15	100.00	....

TOTAL CHATS FROM JIGS.

3	6.680	0.263	4.47	....	3.82	3.0	....
6	3.327	.131	15.89	20.36	6.45	18.2	21.2
10	1.651	.065	24.74	45.10	5.05	22.2	43.4
20	.833	.0328	20.64	65.74	3.30	12.1	55.5
35	.417	.0164	22.10	87.84	4.75	18.7	74.2
65	.208	.0082	8.16	96.00	8.50	12.3	86.5
100	.147	.0058	2.02	98.02	22.60	8.1	94.6
150	.104	.0041	.40	98.42	19.80	1.4	96.0
200	.074	.0029	.77	99.19	16.40	2.3	98.3
<200	.074	.0029	.81	100.00	11.80	1.7	100.0
Total or average..........			100.00	....	5.63	100.0	....

All of the screen analyses and assays in Table 19 are on feeds and products from the same mill. The tailing from the cleaner jig is included in the total chats from all the jigs.

Other points in favor of treating the recrushed chats over chat jigs is that returning them to the rougher jig would (1) increase the quantity of fines and slimes in the feed to that jig, and (2) when the quantity of chats is relatively large, reduce its capacity to a certain extent, and (3) increase the chance for loss in the finest sizes. This also brings out the advisability of desliming at the head of the rougher jig, the reasons for which have already been given.

CLEANER JIGS.

The efficiency of cleaner jigs used in the mills of this district varies from 75 to 85 per cent. At the end of the jig there is usually a dewatering box similar to the kind used at the end of rougher jigs where dewatering screens are not used. The tailing is discharged from the box through a spigot and the overflow passes out into settling tanks. The tailing is seldom discarded, being usually reground for further treatment.

SUGGESTIONS FOR IMPROVEMENT OF JIGGING PRACTICE.

Based on the preceding data the chief suggestions offered for jigging practice in the mills of the Joplin district are as follows: (1) Desliming at the head of the rougher jigs; (2) reducing the quantity of unnecessary top water on the rougher jigs; and (3) retreating recrushed chats over separate chat jigs or tables by the use of a roughing and cleaning system.

Another suggestion that might be added is the use of two or more rougher jigs in mills treating 200 tons or more in 10 hours. In many of the mills visited, the addition of a second rougher jig would seem advisable to prevent the crowding caused by forcing the entire feed over one rougher jig and thus reducing its efficiency, to permit a better separation of the mineral from the gangue, and to have a relatively larger capacity available whenever it should be found necessary. Also, by having two rougher jigs the quantity of water used per ton of ore and the effective action of the jig could be more readily controlled.

CONCENTRATION OF FINE MATERIAL.

During the past few years there has been a marked improvement in the treatment of the finer material of the lead and zinc ores in this district. A few years ago tables were seldom used, but now nearly every mill contains 3 to 10 tables, and in a few of the more recent mills 20 or more tables are used. The small number of tables generally used is due to the small tonnage of ore treated, the capacity of the plants ranging from 100 to 500 tons in 10 hours, and the attempt to minimize the quantity of fine material. The aim in the following pages is to emphasize the more important factors necessary to good table practice and to show possibilities of improving the average practice in the district.

OUTLINE OF PRACTICE IN THE TREATMENT OF FINE MATERIAL.

The feed to the table section, locally known as the "sludge" section, of any mill in this district comes chiefly from the dewatering screens and overflow waters from dewatering boxes at the end of jigs. In a few mills a large proportion of the finer sands and slimes is eliminated from the feed to the jigs, being diverted by launders to the settling tanks. The type of settling tank commonly used is rectangular, about 12 feet wide, 20 to 25 feet long, 2 to 3 feet deep at one end, and 6 to 8 feet deep at the other end, with the bottom sloping toward one corner of the tank. In most mills the material is fed into the settling tank at the deep and overflows at the shallow end, although at a few plants the feed enters the shallow and overflows at the deep end. Some operators are now using double tanks in which the overflow at the shallow end of the first tank flows into the shallow end of a second similar tank, which is placed parallel to the first tank, and discharges at the deep end of the second tank. The double-tank system gives the material a double settling and a rough classification of coarse and fine sands. When the settling tanks, or first tank where the double-tank system is used, has become about two-thirds full of sand, the flow or feed is diverted into another tank or set of tanks, while the material in the filled tank is discharged to the table section of the mill.

Recently Dorr thickeners have been installed at several mills; in general their purpose has been chiefly to settle and thicken slimes, although at a few mills they have been used for treating the coarser sands in order to give a more uniform feed to the tables.

At most mills the feed to the settling tanks enters a V-box with a continuous discharge, thus eliminating much of the coarser sand from the feed overflowing at the top of the box into the settling tank, and incidentally increasing somewhat the settling capacity of the large tanks.

The material discharged from both the V-box and the settling tank is raised by an elevator and discharged onto a trommel. The undersize from the trommel passes through classifiers and thence to tables; the oversize is either discarded as waste or crushed by rolls and returned to the trommel in closed circuit.

FINE SCREENING.

Trommels are most commonly used for screening in the table section of the mills. In a few mills, however, a local type of flat screen known as the Henry screen is used.

USE OF TROMMEL SCREENS.

The efficiency of trommels for screening fine material depends on the same conditions that were given for coarse screening. In general the fine trommel screens are less efficient than the coarser screens, because the openings have more of a tendency to blind and the finer feed particles tend to cling together in smalls balls on the screen as soon as they becomes more or less dewatered. The efficiency of trommels for screening the fines from certain products in the table sections of mills throughout the district varies considerably, as was found from a number of tests. Two examples are given to show both poor and good results.

TEST 1.—SCREENING EFFICIENCY OF TROMMEL WITH 1-MM. OPENINGS; SIZE, 48×72 INCHES; SPEED, 14 REVOLUTIONS PER MINUTE; SLOPE, 1¼ INCHES PER FOOT.

	Trommel feed.	Trommel oversize.
Oversize on 1-mm. screen, per cent	16.1	28.8
Undersize through 1-mm. screen, per cent	83.9	71.2
	100.0	100.0

$$\text{Weight of undersize}=\frac{16.1\times71.2}{28.8}=39.8$$

$$\text{Efficiency}=\frac{\text{Weight of undersize going through the screen}}{\text{Total weight of undersize in feed to screen}}=$$

$$\frac{(83.9-39.8)\times100}{83.9}=52.56 \text{ per cent.}$$

TEST 2.—SCREENING EFFICIENCY OF TROMMEL WITH 1-MM. OPENINGS; SIZE, 48×72 INCHES; SPEED, 16 REVOLUTIONS PER MINUTE; SLOPE, 1½ INCHES PER FOOT.

	Trommel feed.	Trommel oversize.
Oversize on 1-mm. screen, per cent	28.5	63.7
Undersize on 1-mm. screen, per cent	71.5	36.3
	100.0	100.0

$$\text{Weight of undersize}=\frac{28.5\times36.3}{63.7}=16.4.$$

$$\text{Efficiency}=\frac{\text{Weight of undersize passing through screen}}{\text{Total weight of undersize in feed to screen}}=$$

$$\frac{(71.5-16.4)\times100}{71.5}=77.0 \text{ per cent.}$$

The results in the first example show that the rate of speed of the trommel was somewhat slow and that the slope was not sufficient for the speed. It was noticed while the trommel was running that the feed bedded thickly on the screen, preventing a certain proportion of the undersize material from working through the openings. Later this thick bedding was largely remedied by increasing the speed to about 17 revolutions per minute and the slope to 1½ inches per foot. These changes also increased the screening efficiency of the trommel.

In mills where the oversize from sand trommels is discarded as tailing, screen analyses of this oversize should be made and the various screen products assayed to determine in what sizes the mineral values are and what portion is possibly worth saving. The distribution of zinc in an oversize from a sand trommel is shown by the screen analysis and assays in Table 20 following:

TABLE 20.—*Results of screen analysis and assays of oversize from sand trommel.*

Screen.			Weight of screen products.	Cumulative weight.	Zinc content.		
Size of mesh.	Size of openings.				By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
			Per cent.	*Per cent.*	*Per cent.*		
20	0.833	0.0328	45.06	....	1.70	52.0	....
35	.417	.0164	40.87	85.93	.75	20.8	72.8
65	.208	.0082	9.26	95.19	1.55	9.75	82.55
100	.147	.0058	2.36	97.55	4.85	7.75	90.30
150	.104	.0041	.22	97.77	4.85	.75	91.05
200	.074	.0029	1.07	98.84	5.15	3.75	94.80
<200	.074	.0029	1.16	100.00	6.65	5.20	100.00
Total or average..........			100.00	....	1.47	100.00	....

The results show that the zinc content of the oversize was 1.47 per cent zinc. The oversize on the 20-mesh screen assayed 1.70 per cent zinc, whereas the material between 20 and 35 mesh assayed only 0.75 per cent zinc. This indicates that the larger zinc content in the material coarser than 20 mesh was probably due to the presence of particles containing included zinc mineral, or chats, that would have to be reground before further treatment. With all material coarser than 35 mesh eliminated from consideration, let the value of that passing the 35-mesh screen be considered. The calculated zinc content of this undersize is 2.92 per cent, or practically double that of the total oversize. For example, assume that 5 tons of this undersize can be screened from the oversize each day. With an assay value of 2.92 per cent zinc, this quantity would contain 0.146 ton of zinc, or, in terms of 50 per cent zinc concentrates, 0.292 ton. For every 5 tons of this material treated on tables, with a 65 per cent zinc recovery, practically 0.19 ton of 50 per cent zinc concentrate would be saved. At $50 a ton for this product the value of the quantity saved each day would be $9.50. This calculation is simply to show how a profitable, although small, saving can be effected by more efficient screening. In this mill, however, the writer believes that it would have paid to regrind the entire oversize and treat it over the tables with the main table feed.

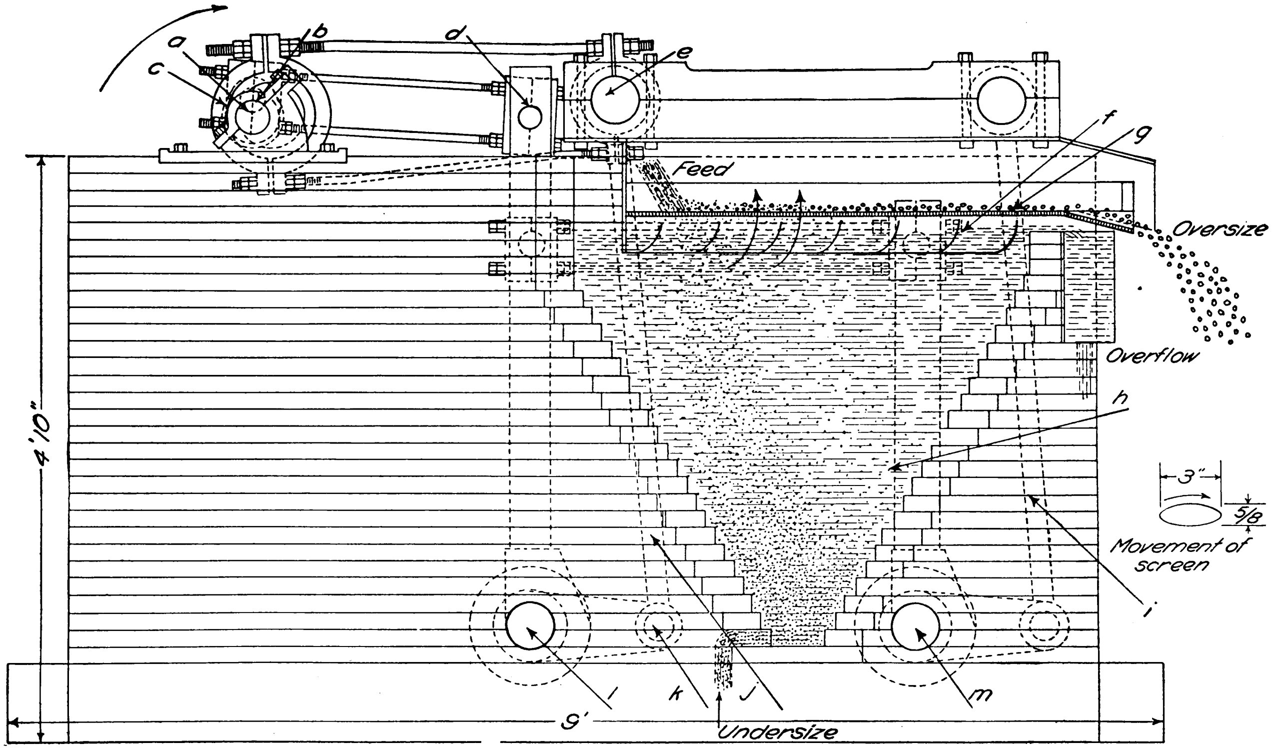


FIGURE 11.—Sketch of Henry screen. *a*, main shaft; *b*, vertical motion crank; *c*, horizontal motion eccentric; *d*, quadrant; *e*, connecitng box; *f*, blades; *g*, screen; *h*, hutch; *i*, *j*, carrier legs; *k*, quadrant; *l*, rear rocker shaft; *m*, front rocker shaft.

THE HENRY SCREEN.

The Henry screen is shown in figure 11. The main frame carries the mechanism that transmits movement to the screen. Within the frame is a hopper-shaped tank, with ample pitch to prevent the fine material adhering to the sides. The driving shaft carries two eccentrics. One of these connects directly to the screen carrier and imparts to the screen a forward-and-backward movement. The second eccentric leads the first 90°, and by means of arms gives the screen an up-and-down movement. The mean of these two movements combined is a rising forward movement during one-half of the revolution of the shaft and a downward backward movement during the other half of the revolution. The material is fed into the rear end of the screen. The resulting action of the screen from each revolution tends to carry the oversize to the forward end, where it is discharged; the undersize passes through the screen into the tank or hutch, where it is drawn off at the bottom by means of a spigot plug.

The water that carries the pulp to the screen passes through the screen into the tank, where its level is maintained, by means of an adjustable gate, at a point below the level of the screen when it is at midstroke. The blades supporting the screen are securely joined to both sides of the screen carrier.

These blades are so shaped that during the downward backward movement of the screen carrier they dip into the water and force it up against the screen with a pulsating effect, thus cleaning the openings of the screen and assisting the forward movement of the oversize particles. As the screen lifts forward from the water it washes the fines through the openings and cleans the coarse sand of adhering particles.

CLASSIFICATION OF FINES.

In the general practice throughout the district, the undersize from the sand trommel passes into classifiers, usually of the ordinary V-box type with water under hydraulic pressure passing up through the sorting column, and discharges onto the tables. The writer believes that separation of the slimes from the sands before classification would be a decided advantage, both in increasing the efficiency of the classifiers and in decreasing the losses in slimes. For desliming, one of the well-known types of desliming classifiers, such as the Dorr, the Akins, or the Federal-Esperanza or some similar type of drag classifier could be used. Both the Dorr and the Akins have been tried in some of the larger mills with satisfactory results, and it is believed that any of the types of deslimers mentioned would greatly increase the efficiency of the table section of any mill. Not only would a better classification of the feeds to the various tables be obtained, but the proportion of water to solids could be more readily regulated for the proper thickness of pulp desired for each table. Also by eliminating the larger proportion of the slimes from the material fed to classifiers these can be more easily adjusted. The pulp which could go direct to some settling tank for thickening, if necessary, would be much thicker than when it passes over the various classifiers and becomes diluted with water from each classifier so as to require a greater ratio of thickening before treatment on subsequent slime tables or by flotation.

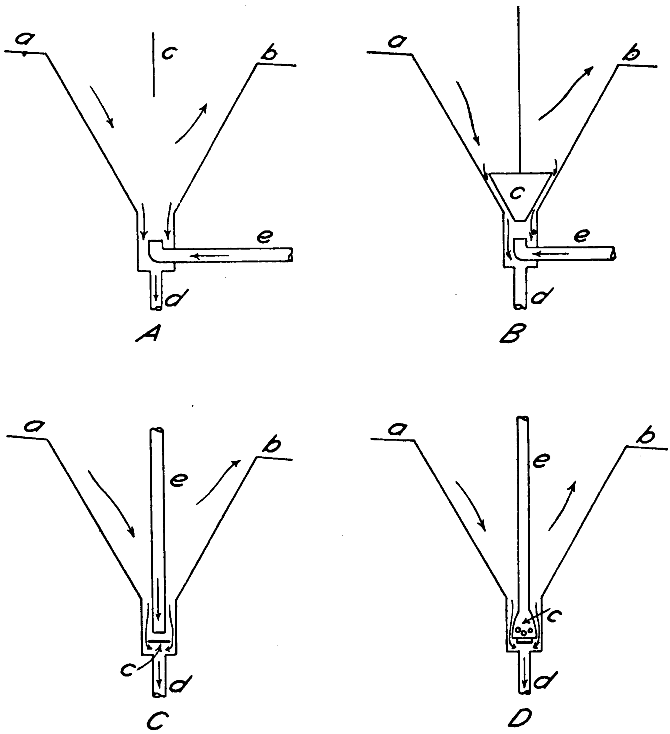


FIGURE 12.—V-type classifiers.

TYPES OF CLASSIFIERS USED.

As stated previously, the usual type of classifier used is a series of V-boxes with hydraulic flow, placed in consecutive order above each table, the overflow from one box flowing through a launder to the next box. The hydraulic water enters each box at a point near the bottom. Different methods of feeding water and pulp are shown in figure 12. In type A, figure 12, the pulp is fed into the classifier at *a*. The relatively coarser and heavier particles drop into the sorting column, through the current of water rising from pipe *e*, and pass through the discharge pipe *d* onto the table. The lighter particles pass under the baffle *c* and overflow at *b*, passing through the launder to the next classifier.

The baffle *c* is placed at right angles to the flow of the feed to retard the speed of the flowing current. Most of the classifiers of the V-box type used in the mills have baffles.

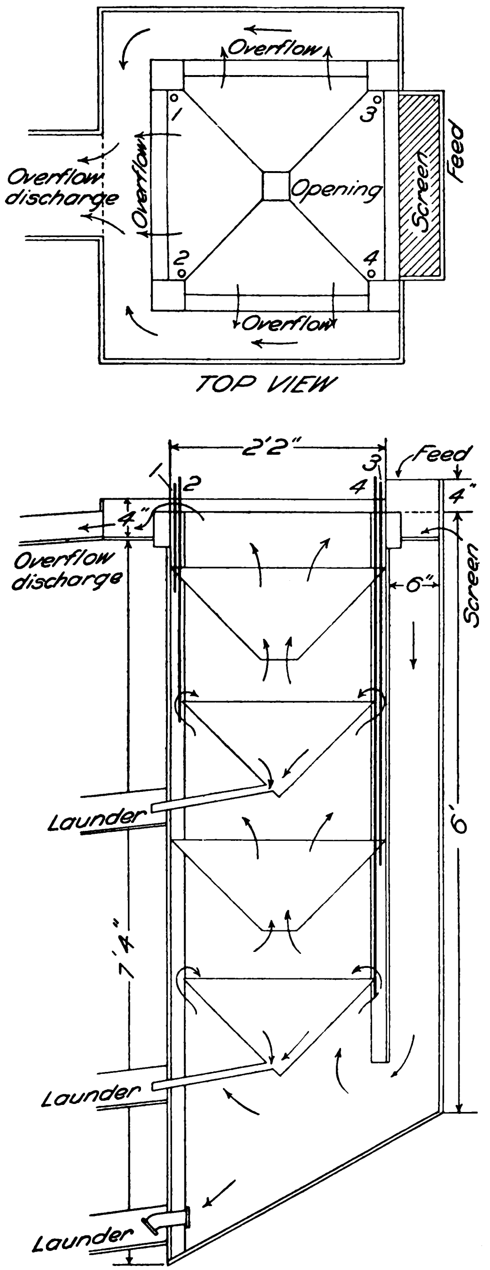


FIGURE 13.—Bird classifier.

Type *B*, figure 12, is the same as type *A*, except that a wooden block *c*, shaped like the frustum of a pyramid, is placed in inverted position at the head of the sorting column, the four sides being parallel with and equidistant from the sloping sides of the classifier. The pulp, as it drops down through the narrow passage thus formed, meets the rising current of water from the pipe *e*; hence all the particles discharged through *d* must pass through this upward flow. The block *c* is fastened to a vertical shaft that can be raised or lowered, thus varying the width of the aperture. The aperture, measured at right angles to the sides, is usually ½ inch to ¾ inch wide, depending on the size of the material passing into the classifier.

In C, figure 12, the water is forced in through a vertical pipe *e* from the top, and not through a horizontal pipe from the side, as in the first two classifiers described. As the water discharges under pressure at the lower end of the vertical pipe *e* it strikes against a horizontal disk or plate *c* of the same shape as the cross section of the sorting column, and consequently tends to rebound up through the sorting column against the downward flow of the pulp. This disk or plate is so placed that its edges are equidistant from the sides of the sorting column, forming an opening ½ to ¾ inch wide. The particles pass through this aperture and discharge at *d*.

The classifier represented in D, figure 12, is found in only a few of the mills. The vertical pipe *e* terminates in a bell that is closed at the end and has around the sides small circular openings, about one-fourth inch in diameter, through which the water passes. The action of the water is about the same as in classifier C, except that the flow is divided by the circular openings into small streams. This type of classifier is said to give good results.

Classifiers of the Richards-Janney type are found in a few mills; also, a local machine known as the Bird classifier. A self-explanatory sketch of this classifier is given in figure 13. The vertical ½-inch air pipes, *1*, *2*, *3*, and *4*, extend into the different parts of the classifier to prevent any disturbance resulting from the formation of air bubbles as the pulp and water rise through the classifier, the air passing out through these pipes.

RESULTS OF SCREEN TESTS OF CLASSIFIER PRODUCTS.

The results of screen tests of feed products to tables from classifiers of the V-box type are shown in Table 21, following. The classifiers all used hydraulic flow, except classifiers 8 and 9. Launders connected the nine classifiers, and each classifier delivered to a separate table. The dimensions of each classifier and the size of the spigot-discharge opening are given in Table 22.

TABLE 21.—*Results of screen tests of spigot products from nine classifiers fed to tables.*

Size of mesh.	Feed to classifiers.		Underflow from classifier 1.		Underflow from classifier 2.		Underflow from classifier 3.	
	Weight.	Cumulative weight.	Weight.	Cumulative weight.	Weight.	Cumulative weight.	Weight.	Cumulative weight.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
On 20	0.36	0.36	0.79	0.79	0.73	0.73	....	....
35	15.03	15.39	25.85	26.64	21.49	22.22	16.12	16.12
65	32.14	47.53	38.49	65.13	42.99	65.21	40.26	56.38
100	18.98	66.51	15.75	80.88	15.67	80.33	18.24	74.62
150	18.71	85.22	10.78	91.66	11.27	92.15	14.29	88.91
200	6.52	91.74	3.60	95.26	3.59	95.74	4.95	93.86
Through 200	8.27	100.01	4.72	99.98	4.25	99.99	6.13	99.99
Total....	100.01	....	99.98	....	99.98	....	99.99	....

Size of mesh.	Underflow from classifier 4.		Underflow from classifier 5.		Underflow from classifier 6.		Underflow from classifier 7.	
	Weight.	Cumulative weight.	Weight.	Cumulative weight.	Weight.	Cumulative weight.	Weight.	Cumulative weight.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent*	*Per cent.*	*Per cent.*	*Per cent.*
On 20	....	....	....	....	....	....	....	....
35	9.32	9.32	3.17	3.17	0.69	0.69	0.23	0.23
65	35.41	14.73	25.60	28.77	11.24	11.93	5.35	5.58
100	21.46	66.19	26.96	55.73	26.17	38.10	16.41	21.99
150	18.25	84.44	24.62	80.35	34.18	72.28	33.84	55.83
200	7.58	92.02	10.00	90.35	13.36	85.64	16.55	72.38
Through 200	7.95	99.97	9.63	99.98	14.42	100.06	27.61	99.99
Total....	99.97	....	99.98	....	100.06	....	99.99	....

TABLE 21.—*Results of screen tests of spigot products from nine classifiers fed to tables*—Continued.

Size of mesh.	Underflow from classifier 8.		Underflow from classifier 9.		Overflow from classifier 9.	
	Weight.	Cumulative weight.	Weight.	Cumulative weight.	Weight.	Cumulative weight.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
On 20	....	....			....	....
35	0.10	0.10	....	....	3.81	3.81
65	1.33	1.43	1.04	1.04	0.14	3.95
100	7.34	8.77	4.78	5.82	4.52	8.47
150	24.75	33.52	26.95	32.77	4.38	12.85
200	21.90	55.42	20.31	53.08	2.97	15.82
Through 200	44.57	99.99	46.91	99.99	84.17	99.99
Total....	99.99	....	99.99	....	99.99	....

TABLE 22.—*Sizes of the nine classifiers.*

Classifier number.	Size of top (square).	Depth.	Diameter of spigot-discharge opening.
	Inches.	*Inches.*	*Inches.*
1	16 by 16	24	1
2	21 by 21	26	$\frac{7}{8}$
3	22 by 22	30	$\frac{7}{8}$
4	28 by 28	36	$\frac{7}{8}$
5	30 by 30	46	$\frac{3}{4}$
6	35 by 35	46	$\frac{1}{2}$
7	41 by 41	54	$\frac{3}{8}$
8	48 by 48	54	$\frac{3}{8}$
9	45 by 45	54	$\frac{3}{8}$

TABLE PRACTICE.

ESSENTIAL FACTORS IN TABLE PRACTICE.

FOUNDATIONS.

In ordinary practice the deck of a table has a speed of 200 to 300 strokes a minute. The table surface should be free from vibrations that make ripples in the water and in the table feed, for these cause losses. Hence the table should have a firm foundation. The best support is a solid concrete foundation extending the entire length. In the Joplin district a concrete pier under the head and another under the opposite end of the table have proved satisfactory. These piers have wide bases. A concrete pier under the head only is usually not firm enough, and a table bolted directly to the floor of the mill is bound to vibrate. For best results, the table supports must be rigid and securely bolted to a firm foundation, free from the floor of the mill, unless the floor is of concrete.

HEAD MOTION.

The head motion advances the ore or pulp along the deck of the table, the speed, or number of strokes per minute, being determined by the length of stroke and the size of material treated. Results obtained with various tables in the district indicated that the best speed for classified material passing a $1\frac{1}{2}$-mm. screen was 220 to 240 strokes a minute with a 1-inch to $\frac{3}{4}$-inch stroke for the coarser sands, 240 to 260 strokes a minute with a $\frac{3}{4}$-inch to $\frac{5}{8}$-inch stroke for intermediate sizes, and 260 to 300 strokes a minute with a $\frac{5}{8}$-inch to $\frac{1}{2}$-inch stroke for the finer sizes. The effectiveness of these speeds and lengths of stroke, however, depends on the slope of the table, the amount of wash water used, the quantity of feed, and the character of the ore treated.

FEED TO TABLES.

Perhaps the most important factor in successful table concentration is uniform feed. If the feed is uniform and continuous, the lines of separation vary less, and the quantity of wash water and the slope of the table can be much more easily regulated. To insure a uniform feed, the material fed must be well classified. In the mills of the Joplin district sands or fine materials are usually classified in simple V-shaped boxes, with or without baffles, a column of water under pressure passing through the discharge end or sorting column. The overflow passes over a launder, which has a slight slope, to the next box, the depth of the boxes increasing with the fineness of the material. The proper thickness of pulp or the quantity of water in the feed to a table, especially for the finer sizes, must be determined by the mill foreman.

SLOPE OF TABLES AND QUANTITY OF WASH WATER.

The quantity of wash water and the slope given to a table should be regulated according the quantity and size of the feed. Coarse sand requires more wash water than a fine pulp. Tilting of the table is used more especially for adjusting the line of separation when the feed varies. The function of the wash water is to clean the concentrate before it leaves the table.

PRODUCTS FROM TABLES.

As the specific gravity of galena (7.4 to 7.6) and of zinc blende (3.9 to 4.1) is decidedly higher than that of flint (2.65), separating the lead and zinc minerals from the flint in the coarse and intermediate size sands by treatment on tables is not difficult, but effecting such separation when the pulp consists mostly of slimes or material passing a 200-mesh screen is not so easy. The iron sulphide, which is usually marcasite (specific gravity 4.85 to 4.9), is difficult to separate from the zinc blende in either coarse or fine feeds, because the difference in specific gravity of the two minerals is small.

A clean lead concentrate and a clean zinc concentrate are usually made, the intermediate mixed lead, iron, and zinc concentrate, known as the "return," being returned in most mills to the original table feed.

The middling is the product between the concentrate and the tailing. Although there are exceptions, the usual practice in the Joplin mills is to return the middlings from the tables to the original table feed without regrinding.

The lead and zinc content of the tailings depends on the richness of the feed to the table, the classification and uniformity of the feed, the action of the table, and the size of the material. As a rule, the finer the material treated, the higher will be the assay value of the tailing.

The personal attention given to tables is an important factor in obtaining a high recovery. Tables are by no means "fool proof" and are very sensitive to variations of feed.

RESULTS OF TABLE TESTS.

Results of several table tests in various mills are given to show the character of the table practice in the district. The variations in zinc recovery are partly caused by the factors already enumerated. In all of the tables the zinc content is given in terms of metallic zinc.

Table 23 shows that when the ore fed to tables is more or less "chatty," the zinc mineral not having been entirely liberated from the gangue by crushing, the percentage of recovery is lower than when the ore is "free"—that is, when the zinc mineral breaks cleanly from the gangue when crushed to table size. It is believed that the percentage of recovery from this chatty ore could be raised by regrinding the middlings from the tables, a point that is brought out more fully on a subsequent page.

TABLE 23.—*Results of tests of table sections in various mills.*

Test No.	Number of tables in operation.	Zinc content of feed to tables.	Zinc content of concentrates produced from tables.	Zinc content of tailings from tables.	Percentage of zinc recovered.	Character of ore treated.
		Per cent.	*Per cent.*	*Per cent.*		
1	1	6.35	58.0	3.75	43.8	Chatty.
2	1	4.90	59.2	1.50	71.2	Free.
3	2	2.95	55.6	1.35	55.5	Chatty.
4	2	4.85	57.4	1.70	67.0	Do.
5	2	4.70	57.0	1.55	69.0	Do.
6	2	3.12	54.5	1.00	69.1	Do.
7	2	4.30	56.2	1.35	70.2	Do.
8	2	5.65	57.1	1.62	73.5	Free.
9	2	7.75	57.2	2.25	73.9	Do.
10	3	3.75	52.8	1.20	69.5	Chatty.
11	3	6.75	54.6	1.55	79.4	Free.
12	4	4.65	42.0	1.30	74.3	Chatty.
13	6	3.95	59.3	1.19	71.3	Free.
14	7	4.65	55.0	1.42	71.3	Chatty.
15	9	5.25	52.1	1.00	82.5	Chatty-free.
16	10	4.90	57.5	1.01	80.8	Do.

EFFECT OF SPEED AND LENGTH OF STROKE.

The best speed and length of stroke for a table must be determined by the mill foreman; they will depend on the character of the feed.

To show the effect of wrong speeds and wrong lengths of stroke the results of two tests are given in Table 24 following. The feed passed a 1¼-mm. screen.

TABLE 24.—*Results of tests showing wrong speeds and lengths of stroke.*

	Test 1.	Test 2.
Speed of table, strokes per minute	175	224
Length of stroke, inches	1¼	½
Zinc content of feed, per cent	9.25	7.75
Zinc content of concentrates produced, per cent	52.2	57.2
Zinc content of tailings, per cent	3.05	3.20
Percentage of zinc recovered	71.3	62.2

The percentages of zinc recovery in Table 24 are poorer than the results indicate, for the feed was considerably richer than the average feed to tables, and should have given, in both tests, much higher recoveries with proper operation. In test 1 the speed of the table was too slow and the stroke too long for the size of material treated; in test 2 the stroke was too short for the speed of the table and the size of material treated.

TESTS OF FEEDS AND PRODUCTS FROM SIX TABLES TREATING "FREE" SHEET-GROUND ORE.

In order to determine the actual efficiency of each table in a mill and to find just where losses are taking place, it is necessary to make screen analyses of the feed and table products. To show the importance of making such screen analyses a few results are given.

Tables 25 and 26 show the results of treating a sheet-ground ore over six tables, and Tables 27 and 28 of treating a "chatty" ore over four tables. In both mills the middlings and mixed "return" concentrate products of lead, iron, and zinc from all tables were returned to the original table feed without being reground.

TABLE 25.—*Results of tests with six concentrating tables treating "sheet-ground" ore.*

Table No.	Size of screen, mesh.	Size of openings.		Feed.			Tailing.		
		Mm.	Inch.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
Table 1 (speed, 260; ⅞-inch stroke)	20	0.833	0.0328	....	....	....	....	....	....
	35	.417	.0164	22.75	0.21	1.85	30.80	0.20	7.35
	65	.208	.0082	38.55	.54	7.90	41.95	.26	13.00
	100	.147	.0058	16.60	2.52	14.60	12.35	.72	10.60
	150	.104	.0041	10.65	6.26	25.30	6.45	1.61	12.35
	200	.074	.0029	1.15	7.87	3.45	1.20	2.31	3.30
	<200	.074	.0029	10.30	12.01	46.90	7.25	6.18	53.40
Total or average	....	....	....	100.00	2.64	100.00	100.00	.84	100.00

TABLE 25.—*Results of tests with six concentrating tables treating "sheet-ground" ore*—Continued.

Table No.	Size of screen, mesh.	Size of openings.		Feed.			Tailing.		
		Mm.	Inch.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
Table 2 (speed, 260; ⅞-inch stroke)........	20	....	....	....	....	....	....	....	....
	35	0.417	0.0164	16.70	0.21	1.12	19.7	0.13	2.60
	65	.208	.0082	41.20	.67	8.87	46.6	.24	11.30
	100	.147	.0058	18.50	2.86	16.98	11.8	.46	5.50
	150	.104	.0041	11.55	6.73	24.98	10.1	1.19	12.10
	200	.074	.0029	.75	8.47	2.10	1.2	1.61	1.95
	<200	.074	.0029	11.30	12.66	45.95	10.6	6.26	66.55
Total or average..	....	....	....	100.00	3.11	100.00	100.0	.99	100.00
Table 3 (speed, 240; ¾-inch stroke)........	20	....	....	....	....	....	....	....	....
	35	0.417	0.0164	6.75	0.23	0.46	8.5	0.15	1.67
	65	.208	.0082	36.95	.39	4.36	38.5	.18	9.18
	100	.147	.0058	27.00	1.33	10.88	30.3	.44	17.60
	150	.104	.0041	17.80	5.98	32.19	12.9	1.49	25.44
	200	.074	.0029	1.80	12.32	6.72	3.8	1.98	9.92
	<200	.074	.0029	9.70	15.49	45.39	6.0	4.57	36.19
Total or average..	....	....	....	100.00	3.31	100.00	100.0	.76	100.00
Table 4 (speed, 230; ¾-inch stroke)........	35	0.417	0.0164	....	....	....	....	....	....
	65	.208	.0082	0.25	1.14	0.05	....	....	....
	100	.147	.0058	9.60	.57	.92	15.2	0.34	3.32
	150	.104	.0041	38.70	1.35	8.79	44.45	.62	17.83
	200	.074	.0029	4.95	2.67	2.22	5.85	1.12	4.23
	<200	.074	.0029	46.50	11.28	88.02	34.50	3.35	74.62
Total or average..	....	....	....	100.00	5.95	100.00	100.00	1.55	100.00
Table 5 (speed, 300; ½-inch stroke)........	65	0.208	0.0082	....	....	....	....	....	....
	100	.147	.0058	8.05	1.38	1.63	....	....	....
	150	.104	.0041	29.85	2.13	9.29	32.0	0.73	6.35
	200	.074	.0029	6.45	3.33	3.14	9.7	1.12	2.94
	<200	.074	.0029	55.65	10.56	85.94	58.3	5.72	90.71
Total or average..	....	....	....	100.00	6.84	100.00	100.0	3.68	100.00
Table 6 (speed, 300; ½-inch stroke)........	100	0.147	0.0058	....	....	....	....	....	....
	150	.104	.0041	5.5	1.19	1.01	4.05	0.60	0.67
	200	.074	.0029	3.3	1.51	.75	2.70	.62	.45
	<200	.074	.0029	91.2	6.94	98.24	93.25	3.85	98.88
Total or average..	....	....	....	100.0	6.48	100.00	100.00	3.63	100.00

SUMMARY.

	Speed, strokes per minute.	Length of stroke.	Zinc content of feed.	Zinc content of concentrate.	Zinc content of tailing.	Percentage of zinc recovery.
		Inch.	*Per cent.*	*Per cent.*	*Per cent.*	
Table 1..................................	260	⅞	2.64	61.15	0.84	69.1
Table 2..................................	260	⅞	3.11	59.59	.99	69.3
Table 3..................................	240	¾	3.31	59.38	.76	78.1
Table 4..................................	230	¾	5.95	57.10	1.55	76.1
Table 5..................................	300	½	6.84	57.93	3.68	49.3
Table 6..................................	300	½	6.48	56.68	3.63	47.0
Total..................................	....	....	3.95	59.3	1.19	71.3

DISCUSSION OF RESULTS WITH EACH OF THE SIX TABLES.

Concentrating table 1 treated the coarsest material; its feed contained 2.64 per cent zinc. The screen product between 35 and 65 mesh represented the largest proportion of material treated, or 38.55 per cent of the feed, and contained only 0.54 per cent zinc, or only 7.90 per cent of the total zinc content in the feed to the table. The screen analysis shows that 10.3 per cent of the feed passed the 200-mesh screen, which indicates poor classification. The material finer than 200-mesh contained 12.01 per cent zinc, or 46.90 per cent of the total zinc content in the feed. Evidently a considerable proportion of the zinc in this material passed into the tailings because the table was not adjusted for treating such fine material. The screen product of tailing finer than 200-mesh assayed 6.18 per cent zinc, and represented 53.40 per cent of the total zinc content in the tailing. The total tailing from this table assayed 0.84 per cent zinc, but would have contained considerably less with better classification. The percentage of zinc recovered from this table was 69.1 per cent, which is low for the size of material the table was supposed to treat. The feed was not uniform and the table was crowded at times.

Poor classification of the feed to table 2 is shown by the percentage of slimes present. The largest proportion of zinc loss in the tailing, or 66.55 per cent, was in the material passing the 200-mesh screen. The feed was not uniform.

Table 3 shows, relatively, better classification and a consequently higher zinc recovery. The percentage of material passing the 200-mesh screen was 9.70 per cent, or relatively much less than for tables 1 and 2, as this table was treating the material from the third classifier. The recovery from the finer sizes was also higher, as indicated by the differences in assay value of the corresponding screen products between the feed and tailing.

The recovery with table 4 was good, especially for the material passing the 200-mesh screen. The speed of the table, 230 strokes per minute, was too slow for the size of material that was being treated.

Tables 5 and 6 were supposed to be treating slimes or material finer than 200-mesh. However, table 5 was receiving a feed of which 44.35 per cent would not pass the 200-mesh screen, and table 6 was receiving a fairly uniform slime of which 91.2 per cent was finer than 200-mesh. The respective low recoveries, however, were due rather to too much water in the feed and on the table than to the classification. The pulp was too thin for efficient work.

COMPARISON OF RESULTS WITH THE SIX TABLES.

The average results of the tests given in Table 25 show a total zinc recovery of 71.3 per cent from the six concentrating tables. This recovery is not very good, considering the character of the ore, which was not "chatty." The poorer recoveries with the first two tables were probably due chiefly to poor classification, although the speeds and the lengths of stroke affected the quality of the work.

Speeds and lengths of stroke of tables.	Speed, strokes per minute.	Length of strokes, inches.
Table 1	260	$\frac{7}{8}$
Table 2	260	$\frac{7}{8}$
Table 3	240	$\frac{3}{4}$
Table 4	230	$\frac{3}{4}$
Table 5	300	$\frac{1}{2}$
Table 6	300	$\frac{1}{2}$

A comparison of the data in Table 25 shows that the length of stroke decreased as the size of material decreased. The lengths were approximately correct, but the speeds decreased in the same direction, which is contrary to good practice. If the speeds on the first four tables had been 230, 240, 250, and 260 or 270 per minute, respectively, with strokes of 1 inch, $\frac{7}{8}$ inch, $\frac{3}{4}$ inch, and $\frac{5}{8}$ inch, it is believed that the tables would have shown better recoveries. However, the low efficiency of the first two tables was chiefly caused by rather poor classification and the lack of uniform feed.

Although the results may not show high recoveries, they indicate some of the necessary factors of good table practice; thus, (1) tables 1 and 2 gave poor recoveries owing to poor classification, lack of uniform feed, and wrong speeds; (2) tables 3 and 4 gave better results because of better classification; and (3) tables 5 and 6 gave poor results because there was too much water in the feed, the pulp being too thin.

The screen analysis and assays of the total tailings from the six tables, given in Table 26, show that 15.5 per cent of the tailings passed the 200-mesh screen, and this part assayed 5.07 per cent zinc, representing 65.78 per cent of the total loss in the tailings. These figures indicate that the pulp of this size (less than 200-mesh) should be of the proper thickness when treated on slime tables to effect the best commercial saving, and that the classification should be such that the finer sizes of material are found on the tables treating fine material and not on the tables treating the coarse sands.

TABLE 26.—*Results of analyses of total tailings from all six tables.*

Size of mesh.	Size of openings.		Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
	Mm.	Inches.			
20	0.833	0.0328			
35	.417	.0164	16.9	0.36	5.03
65	.208	.0082	32.7	.21	5.74
100	.147	.0058	16.5	.47	6.50
150	.104	.0041	15.50	.99	12.80
200	.074	.0029	2.9	1.92	4.15
<200	.074	.0029	15.5	5.07	65.78
Total or average			100.0	1.19	100.00

TESTS OF FEEDS AND PRODUCTS OF FOUR TABLES TREATING "CHATTY" ORE IN MIAMI DISTRICT.

In Table 27 the results represent the treatment of a "chatty" ore on four concentrating tables, not a "free" ore as in Table 25, as the zinc mineral was not entirely liberated from the gangue before treatment. This ore also carried much lead and iron sulphides that reduced the grade of the zinc concentrates. Moreover, the gangue material was entirely different, and the comparatively low zinc assays of the finer screen products were probably due to the gangue containing considerable "soapstone," which seemed to break into very fine pieces that mixed with the finer particles of ore.

TABLE 27.—*Results of tests with four concentrating tables treating "chatty" ore.*

[Miami District.]

Table No.	Size of screen, mesh.	Size of openings.		Feed.			Tailing.		
		Millimeters.	Inches.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
Table 1 (speed, 252; 1-inch stroke).	10	1.651	0.065	....	....	....	....	....	....
	20	.833	.0328	14.5	11.45	15.5	28.65	1.75	29.0
	35	.417	.0164	40.8	13.25	50.65	49.5	2.00	57.3
	65	.208	.0082	24.1	10.17	22.9	10.9	.90	5.65
	100	.147	.0058	9.4	6.67	5.9	3.3	.55	1.05
	150	.104	.0041	5.3	5.12	2.5	2.35	.70	.95
	200	.074	.0029	2.3	4.70	1.0	1.5	.95	.85
	<200	.074	.0029	3.6	4.45	1.55	3.8	2.35	5.2
Total or average..	....	....	....	100.0	10.65	100.00	100.00	1.73	100.00
Table 2 (speed, 236; 1⅛-inch stroke).	10	1.651	0.065	....	....	....	....	....	....
	20	.833	.0328	0.5	2.65	0.4	....	....	....
	35	.417	.0164	12.45	3.45	13.2	16.55	2.00	34.3
	65	.208	.0082	45.50	2.90	40.65	40.10	.65	27.0
	100	.147	.0058	12.60	3.25	12.6	20.70	.55	11.8
	150	.104	.0041	15.35	3.32	15.7	12.25	.52	6.6
	200	.074	.0029	7.35	4.00	9.05	4.75	1.22	6.0
	<200	.074	.0029	6.25	4.35	8.4	5.65	2.45	14.3
Total or average..	....	....	....	100.00	3.25	100.00	100.00	.96	100.0
Table 3 (speed, 245; 1¼-inch stroke).	20	0.833	0.0328	....	....	....	....	....	....
	35	.417	.0164	1.3	2.05	1.2	12.65	0.95	14.2
	65	.208	.0082	15.5	.95	6.9	17.35	.37	7.65
	100	.147	.0058	33.0	1.42	21.9	21.90	.37	9.65
	150	.104	.0041	10.4	1.85	9.0	21.20	.50	12.60
	200	.074	.0029	13.4	2.55	16.0	7.20	.65	5.60
	<200	.074	.0029	26.4	3.65	45.0	19.70	2.15	50.30
Total or average..	....	....	....	100.0	2.14	100.0	100.00	.84	100.00
Table 4 (speed, 245; 1-inch stroke).	35	0.417	0.0164	....	....	....	....	....	....
	65	.208	.0082	3.0	0.40	0.7	4.0	0.30	1.4
	100	.147	.0058	21.6	.75	9.0	29.6	.40	14.0
	150	.104	.0041	30.1	1.15	19.2	21.4	.40	10.2
	200	.074	.0029	23.2	2.45	31.7	15.6	.70	13.0
	<200	.074	.0029	22.1	3.20	39.4	29.4	1.75	61.4
Total or average..	....	....	....	100.00	1.80	100.00	100.0	.84	100.0

TABLE 27.—*Results of tests with four concentrating tables testing "chatty" ore*—Contd.

SUMMARY.

Table.	Speed, strokes per minute.	Length of stroke.	Zinc content of feed.	Zinc content of concentrates.	Zinc content of tailing.	Percentage of zinc recovery.
		Inches.	*Per cent.*	*Per cent.*	*Per cent.*	
Table 1	252	1	10.65	40.86	1.73	87.5
Table 2	236	$1\frac{1}{8}$	3.25	43.15	.96	72.0
Table 3	245	$1\frac{1}{4}$	2.14	43.74	.84	62.1
Table 4	245	1	1.80	42.11	.84	54.4
Total or average	....	....	4.65	42.00	1.30	7[illegible].3

DISCUSSION OF RESULTS WITH EACH TABLE.

The screen analysis of the feed to the first table shows a better classification than for the feed to the first table in the mill treating "free" ore. The assay values of the screen products are higher in the coarser sizes of material than in the finer, probably because a large quantity of middling and "return," the latter a mixed concentrate of lead, iron, and zinc, was being returned to the feed. The tailing products on the 20 and 35 mesh screens from this table assayed 1.75 and 2.00 per cent zinc respectively, whereas the zinc content of the next finer screen products was much lower. Evidently a considerable part of this middling needs to be reground before the zinc mineral in it can be recovered on tables, as a comparison of the 35-mesh screen products in the tailings from the first three tables, with the total tailings from all tables, will show. In each test the zinc content of the material coarser than 35 mesh is considerably higher than that of the screen products finer than 35 mesh, except in the material that passed the 200-mesh screen. Examination of this size material with a magnifying glass gave additional proof. In the tailing from table 1 the combined products on the 20 and the 35 mesh screens contained 86.3 per cent of the total zinc in the tailing, and a much lower recovery would have resulted except for the high zinc tenor of the feed. Because of this high zinc content, and with fairly good classification, table 1 showed a zinc recovery of 87.5 per cent.

The classification of the feed to the second table was reasonably good in view of the character of the ore. The relatively high zinc content in the coarsest screen product of the tailing, representing 34.3 per cent of the total loss from the table, shows the presence of chatty material, and indicates, as in table 1, the necessity of regrinding middlings from tables treating such material. The length of stroke was too long for the size of material treated. In view of the low zinc content of the feed and the character of the ore the percentage of recovery was fairly good.

The low percentage of recovery with the third table resulted from poor classification, as shown by the proportion of slimes, 26.4 per cent, in the feed to this table. Most of the loss in the tailing was in the material passing the 200-mesh screen, which represented 50.3 per cent of the total zinc in the tailing. The length of stroke was too long for the size of material treated.

As the feed to the fourth table assayed 1.80 per cent zinc, a relatively high recovery was not possible. The classification of the feed was not of the best. If the classification of the feed to table 3 had been such that a considerable part of the material passing the 200-mesh screen had gone to table 4, the writer believes that better work would have been done by both tables. Still better results would have followed treating the slimes on slime tables with a proper thickness of pulp. Also, the length of stroke of this table was too long and the speed too slow for the size of material treated.

COMPARISON OF RESULTS WITH THE FOUR TABLES.

The chief points brought out in the results just described are (1) the necessity of regrinding middlings, (2) the necessity of good classification of feeds, and (3) the use of speeds and lengths of stroke best suited to the material treated. As these points are essential to good table practice, more attention should be paid to them by the operators or mill foremen.

The total percentage of zinc recovery made from the four tables, 74.3, was fairly good considering the character of the ore. The concentrates produced contained only 42.0 per cent zinc, but were retreated over separate tables later.

Although the relatively poor work of each table was due mostly to improper classification, wrong speeds and lengths of stroke, as shown by Table 27, contributed to the low recoveries.

The screen analysis of the total tailings from all four tables, presented in Table 28, shows that the screen products on the 20 and 35 mesh screens contained 1.59 and 1.90 per cent zinc, respectively, and that the combined zinc content of these two products represented 53.25 per cent of the total zinc contained in the tailings, and 39.0 per cent of the total material discarded as tailings.

TABLE 28.—*Results of analyses of total tailings from all four tables.*

Size of screen, mesh.	Size of openings.		Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
	Mm.	Inches.			
10	1.651	0.065	....	....	....
20	.833	.0328	10.2	1.59	11.15
35	.417	.0164	28.8	1.90	42.1
65	.208	.0082	18.2	.80	11.15
100	.147	.0058	12.3	.53	5.15
150	.104	.0041	9.6	.50	4.3
200	.074	.0029	5.6	.80	3.85
<200	.074	.0029	15.3	1.90	22.30
Total or average			100.0	1.30	100.00

If the table section of the mill is assumed to have produced 100 tons of tailings a day, the two screen products on 20 and 35 mesh screens would have yielded 39 tons of tailings assaying 1.82 per cent zinc, the calculated assay value of the two combined products. As this 1.82 per cent represents metallic zinc, it would figure 3.64 per cent in terms of 50 per cent zinc concentrates. The quantity of 50 per cent zinc concentrates contained in the 39 tons of tailings would be 1.419 tons. If this material had been reground and treated over separate tables, a 60 per cent recovery should have been possible, or a saving of 0.851 ton of concentrate. If only 25 tons of tailings was produced from the tables there would have been a saving of 0.213 ton from these 25 tons, and if 50 tons had been treated, a saving of 0.425 ton. This simple example, one of several that might be figured out, shows that a few changes in the flow sheet of a mill may pay for themselves within a short time.

TESTS WITH 10 TABLES TREATING FAIRLY "FREE MILLING" ORE.

In Table 29 are the results obtained from the table section of a mill treating an ore of a fairly free character over 10 concentrating tables. The overflow from the last classifier, consisting mostly of slimes, was returned to the settling tanks. Since the tests were made, however, one Dorr thickener and four additional tables have been added to take care of the slimes overflowing from the settling tanks and classifiers in the mill.

TABLE 29.—*Results of tests with 10 concentrating tables treating rather free milling ore.*

Table No.	Size of screen, mesh.	Size of openings.		Feed.			Tailing.		
		Millimeters.	Inches.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
Table 1 (speed 239; 1-inch stroke).	10	1.651	0.065	0.40	0.98	0.10	....	....	....
	20	.833	.0328	.87	.72	.15	0.58	0.41	0.3
	35	.417	.0164	44.18	1.08	9.90	52.85	.26	15.6
	65	.208	.0082	35.06	4.86	35.25	33.68	.52	19.95
	100	.147	.0058	10.47	13.87	30.05	4.37	1.75	8.7
	150	.104	.0041	1.41	16.68	4.9	.85	2.22	2.15
	200	.074	.0029	5.13	12.90	13.7	.92	3.30	3.45
	<200	.074	.0029	2.48	11.58	5.95	6.75	6.49	49.85
Total or average..	....	....	....	100.00	4.83	100.00	100.00	.88	100.00
Table 2 (speed, 242; 1-inch stroke).	35	0.417	0.0164	36.55	0.75	5.95	45.20	0.35	23.80
	65	.208	.0082	45.61	3.30	32.8	43.14	.55	35.70
	100	.147	.0058	10.79	16.30	38.3	7.49	2.05	23.10
	150	.104	.0041	1.43	20.30	6.35	1.83	2.10	5.78
	200	.074	.0029	3.54	14.80	11.45	.89	2.40	3.24
	<200	.074	.0029	2.08	11.40	5.15	1.45	3.85	8.38
Total or average..	....	....	....	100.00	4.70	100.00	100.00	10.66	100.00

TABLE 29.—*Results of tests with 10 concentrating tables treating rather free milling ore*—Continued.

Table No.	Size of screen, mesh.	Size of openings.		Feed.			Tailing.		
		Millimeters.	Inches.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
Table 3 (speed, 242; $1\frac{1}{8}$-inch stroke).	35	0.417	0.0164	15.73	0.51	1.6	25.98	0.26	8.6
	65	.208	.0082	47.87	1.44	13.8	56.26	.72	51.7
	100	.147	.0058	20.53	10.80	44.3	12.07	1.60	24.7
	150	.104	.0041	3.96	12.50	9.9	.75	1.81	1.7
	200	.074	.0029	7.95	13.38	21.25	3.47	1.64	7.3
	<200	.074	.0029	3.96	11.60	9.15	1.47	3.15	6.0
Total or average	....	....	....	100.00	5.0	100.00	100.00	.78	100.0
Table 4 (speed, 243; $\frac{7}{8}$-inch stroke).	35	0.417	0.0164	4.21	0.36	0.3	7.59	0.26	3.45
	65	.208	.0082	40.04	.98	8.2	58.82	.36	37.30
	100	.147	.0058	30.75	4.55	29.25	25.14	.83	36.70
	150	.104	.0041	11.69	11.10	27.1	4.89	1.34	11.55
	200	.074	.0029	7.33	12.59	19.3	1.97	1.34	4.65
	<200	.074	.0029	5.98	12.70	15.85	1.59	2.27	6.35
Total or average	....	....	....	100.00	4.78	100.00	100.00	0.57	100.00
Table 5 (speed, 245; 1-inch stroke).	35	0.417	0.0164	1.00	0.74	0.15	1.39	0.45	0.60
	65	.208	.0082	23.89	.52	2.50	34.51	.40	13.40
	100	.147	.0058	39.99	3.62	29.00	42.20	.80	32.70
	150	.104	.0041	15.46	6.40	19.8	13.46	1.00	13.1
	200	.074	.0029	11.36	10.31	23.45	5.02	1.25	6.1
	<200	.074	.0029	8.30	15.10	25.1	3.42	2.50	34.1
Total or average	....	....	....	100.00	5.0	100.00	100.00	1.03	100.00
Table 6 (speed, 245; $\frac{3}{4}$-inch stroke).	65	0.208	0.0082	5.04	0.55	0.53	18.61	0.47	5.8
	100	.147	.0058	27.44	1.10	5.79	62.42	1.32	55.0
	150	.104	.0041	28.78	3.70	20.43	6.26	2.58	10.80
	200	.074	.0029	21.72	7.70	32.10	6.16	2.88	11.85
	<200	.074	.0029	17.02	12.60	41.15	6.55	3.78	16.55
Total or average	....	....	....	100.00	5.22	100.00	100.00	1.50	100.00
Table 7 (speed, 270; $\frac{9}{16}$-inch stroke).	35	0.147	0.0164	0.16	1.60	0.06	0.45	0.72	0.25
	65	.208	.0082	2.58	1.38	.70	5.02	.51	2.25
	100	.147	.0058	43.82	2.45	21.40	40.51	.70	24.45
	150	.104	.0041	20.57	4.25	17.42	27.48	1.03	24.45
	200	.074	.0029	19.67	7.30	28.64	14.26	1.55	19.10
	<200	.074	.0029	13.18	12.10	31.78	12.28	2.78	29.5
Total or average	....	....	....	100.00	5.01	100.00	100.00	1.16	100.00
Table 8 (speed, 270; $\frac{9}{16}$-inch stroke)	35	0.417	0.0164	0.45	3.85	0.35	1.37	1.15	1.15
	65	.208	.0082	1.26	3.13	.80	.78	.40	.20
	100	.147	.0058	29.44	1.22	7.1	14.35	.40	4.20
	150	.104	.0041	35.21	3.76	26.1	35.07	.90	23.05
	200	.074	.0029	18.89	6.69	24.9	29.53	1.45	31.30
	<200	.074	.0029	14.75	14.00	40.75	18.90	2.90	40.10
Total or average	....	....	....	100.00	5.07	100.00	100.00	1.37	100.00
Table 9 (speed, 274; $\frac{5}{8}$-inch stroke)	35	0.417	0.0164	1.04	4.70	0.94	....	....	....
	65	.208	.0082	.92	2.80	.49	....	....	....
	100	.147	.0058	21.68	3.35	14.05	8.24	0.50	2.45
	150	.104	.0041	40.73	4.30	33.90	32.59	.60	11.75
	200	.074	.0029	26.92	6.10	31.76	34.17	1.50	30.90
	<200	.074	.0029	8.71	11.20	18.86	25.00	3.64	54.90
Total or average	....	....	....	100.00	5.17	100.00	100.00	1.66	100.00

TABLE 29.—*Results of tests with 10 concentrating tables treating rather free milling ore*—Continued.

Table No.	Size of screen, mesh.	Size of openings.		Feed.			Tailing.		
		Millimeters.	Inches.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
Table 10 (speed, 276; ¾-inch stroke)	65	0.208	0.0082	0.70	5.99	0.75	1.86	1.72	1.7
	100	.147	.0058	3.78	1.24	.85	4.58	.51	1.25
	150	.104	.0041	27.08	1.44	7.0	13.01	.62	4.3
	200	.074	.0029	42.15	5.30	40.2	38.91	1.60	32.65
	<200	.074	.0029	26.29	10.82	51.2	41.64	2.75	60.1
Total or average	....	....	....	100.00	5.55	100.00	100.00	1.90	100.00

SUMMARY.

Table No.	Speed, strokes per minute.	Length of stroke.	Zinc content of feed.	Zinc content of concentrate.	Zinc content of tailing.	Percentage of zinc recovery.
		Inches.	*Per cent.*	*Per cent.*	*Per cent.*	
Table 1	239	1	4.83	58.20	0.88	83.1
Table 2	242	1	4.70	61.80	.66	86.9
Table 3	242	1⅛	5.00	58.78	.78	85.8
Table 4	243	⅞	4.78	53.92	.57	89.0
Table 5	245	1	5.00	58.00	1.03	80.8
Table 6	245	¾	5.22	54.05	1.50	73.3
Table 7	270	9/16	5.01	55.44	1.16	78.5
Table 8	270	9/16	5.07	45.16	1.37	75.0
Table 9	274	⅝	5.17	49.60	1.66	70.2
Table 10	276	¾	5.55	50.82	1.90	68.3

GENERAL CONSIDERATIONS OF TABLE PRACTICE.

In several mills in the Joplin district the table practice is good; in most mills, however, attention is paid to the grade of concentrates produced rather than to the work of each table. If the essentials of good table practice were carried out, a higher recovery of products of the desired grades would be possible in many mills. Detail tests made from time to time would show what each table is doing. The following considerations bearing on higher recoveries by tables seem to be worth mentioning.

CAPACITY OF TABLES.

The capacity of a table depends on the rate at which the ore travels over the deck, and therefore chiefly on the length of stroke and the number of strokes per minute. In general, the capacity ranges from 4 to 40 tons in 24 hours, according to the fineness or coarseness of the feed; "roughing" tables usually have two or three times the capacity of ordinary sand tables treating the same kind of material.

The main bulk of the sand that reaches the tables is the undersize of the dewatered tailings from the rougher jigs and that in the overflow waters from the other jigs. A common method of dewatering the tailings from the rougher jigs is with a trommel placed at right angles to the end of the jig. The tailings flow off an apron at the end of the jig and fall on the outside of the revolving screen. The undersize is washed through by the water in the tailings. A spray of water plays on the oversize before it leaves the trommel and forces more of the undersize through the screen openings, increasing somewhat the screen efficiency. These dewatering trommels are 4 to 5 feet in diameter, with screen openings of usually 1½ to 2 mm., and revolve in the direction of the flow, at speeds of 1½ to 4 revolutions per minute.

The efficiency of the dewatering trommel, or the proportion of the feed that passes the openings, is 40 to 60 per cent. This low efficiency results from the amount of undersize, depending almost entirely upon the amount of water used to force it through. Having determined the efficiency of the dewatering screen, and knowing approximately the tonnage passing over the rougher jig, one can estimate fairly closely the quantity of undersize from this screen that passes into the sand tanks.

In mills where dewatering screens are used, it would seem advisable to determine their efficiency, as this varies widely and a considerable proportion of the table-size material containing zinc may pass over the screen with the coarse tailings. If such be the case, increased screening efficiency and better saving of the values can readily be effected.

Screen analyses have shown that the tailings from rougher jigs before dewatering usually contain 10 to 30 per cent of table-size material, according to the character of the ore and the size to which it has been crushed before jigging.

The results obtained in treating classified material on nine concentrating tables in actual practice in a mill treating sheet-ground ore are shown in Table 30. The table shows the rate of feed, the percentage of zinc recovery, and other data, for each concentrating table.

TABLE 30.—*Results of tests showing tonnage of a classified feed in mill treating sheet-ground ore on nine concentrating tables.* [a]

[Covering a period of 75 hours actual running.]

Table No.	Zinc content of— Feed.	Tailing.	Concentrate.	Quantity of zinc concentrate produced.	Quantity of zinc in feed.	Quantity of zinc in concentrate.	Percentage of zinc recovery.	Quantity of tailings.	Total quantity of feed to tables.	Rate of feed, tons per hour.
	Per ct.	*Per ct.*	*Per ct.*	*Tons.*	*Tons.*	*Tons.*		*Tons.*	*Tons.*	
1	4.7	1.00	51.8	7.105	4.582	3.680	80.3	90.395	97.50	0.73
2	4.3	.60	51.7	7.713	4.584	3.988	87.0	98.887	106.60	.795
3	4.0	.75	53.1	5.481	3.582	2.910	82.4	84.059	89.54	.67
4	4.0	.70	53.7	3.654	1.990	1.662	83.5	46.106	49.76	.37
5	4.1	1.10	53.2	3.653	2.602	1.943	74.7	59.807	63.46	.47
6	4.0	1.10	51.3	1.827	1.265	.938	74.1	29.803	31.63	.235
7	4.8	2.35	47.9	1.015	.905	.486	53.7	17.835	18.85	.14
8	5.2	3.50	44.2	1.217	1.516	.538	35.5	27.933	29.15	.22
9	5.25	2.20	34.3	1.015	.560	.348	62.1	9.645	10.66	.08
	4.34	1.1	50.5	32.680	21.586	16.493	76.5	464.470	497.15	3.71

[a] For the size of material treated on each table see Table 21 (p. 81).

PRODUCTS FROM THE TABLES.

As previously stated, the usual practice in the mills of the Joplin district is to make a clean lead concentrate and a clean zinc concentrate and to return the intermediate mixed lead, iron, and zinc concentrate to the original table feed. It is believed that a better practice would be to save this product and treat it separately from time to time, as under present practice the iron sulphide accumulates and must eventually contaminate the lead or zinc concentrates, unless eliminated in some way. If the lead content of the ore is small, the iron sulphide might better go with the lead concentrate rather than be returned to the feed.

Middlings from tables are usually returned to the original table feed, although in a few mills they are treated on separate tables. Returning these middlings to the feed cuts down the relative capacity of each table and lowers their efficiency, especially if the table middlings are not reground before being returned. When the middlings consist of free mineral and gangue, already a concentrated product, it would seem advisable to treat this product on separate tables. By treating the middlings from tables separately, the quantity of middlings produced from the primary tables is cut down, and the relative capacity of the tables increased.

STAGE CONCENTRATION.

Where the ratio of concentration is as high as it is in the mills of this district, the use of a "roughing" and "cleaning" system of concentration by tables would seem preferable in order to assure high recoveries. Such a change would require the use of two or more roughing tables of relatively large capacity and the treatment of the enriched product from these tables on the usual sand tables. A considerable part of the lead might possibly be removed by this treatment before the material reached the sand tables. The chief advantages of the plan would be the higher recoveries obtainable and the greater relative capacity of the sand tables, or possibly a decrease in the number of sand tables used. This plan is being followed in a few of the mills with good results.

Another effective practice in treating fine material on tables is to use, for each classification, two tables, one placed a table's width to the side of the other and about 2 feet lower, both being set on a concrete foundation. From the first and higher table, concentrates only are taken. The mixed product of lead, iron, and zinc from the concentrate end, the middling, and the tailing, from the first table all go by gravity to the feed box of the second table. From this second table concentrates are produced, the middling is reground and sent over separate tables, and the tailing is discarded. The writer had the privilege of testing this system of tabling lead and zinc ores at a mill in Sardinia, Italy, by which high recoveries were obtained.

RETREATMENT OF CONCENTRATES ON TABLES.

When the concentrates from the jig and table sections treating the coarse material are fairly low grade, containing 2 to 3 per cent lead, 4 to 10 per cent iron, and 40 to 50 per cent zinc, the remainder being gangue, it has been found that crushing the material to table size and treating it on tables yields a much better grade of product.

In mills where the concentrates have been re-treated in this way the lead content has been reduced to 0.2 to 0.4 per cent and the iron content to less than 2 per cent. However, in one mill the iron content was reduced from 8 or 9 to only 5 per cent, while the zinc tenor was increased several points. When the content of iron sulphide is high, elimination of that mineral from the zinc-blende concentrates is difficult because of the small difference in specific gravity between the two minerals, whereas the lead content is fairly easy to cut down. To obtain good results with minerals differing but slightly in specific gravity, very close classification is necessary. Therefore, classification by sizing, also known as grading, would seem feasible. The concentrates could be screened to three or more different sizes and the sized products treated on separate tables, the concentration then depending more on the differences in specific gravity of the minerals. The flow sheet for such retreatment on tables would have to be designed with reference to the quantity of concentrates to be treated and the proportionate quantity of the various screen products.

If the concentrates produced from the mill, before retreatment should be high in iron and consequently low in zinc, say about 40 per cent zinc, and the ore from the mine should be fairly uniform, it would probably pay to install a small roasting and magnetic separating plant, rather than to treat the ore on tables. Giving the concentrates a superficial roast and then passing them through magnetic separators has proved satisfactory in the Wisconsin field.[a]

ESSENTIAL CONDITIONS FOR GOOD TABLE PRACTICE.

The essential points to be considered in good table practice can be summed up as follows: (1) The character of the ore, (2) the richness of the feed, (3) the size of the material treated, (4) the quantity of feed, (5) the previous classification of the feed, (6) the uniformity of the feed coming onto the table, (7) the thickness of the pulp treated,

[a] Wright, C. A., Mining and milling of lead and zinc ores in the Wisconsin district, Wis., Tech. Paper 95, Bureau of Mines, 1915, pp. 27-28.

(8) the operating conditions of the table, such as speed, length of stroke, amount of wash water, slope of deck, and firmness of foundations, (9) the character and quantity of the middlings produced and their further treatment, (10) the kind and grade of the concentrates produced, (11) the further treatment of the "returns" or mixed concentrate products of lead, iron, and zinc, (12) the possibilities of stage concentration by tables.

All of these conditions are, of course, dependent on the quantity of ore to be treated, the number of tables available or that can be added, and the personal attention given to the tables by the men in charge.

CHARACTER OF OVERFLOW FROM SETTLING TANKS.

Anyone familiar with the mills in the Joplin district knows that the practice has been to produce the smallest amount of fine material possible, as a large proportion of the losses has been in the finest sizes. The coarse tailings from the "rougher" jigs, after being dewatered, are seldom reground for further treatment, but are sent direct to the tailings pile. Consequently, the quantity of slimes is relatively small as compared with that produced by jigs and tables in other parts of the country.

As the treatment of slimes containing zinc blende is now possible by improved concentrating methods, especially by flotation, the production of a somewhat larger proportion of fine material should not be considered such a disadvantage in the milling of the Joplin ores.

LOSSES IN OVERFLOW.

In figuring the percentage of recovery in a mill all of the losses should be included, a part of which is in the overflow waters from the settling tanks. The zinc assays, as a rule, are relatively much higher in the slimes and overflow from the settling tanks than in the table sands and coarser material.

RESULTS OF MILL TEST IN MILL TREATING SHEET-GROUND ORE.

The results of a mill test showing the quantity of zinc lost in the overflow, the relatively higher assays in the finer sizes of the mill tailings, not including the overflow, and the percentage of loss in the various screen products, are presented in Table 31.

TABLE 31.—*Results of a mill test in mill treating sheet-ground ore.*

Screen analysis of mill tailings not including overflow.					Zinc (Zn) content.		
Size of mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
			Per cent.	*Per cent.*	*Per cent.*		
	13.33	0.525	6.89	....	0.30	5.6	....
3	6.680	.263	32.08	38.97	.22	19.3	24.9
6	3.327	.131	26.75	65.72	.20	14.6	39.5
10	1.651	.065	18.40	84.12	.19	9.5	49.0
20	.833	.0328	9.07	93.19	.30	7.4	56.4
35	.417	.0164	2.70	95.89	.56	4.1	60.5
65	.208	.0082	1.80	97.69	1.27	6.3	66.8
100	.147	.0058	.44	98.13	2.93	3.5	70.3
150	.104	.0041	.39	98.52	3.35	3.6	73.9
200	.074	.0029	.21	98.73	5.70	3.3	77.2
<200	.074	.0029	1.27	100.00	6.55	22.8	100.00
Total or average			100.00	....	.37	100.00	....
Check sample			....	....	.41	....	....

		Tons.
Ore treated		1,994
Concentrates produced		35.5
Tailings		1,958.5
Quantity of solids in overflow from settling tanks		34.73
Quantity of tailings, not including material in overflow		1,923.77
Quantity of metallic zinc in—	Per cent.	
Concentrates	59.4	21.087
Tailings without overflow	.41	7.887
Overflow	5.01	1.740
Total quantity of zinc in ore treated		30.714
Percentage of zinc recovery (27.087÷30.714)×100	68.61	

The results of this test, made in a mill treating "sheet-ground" ore, show a zinc recovery of 68.6 per cent.

As shown by the screen analysis and the zinc assays of the various screen products in the mill tailings, not including the overflow from the settling tanks, 22.8 per cent of the total zinc was in the material finer than 200 mesh. This material represented 1.27 per cent of the total weight of the mill tailings. The quantity of solids in the overflow for the quantity of ore treated (1,994 tons) was 34.73 tons, with an assay value of 5.01 per cent zinc. The quantity of zinc in the overflow, therefore, was 1.740 tons. As the mill tailings contained 7.887 tons of zinc, the total quantity of zinc lost for the tonnage treated was 9.627 tons. From these figures, the percentage of the total loss represented by the overflow was $(1.740 \div 9.627) \times 100 = 18.1$ per cent. If the percentage of recovery made by the mill not including the values lost in the overflow from the settling tanks were calculated, the result would show a zinc recovery of 72.8 per cent. In other words, the apparent recovery would be 4.2 per cent better if the values in the overflow were not included in the total zinc lost.

RESULTS OF SCREEN ANALYSIS OF SOLIDS IN OVERFLOW FROM SETTLING TANKS AT MILL TREATING SHEET-GROUND ORE.

To show more fully the values in the various sizes of solids in the overflow from settling tanks at a representative mill treating sheet-ground ore, the writer presents the following screen analyses and the zinc assays of the different screen products.

TABLE 32.—*Screen analysis of solids in overflow from settling tank of mill treating sheet-ground ore.*

Size of screen mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	Zinc content.		
	Mm.	Inches.			By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
			Per cent.	*Per cent.*	*Per cent.*		
100	0.147	0.0058	8.19	....	5.00	6.05	....
150	.104	.0041	7.60	15.79	4.50	5.06	11.11
200	.074	.0029	8.22	24.01	5.20	6.32	17.43
<200	.074	.0029	75.99	100.00	7.35	82.57	100.00
Total average			100.00	....	6.76	100.00	....
Check sample			....	....	6.60	....	....

RESULTS OF TESTS OF OVERFLOW FROM SETTLING TANKS AT SEVERAL MILLS.

Table 33 shows the weight of solids per gallon and the lead and zinc assays of solids in the overflow from the settling tanks of several mills.

TABLE 33.—*Data on overflow from settling tanks at seven different mills.*

Mill No.	Quantity of solids per gallon.[a]	Ratio of water to solids.	Per cent of solids in overflow.	Assay of solids.		Remarks.
				Lead (Pb).	Zinc (Zn).	
	Grams.			*Per cent.*	*Per cent.*	
1	11.67	323:1	0.309	1.20	5.01	Single tanks.
2	12.67	298:1	.335	2.41	7.04	Do.
3	13.85	272:1	.365	1.45	5.25	Do.
4	9.35	405:1	.246	.37	6.38	Double tanks.
5	25.33	161:1	.619	.14	1.13	Single tanks.
6	9.66	391:1	.254	2.78	6.60	Double tanks.
7	9.30	406:1	.245	1.90	8.40	Do.

[a] 1 gallon contains 3,785 c. c., or 3,785 grams of water (specific gravity, 1.00); specific gravity of solids was taken as 2.7.

METHOD OF CALCULATING VALUE OF LOSSES AND PROBABLE RECOVERY.

As the table shows, the percentage of solids in the overflow waters was low. In order to treat the contained slimes these overflows would have to be thickened to a fairly thick pulp. In figuring the zinc loss in the overflow, the number of gallons of overflow a minute would have to be measured at each individual mill. The overflow from the settling tanks is about 700 to 1,200 gallons a minute, depending, of course, on the tonnage treated and the quantity of water used. If 1,000 gallons a minute be taken as an average, an approximate figure for the zinc values lost during a period of 10 hours can be determined for the mills represented in the preceding table, as shown in Table 33 following.

TABLE 34.—*Estimated zinc losses in overflow at seven mills, calculated from data in Table 33.*

[Zinc values figured on the basis of 1,000 gallons per minute overflow.]

Mill No.	Quantity of solids in overflow for 10 hours.		Zinc content.	Quantity of zinc in overflow for 10 hours.	Quantity of "50 per cent zinc" concentrates in overflow for 10 hours.
	Kilograms.	Pounds.[a]			
			Per cent.	*Pounds.*	*Pounds.*
1	7,002	15,437	5.01	773	1,546
2	7,602	16,759	7.04	1,159	2,318
3	8,310	18,320	5.25	962	1,924
4	5,598	12,341	6.38	787	1,574
5	13,998	30,860	1.13	349	698
6	5,796	12,778	6.60	843	1,686
7	5,580	12,301	8.40	1,033	2,066

[a] 1 pound is figured as equal to 453.6 grams.

After computing the quantities of zinc lost in the overflow for 10 hours, and figuring that a saving of 50 to 80 per cent is possible by flotation or some other method, values can be calculated in dollars and cents. Assuming that the product obtained, after treatment, will have a value of $30, $40, $50, or $60 per ton for 50 per cent zinc concentrates, and figuring these values on a 50 and an 80 per cent zinc recovery, one obtains the figures as shown in Table 35.

TABLE 35.—*Value of "50 per cent" zinc concentrates in overflow during 10 hours from various mills.*

Mill No.	Quantity of "50 per cent zinc" concentrates in 10 hours.[a]	Value at $30 per ton.	Value at $40 per ton.	Value at $50 per ton.	Value at $60 per ton.
	Pounds.				
1	1,546	$23.19	$30.92	$39.65	$46.38
2	2,318	34.77	46.36	57.95	69.54
3	1,924	28.96	38.48	48.10	57.72
4	1,574	23.61	31.48	39.35	47.22
5	698	10.47	13.90	17.45	20.94
6	1,686	25.19	33.72	42.15	50.58
7	2,066	30.99	41.32	51.65	61.98

[a] Figures are those given in Table 34.

TABLE 35.—*Value of "50 per cent" zinc concentrates in overflow during 10 hours from various mills*—Continued.

VALUES BASED ON 50 PER CENT SAVING.

Mill No.	Quantity of "50 per cent zinc" concentrates in 10 hours, pounds.	Value at $30 per ton.	Value at $40 per ton.	Value at $50 per ton.	Value at $60 per ton.
1	1,546	$11.59	$15.46	$19.82	$23.19
2	2,318	17.38	23.18	28.97	34.77
3	1,924	14.48	19.24	24.05	28.96
4	1,574	11.80	15.74	19.67	23.61
5	698	5.23	6.98	8.72	10.47
6	1,686	12.59	16.86	21.07	25.19
7	2,066	15.49	20.66	25.82	30.99

VALUES BASED ON 80 PER CENT SAVING.

1	1,546	$18.55	$24.74	$31.72	$37.10
2	2,318	27.82	39.09	46.36	55.63
3	1,924	23.17	30.78	38.48	46.18
4	1,574	18.89	25.18	31.48	37.78
5	698	8.38	11.17	13.96	16.75
6	1,686	20.15	26.98	33.72	40.46
7	2,066	24.79	33.06	41.32	49.58

At No. 1 mill there was 1,546 pounds of 50 per cent zinc concentrates in the overflow during 10 hours. This quantity, figured at $30, $40, $50, and $60 per ton, gives values of $23.19, $30.92, $39.65, and $46.38. If a 50 per cent saving be assumed, the values become $11.59, $15.46, $19.82, and $23.19. The net profit will be the difference between these values and the cost of treatment. The figures given are based on a 10-hour day; if the mills were in use 20 hours or 24 hours a day the values would be proportionately greater.

These figures bring out the possibilities of saving more of the minerals now going to waste and show one of the reasons why 30 to 40 per cent of the zinc in the ore treated in the mills of the Joplin district is lost. Hence the management of each mill should determine what percentage of the total loss from the mill is represented by the overflow from the settling tanks, and, if the loss warrants them, what improvements or changes will make a greater commercial saving. The commercial possibilities for a greater saving of the values contained in the slimes or overflow slimes under the present milling practice will, of course, depend on the quantity of fine material produced, so that, unless a slime-treatment unit or a flotation unit of small enough capacity can be added, any saving on a profitable basis will probably be limited to the larger mills of the district.

However, as has previously been stated, the fact that it is now possible to obtain a relatively high recovery by flotation from slimes containing sulphides of lead and zinc indicates that a greater proportion of the fine material produced by finer grinding could be profitably saved with proper equipment. Especially the middlings from tables and the "chats," or particles containing included mineral, from the jigs, could be ground finer.

POSSIBILITIES OF FLOTATION.

A number of flotation tests of the lead and zinc ores of this district have shown that floating the sulphides is relatively easy. The ore consists mainly of sphalerite, the zinc sulphide, with varying smaller quantities of lead sulphide, galena, and flint being the chief gangue material. Certain problems, however, such as the quantity of material that can be floated without further crushing and the possibilities of finer grinding, must be worked out to insure success on a commercial scale. Therefore, the mill men of this district, because of the difficulties in applying flotation to the treatment of these ores, have on the whole been somewhat reluctant to adopt the process. Although at present flotation does not hold as important a place in the treatment of Joplin ores as in many of the larger copper and zinc mills of the West, yet the writer believes that before long many of the mills will have small flotation units for saving a large proportion of the zinc now going to waste in the fine material. Five or six mills are using flotation successfully, and others are testing it. The grade of flotation concentrates produced by the mills using the flotation process varies from about 55 to 59 per cent zinc.

In the Joplin district the mills are small, their capacity being 100 to 500 tons of ore treated in 10 hours. They are equipped with crusher, rolls, jigs, and tables. At most of them the material that reaches the jigs has passed a ½-inch trommel screen. The middlings or "chats" from the jigs are recrushed for further treatment on sand jigs or tables, or, as is the practice in some mills, returned to the rougher jigs. Middlings from the tables are seldom reground. The first point to be considered, therefore, is how much material now being produced is fine enough to be amenable to flotation, or in other words, what percentage of the total tonnage treated is crushed to a suitable size for flotation and what proportion of this size can be saved for retreatment.

PROPORTION OF MATERIAL OF SUITABLE SIZE FOR FLOTATION.

RESULTS OF TEST.

To show more clearly the quantity of material available and of suitable size for flotation the results of a simple mill test are presented in Table 36 following. The ore treated was from a hard-ground mine, better known as sheet-ground ore, consisting chiefly of flint as the gangue with a small percentage of zinc and practically no lead.

TABLE 36.—*Results of sizing test of tailings in a mill treating sheet-ground ore.*

Screen analysis of mill tailings not including overflow.			Weight of screen products.	Cumulative weight.	Zinc content.		
Size of mesh.	Size of openings.				By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
			Per cent.	*Per cent.*	*Per cent.*		
....	13.33	0.525	6.89	....	0.30	5.6	....
3	6.680	.263	32.08	38.97	.22	19.3	24.9
6	3.327	.131	26.75	65.72	.20	14.6	39.5
10	1.651	.065	18.40	84.12	.19	9.5	49.0
20	.833	.0328	9.07	93.19	.30	7.4	56.4
35	.417	.0164	2.70	95.89	.56	4.1	60.5
65	.208	.0082	1.80	97.69	1.27	6.3	66.8
100	.147	.0058	.44	98.13	2.93	3.5	70.3
150	.104	.0041	.39	98.52	3.35	3.6	73.9
200	.074	.0029	.21	98.73	5.70	3.3	77.2
<200	.074	.0029	1.27	100.00	6.55	22.8	100.0
Total			100.00	....	*a* .37	100.0	....
Check sample			....	....	.41	....	....

a Calculated.

		Tons.
Ore treated		1,994
Concentrates produced		35.5
Tailings		1,958.50
Solids in overflow from settling tanks		34.73
Tailings, not including solids in overflow		1,923.77
Metallic zinc in—	Per cent.	
Concentrates	59.40	21.087
Tailings without overflow	.41	7.887
Overflow	5.01	1.740
Total quantity of zinc in ore treated		30.714
Percentage of zinc recovery (21.087+30.714) × 100	68.6	

DISCUSSION OF RESULTS OF TEST.

The results of the tests in the preceding table indicate a zinc recovery of 68.6 per cent. As shown by the screen analysis of the mill tailings, not including the solids in the overflow from the settling tanks, 2.31 per cent passed the 65-mesh screen, a product fine enough for flotation. The quantity treated during this test was 1,994 tons, containing 44.44 tons of material passing the 65-mesh screen. The calculated zinc content of this size material was 5.24 per cent, or 2.328 tons. On the assumption that the overflow is suitable to flotation, there are 34.73 tons of solids containing 5.01 per cent zinc, or 1.74 tons of zinc. The sum of these two products makes a total of 4.068 tons of zinc in the material suitable to flotation, or 79.17 tons with a calculated content of 5.14 per cent zinc. If this material would be available and if 80 per cent of its zinc content could be recovered by flotation, 3.256 tons of zinc would be saved. The total zinc content of the 1,994 tons of ore treated was 30.714 tons. Adding to the 21.087

tons of zinc saved in the concentrate the 3.256 tons that could be recovered from the 79.17 tons of floatable material by flotation, we have 24.343 tons of zinc saved out of a possible 30.714 tons, or a zinc recovery of 79.2 per cent. In other words, the recovery would be increased from 68.6 to 79.2 per cent, representing an additional saving of 10.6 per cent of the total zinc in the ore.

The above figures bring out the possibilities of marked savings by flotation with a relatively large tonnage. Unfortunately, the average daily tonnage treated by the mills is considerably less than 1,994 tons, the tonnage treated during the time the test was being made. It is reasonable to believe, however, that the mills treating 500 to 1,000 tons of ore daily could install a small flotation unit at a profit, the size of the unit depending on the quantity of fine material produced.

In milling soft-ground ore generally the proportion of fines is much larger than with hard-ground ore; hence enough fine material would be produced in a mill treating 200 to 300 tons a day to operate a 20 to 30 ton flotation unit.

DIFFICULTY OF GRINDING TO SIZES SUITABLE FOR FLOTATION.

The next step in determining the possibilities of flotation for the ores of the Joplin district is the profitable limit of fine grinding, which would be one of the main problems in flotation, as the gangue consists mostly of flint. The question, therefore, is one of fine grinding rather than of floating the mineral, at least in mills where not enough material is already available for profitable flotation. Many kinds of fine grinding mills have been tried in this district with more or less discouraging results, as the flint is highly abrasive. In fact, the fine grinding of Joplin ores, either by ball mills or some other means, is still an unsolved problem.

Certain of the mill products, however, would be less resistant and more easily ground. Some of these products are: (1) "chats" from the jigs, (2) tailings from the tables treating fine material, (3) all table middlings that need to be reground before further treatment, and (4) possibly the oversize often discarded from sand screens. The important factors to be considered in the possible treatment of these products by flotation are the cost of fine grinding, the quantity of material to be reground, and the zinc content of the material.

If the material contained 1 per cent metallic zinc, equivalent to 2 per cent of a 50 per cent zinc concentrate, and the concentrate was worth \$50 a ton, its value would be \$1 a ton or \$0.80 a ton on a basis of 80 per cent zinc recovery. For material containing 2 per cent zinc these values would be \$2 a ton and \$1.60 for 80 per cent recovery. In other words, it is possible to figure from the zinc content and the possible zinc recovery what value can be placed on the material to be reground and treated.

The cost of fine grinding would depend upon the type of grinding mills used and on the character and coarseness of the material. The greater the tonnage treated the less would be the unit cost of grinding and subsequent treatment. The possibilities are, therefore, greatly increased when one considers the larger tonnage available for flotation by finer grinding. In addition slimes or fine sands that are at present being treated on slime tables could be treated by flotation, and a higher zinc recovery effected than is usually possible on slime tables.

In view of the demand for high-grade zinc concentrates by smelters, zinc ores containing a comparatively large proportion of lead and iron sulphides would require either differential flotation or retreatment of the concentrates on tables. At a few mills the floatable material is given a preliminary treatment on tables to eliminate the lead and iron, but this requires extra tables and is of doubtful efficacy where the content of iron and lead in the feed is high.

Retreatment of the concentrates on tables seems to be the most feasible method of separating the lead and iron sulphides from the zinc blende and could be cheaply done. A small diaphragm pump, operated in closed circuit, could be used to return the tailings from the tables to the flotation system. This method of treatment has given satisfactory results in some of the local mills. The froth from the concentrates is largely ''killed'' by spraying with water, or by some other effective means, before it reaches the table. The ease with which the froth is killed depends largely upon the oil mixture or frothing agent used.

EXPERIMENTS WITH DIFFERENT OILS.

The composition of the gangue of the Joplin ores varies somewhat as does the mineral content of the ores. Consequently, although many oils are available that give acceptable results, there is no one oil or certain quantity of oil that under the same operating conditions will give the best possible results for all the lead and zinc ore of this district. Hence the best oils or the best mixture to use must be determined by experiment.

The Bureau of Mines, in cooperation with the Missouri geological survey, started flotation experiments in the Joplin district in the latter part of the year 1914 and continued them until the spring of 1915. In the first experiments different oils were tried in varying quantities to decide which oil was most suitable. When this point had been determined, similar tests were made with the addition of sulphuric acid in varying quantities. The effect of temperature was then considered.

The results of these tests indicated that separation of the sulphides from the gangue in ores of this district by flotation is not difficult, and that a fairly good grade of concentrate can be obtained by the use of rougher and cleaner cells. The use of oils with a coal-tar base gave a high recovery, but a low grade of concentrate, a consideragle amount of gangue being entrained in the froth. Other oils gave a higher grade of concentrate, but not as good a recovery under like conditions. The best results were obtained with wood creosote and pine oils. As coal tar and some petroleum products are good collectors and pine oil and other suitable oils from wood distillation are good frothing agents, a mixture of 80 per cent coal tar and 20 per cent pine oil or wood creosote, or some other combination of these oils, would seem to be best suited to the treatment of the Joplin ores by flotation.

Also, several tests were made with mixtures of turpentine and rosin. The rosin was dissolved in the turpentine in proportions of 10, 20, and 30 per cent, respectively. Most of the tests were roughing tests to determine what grades of concentrate and what percentages of recovery could be made with these mixtures. The 20 and 30 per cent mixtures gave the best results, both as to grade and recovery. When 1 to 5 pounds of mixture was used per ton of ore treated in an acid pulp of about 5:1 thickness (water to solids), the rough concentrates varied from 37 to 56 per cent in zinc content, and zinc recoveries from 50 to 80 per cent. In most tests the higher the grade of concentrate obtained the lower was the recovery. However, the relatively low recoveries may be considered good in view of the low zinc content of the heads, which assayed 1.05 to 2.18 per cent zinc. It is believed that with richer feeds much higher recoveries could have been obtained.

TESTS AT SALT LAKE CITY STATION.

To verify the data obtained from this preliminary work, slimes produced in a mill treating ore from one of the sheet-ground mines near Webb City, Mo., were shipped to the experiment station of the Bureau of Mines at Salt Lake City, Utah, where the bureau, in cooperation with the University of Utah, has been conducting flotation experiments. The results of the experiments, which were performed by O. C. Ralston and G. L. Allen, are presented in Tables 37 and 38, following. They show that the sphalerite can be floated from the gangue with comparative ease by using warm solutions and about 1 pound of any suitable oil, either from wood or coal distillation, per ton of ore, and that acidifying the pulp, although it does not seem to be necessary, permits the froth and tailing to separate more quickly. Cold solutions gave as high recoveries as warm solutions, but the grade of product was not as good.

TABLE 37.—*Results of roughing tests in flotation of slimes from sheet-ground ore.*[a]

[Conditions of tests: Assay of slimes tested, 8.54 per cent zinc, 0.51 per cent lead, 0.7 per cent iron; charge used in each test, 500 grams, with 1,500 to 2,500 c. c. of water; temperature of pulp, 60° C., except in test 9, where it was 20° C.; length of agitation, 18 minutes.]

VARYING ACID CONTENT.

Test No.	Acid used, pounds per ton of ore treated.	Oil used.		Weight of froth, grams.	Grade of concentrate, per cent zinc.	Percentage of zinc recovery.	Remarks.
		Kind.	Quantity, pounds per ton of ore treated.				
1	7.36	Pine oil	1.37	105.5	35.9	88.8	
2	14.72	.....do	1.37	96.5	39.3	88.8	
3	36.8	.....do	1.37	95.5	38.9	87.0	
4	73.6	.....do	1.37	100.0	38.8	91.0	Distinct froth line.

VARYING OIL CONTENT.

Test No.	Oil used.		Weight of froth, grams.	Grade of concentrate, per cent zinc.	Percentage of zinc extraction.	Remarks.
	Kind.	Quantity, pounds per ton.				
5	Pine oil	0.5	104.5	36.5	89.5	
6	.....do	1.0	86.5	43.2	87.5	
7	.....do	2.0	103.5	36.2	87.8	
8	.....do	4.0	94.0	39.7	87.5	
9	.....do	1.0	120.5	31.6	89.2	Cold solution used (20° C.).
10	Crude cedar oil	2.5	93.5	40.1	87.8	Rapid separation.
11	Special pine-tar oil	1.8	73.5	49.6	85.5	Very rapid separation.
12	Wood creosote	1.7	93.0	40.8	89.0	Strong froth.
13	Coal creosote	2.0	78.0	46.8	85.8	Strong froth, rapid separation.
14	Eucalyptus oil	1.7	110.0	34.2	88.2	Strong froth.

Samples of each froth from the roughing tests were taken, and the rest was saved for cleaning tests. All of the roughing froths combined made enough material for two cleaning tests, the results of which were as follows:

TABLE 38.—*Results of cleaning tests with froth from roughing tests.*[a]

TEST 1.

Quantity of froth used	grams	500
Assay of head:		
Zinc content	per cent	39.8
Lead content	do	1.3
Quantity of product	grams	366.5
Assay of product:		
Zinc content	per cent	51.7
Lead content	do	2.35
Percentage of recovery	do	95.3

[a] Tests by O. C. Ralston and G. L. Allen.

TEST 2.

Quantity of froth used	grams	479
Assay of head:		
Zinc content	per cent	39.8
Lead content	do	1.3
Quantity of products:		
Froth	grams	346
Middling	do	47
Tailing	do	77
Assay of products:		
Zinc content of—		
Froth	per cent	51.5
Middling	do	13.5
Tailing	do	1.0
Lead content of—		
Froth	do	2.04
Middling	do	2.66
Tailing	do	0.04
Percentage of recovery:		
Froth	do	93.2
Middling	do	3.3
Tailing	do	4.0

DISCUSSION OF RESULTS OF TESTS.

Tests 1 to 4, Table 37, were made to determine the effect of different quantities of acid on the percentage of recovery, the grade of concentrate, and the sharpness of the line of separation between the yellow froth and white tailings, the quantity of oil used being constant. The results indicate that the quantity of acid used has little effect upon either the grade of concentrate or the percentage of recovery, but it was found that the line of separation between concentrate and tailing, as seen through the glass side of a laboratory machine, is much sharper and more quickly defined when a large quantity of acid is used.

The next four tests, Nos. 5 to 8, were run with varying amounts of oil, some acid having been added to permit the experimenters to see the froth and tailing separate more quickly and to clean the surfaces of the sulphide particles. No distinct effect was observed except that the use of 1 pound of oil per ton of material treated seemed to give the highest grade of concentrate. Test 9 was run to parallel test 6, except that a cold pulp at 20° C. was used in place of the warm pulp at 60° C. tests. The lower grade of the froth in test 9 indicates that the froth and tailing separate more readily at the higher temperatures. Tests 10 to 14 were run with various kinds of oils.

After the froth concentrates had been sampled, they were mixed together, and the two tests shown in Table 38 were run, the flotation cell being used as a "cleaner" unit. The previous tests had been run with the cell as a "rougher" unit, with a high recovery of zinc as the end in view. In cleaning the rougher froth a concentrate of high zinc tenor should be the aim, but these tests were carried to nearly complete removal of the zinc. Hence only a small amount of middling was left for returning to the rougher cell, and the concentrates were perhaps not as high in zinc as could have been obtained by returning a larger quantity of middling.

CONCLUSIONS AS TO FLOTATION OF JOPLIN ORES.

The use of rougher and cleaner units of flotation cells seems essential in flotation of Joplin ores in order to obtain a high recovery and have the concentrates of acceptable grade.

In the flotation experiments made in the Joplin district, the thickness (ratio of solids to water) of the pulp treated ranged from 1 : 3 to 1 : 7. In practice the most favorable ratios for different materials would have to be determined by experiment. Generally a mixture of fine material and slimes requires a denser pulp, whereas for treating a mixture consisting wholly of slimes (finer than 200-mesh) the use of a thinner pulp is desirable. Thickening of the slimes as produced in the mills of this region would be necessary. This could be accomplished by the use of Dorr thickeners or some other device that would give the flotation machines a uniform feed of constant density.

The addition of acid may not be absolutely essential for all the ores of this district, although the local tests showed that a small quantity of acid is desirable, especially in cleaning the rougher concentrates. It was also found that when acid was not used sodium carbonate was beneficial in assisting the selective action of the oil or oil mixture, thus raising the grade of concentrate produced.

In conclusion, the problems to be solved in the application of flotation to the ores of this region may be summarized as follows: (1) The quantity of material available for treatment by flotation without further crushing; (2) the possibility of regrinding certain products from the various stages in concentration in the present milling practice; (3) the oils, oil mixtures, or frothing agents, best suited to the material to be treated; (4) the effect of the acid waters present in some of the mines of the district.

CHARACTER OF CONCENTRATES PRODUCED.

Zinc concentrates in the Joplin district assay 50 to 63 per cent metallic zinc, whereas the lead concentrates contain, on the average, 75 to 80 per cent lead. The average assay for the zinc concentrates for the district is about 57 to 58 per cent zinc, although the zinc concentrates produced from sheet-ground ore and from some of the new mines in Oklahoma and Kansas often run 2 to 3 per cent higher. The content of impurities varies. Some concentrates may contain 0.1 to 5 per cent iron and 0.1 to 2 per cent lead and some lots contain small proportions of lime. To show more fully the composition of the zinc concentrates the following analyses by W. G. Waring, of Webb City, Mo., are given in Tables 39 to 43:

TABLE 39.—*Results of analysis of composite sample of 3,800 shipment lots of Joplin zinc-blende concentrates sold in 1904.*

	Per cent.
Zinc	58.260
Cadmium	.304
Lead	.70
Iron	2.23
Copper	.049
Sulphur	30.42
Manganese	.01
Silica	3.95
Barium sulphate	.82
Calcium carbonate	1.88
Magnesium carbonate	.85
	99.773

TABLE 40.—*Analyses of products showing degree of separation attained in jigging zinc blende from calamine in mixed ore from the same mine. Wentworth district.*

Constituent.	Zinc blende.		Calamine.	
	Sample 1.	Sample 2.	Sample 1.	Sample 2.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Total zinc (Zn)	56.4	58.4	57.3	51.80
Zinc as ZnS	39.8	40.1	18.2	8.65
Zinc as $ZnSiO_3$	16.6	18.3	33.1	43.15
Fe	2.2	3.4	1.5	1.3
CaO	0.45	0.15	0.42	0.40
MgO	Trace.	Trace.		
Pb	0.34	0.52		
Insoluble	11.67			
S				

TABLE 41.—*Analysis of product from mill treating calamine ore. Princeton district.*

	Per cent.
Zinc (Zn)	40.0
Iron (Fe)	0.8
Lime (CaO)	4.05
Magnesia (MgO)	1.44
Lead (Pb)	
Insoluble	
Sulphur (S)	1.43

TABLE 42.—*Analyses of lead concentrates from mill treating lead and zinc ore, Granby district.*

		Sample 1.	Sample 2.
Pb (total)	per cent.	74.3	80.0
Zn	do.	5.2	5.16
Fe	do.	4.2	3.9
$PbCO_3$	do.	....	6.5
$PbSO_4$	do.	....	......
Insoluble	do.	1.3	......

A tabular summary showing the highest, lowest, and average assays of 426 shipments of lead and zinc concentrate from 34 chief producing mines in the Joplin district (including 7 Miami, Okla., mines) during January, February, and March, 1916, is presented in Table 43.

TABLE 43.—*Data on 426 shipments of lead and zinc concentrates.*

Mine No.	Zinc-blende concentrates (first-grade jig concentrates).										Lead (galena) concentrates.						Table concentrates (mixed ore and tailing ore).			
	High.			Low.			Average.				Lead content of jig concentrates.			Lead content of table concentrates.			Average content of—			
	Zn.	Fe.	Pb.	Zn.	Fe.	Pb.	Zn.	Fe.	Pb.	CaO.	High.	Low.	Average.	High.	Low.	Average.	Zn.	Fe.	Pb.	CaO.
1	63.4	2.4	0.4	62.8	1.8	1.50	63.09	2.04	0.25								53.40	5.00	0.97	
2	63.4	2.3	.85	62.0	1.7	1.20	62.53	1.96	.47		79.2	75.7	78.18				58.37	3.48	1.20	
3	61.1	1.9		60.2	1.3		60.65	1.55									56.43	2.51	1.30	
4	60.8	4.1	1.0	56.0	1.9	.50	57.78	3.29	.76		81.8	80.3	81.00				57.04	2.45	2.50	
5	59.5	3.1	.45	58.6	2.4	.18	59.03	2.84	.31								51.64	6.63	.85	
6	63.0	1.3	1.1	62.2	.9	.40	62.62	1.05	.80		77.2	75.3	76.67			72.50	54.70	2.72	1.45	
7	63.4	2.1	.68	62.2	1.1	.21	62.81	1.39	.45							69.20	57.70	2.21	.61	
8	63.5	2.4	.15	61.3	1.7	.02	62.46	2.02	.12								52.95	6.25	1.12	
9	61.6	1.7	.12	60.6	.7	.04	61.16	1.28	.06	1.55							53.50	2.40	.04	
10	55.0	5.4		52.6	3.5		53.90	4.30	.84								48.62	5.70	1.51	0.55
11	56.8	3.4		51.2	2.1		53.78	2.70	2.86	.47			80.85			70.90	49.30	3.40	3.83	
12	63.0	1.1	.1	62.2	.8	.50	62.72	.90	.08								51.80	3.60	1.00	
13	57.9	3.9	.3	56.0	2.9	1.40	57.14	3.29	1.97				81.80							
14	61.6	2.05	4.5	61.5	2.1	.32	61.52	2.00	.25											
15	60.5	3.9	1.9	58.0	1.9	.30	58.96	2.72	.87											
16	58.9	2.2	2.0	56.9	1.4	1.00	57.68	1.56	1.41				82.80							
17	63.3	2.7	3.0	60.8	1.4	.10	62.02	2.06	.16											
18	56.0	2.9	4.0	52.8	2.5	2.65	54.57	2.69	3.34							54.4				
19	62.2	1.6	.4	60.8	1.3	.10	61.82	1.48	.16											
20	61.3	3.3	.5	55.9	1.2	.15	59.05	1.97	.22											
21	56.0	3.8		54.0	2.9		54.93	3.31	1.67											
22	57.9	3.9		56.0	2.9		57.14	3.29	1.97		81.9	81.0	81.53							
23	54.9	7.8	.23	51.2	2.7	.03	53.41	5.97	.12	1.17										
24	60.3	2.6	.29	58.8	.7	.03	59.63	1.80	.17	1.64			74.35			58.1				
25	55.7	2.4		53.0	1.6		54.32	1.94	1.15	.15			80.30							
26	60.5	2.9		57.8	2.1		59.85	2.46												
27	47.6	13.8	1.0	36.7	7.2	.36	42.10	10.75	.62	.32			80.62			53.70				
28	57.7	2.3	1.0	55.8	2.1	.50	56.31	2.16	.65				78.00							
29	59.4	5.2	1.4	52.7	1.5	.80	56.98	2.80	1.12				73.70							
30	59.7	3.1		59.0	2.5		59.25	2.75	.37											
31	a 51.7			48.1			50.08													
32	a 43.1			32.0			38.05													
33							38.50													
34							42.80													

a Calamine.

HANDLING OF CONCENTRATES AND POSSIBLE CAUSES OF DEPRECIATION IN WEIGHT DURING SHIPMENT.

The finished products from the mills are stored in bins just outside of the mill. After the concentrate has been sold it is shoveled into wagons and hauled to the shipping point, the hauling and freight charges being paid by the smelter or buyer.

It would be practically impossible to determine definitely the loss in weight of lead and zinc concentrates from the time they leave the mine or mill until they reach their destination, because of the many factors that may cause such loss.

The chief cause of this depreciation in weight is loss of moisture. The moisture loss, however, depends on the kind of ore from which the concentrates were derived, whether for zinc concentrates, from silicate, carbonate, or sulphide ore; whether the concentrates are coarse or fine; the moisture content at the start; the distance and time in transit; the weather conditions; and the season of the year in which the shipment is made. In winter the concentrates hold moisture longer than in summer. Shrinkage in weight through loss of moisture is generally more marked in fine than in coarse concentrates, partly because of the usually higher moisture content before shipment.

From a number of cars of medium coarse concentrates of zinc sulphide shipped during the months of December and January from a certain mine to the smelter a distance of about 440 miles, the depreciation in weight by evaporation of moisture in transportation was found to be 1.4 per cent. This difference no doubt would have been greater if the concentrates had been shipped during the summer. Moreover, the loss in moisture depends on the time the concentrate is in transit from the mill to the smelter and the weather conditions en route. The following averages of moisture content of zinc concentrates, figured from 10 samples for each month, during the fiscal year 1914 to 1915 as shipped to the smelter, may be of interest:

Average moisture content of zinc concentrates for 12 months.

	Moisture content, per cent.
1914.	
July	3.07
August	3.39
September	3.16
October	3.56
November	3.52
December	3.53
1915.	
January	3.91
February	4.22
March	3.30
April	3.69
May	3.93
June	3.47

NOTE.—A great deal of rain fell during the months of May and June, 1915, in contrast to the fairly dry month of June, 1914, when the average moisture content was 2.91.

Other causes of loss of weight in the shipment of concentrates from the mine or mill to the smelter are in hauling the material from the mill to the shipping cars, in loading and unloading the cars, and during rail transportation. Some ore buyers have stated that the depreciation in weight during transportation from causes other than the loss of moisture may amount to 0.5 per cent or less, whereas others claim that the loss is slight and that the net weights of the concentrates at the mine or mill and at the smelter usually check in the long run.

Also differences in weights have been found to be due to the use of inaccurate scales either at the mine or mill or at the smelter. To avoid difficulties from lack of agreement in the weight as determined from the scales at the mine or smelter and as determined by the railroads, the cars should always be weighed before loading, as the tare of a car will change in time through wear or through repairs, etc.

The preceding statements show how difficult it would be to obtain exact data regarding the depreciation in weight of concentrates during transit. The only method of procedure to follow in obtaining such data would be to take the matter up directly with the different smelters and from their figures calculate the differences in net weight of mineral content over a period of two or three years; and then to distribute this loss, as nearly as possible, to the various causes along the entire route of transportation.

Depreciation or change in assay value during transportation owing to chemical reaction of the ore or to any foreign material being mixed with the ore, is believed to be slight, if any.

DISPOSAL OF TAILINGS.

Usually throughout the region the tailings from the mills are piled near the mills by means of elevators and launders. As a pile of tailings increases in height the tailings from the launder are directed into the boot of another elevator and raised again as shown in Plate XV, *A*. Other means have been tried, such as belt conveyors and trams, but elevators seem to find favor among most of the operators.

Except for a few of the older and richer piles, the tailings are rarely retreated. This is especially true of tailings from the sheet-ground mines where the lead and zinc content of the ore is relatively low. Tailing piles of any considerable size are, as a rule, purchased by the railroads for use as road ballast. The tailings are usually loaded by means of steam shovels, as shown in Plate XVI, or the tailings from the mill may be permitted to flow directly through launders into railroad cars.

A. METHOD OF DISPOSING OF TAILINGS.

B. CONVEYING TAILINGS INTO MILL WITH BUCKET ELEVATOR FOR RE-TREATMENT.

LOADING TAILINGS FOR RAILROAD BALLAST.

RETREATMENT OF TAILINGS.

In the retreatment of tailings, certain conditions are essential to insure a good recovery of the mineral content. A few tailing piles have yielded good profits, but many piles have been retreated without sufficient testing previous to retreatment to determine the character of the material and the mineral recovery possible. Screen analyses of representative samples and assays of the screen products are necessary in determining the value of a tailing pile and the kind of equipment necessary for obtaining the best results. Screen analyses and assays of samples taken from many different tailing piles throughout the district have shown that the recoverable values were in the finer size material, whereas others, where the ore treated was chatty, showed the values in the coarser sizes. Screen analyses of tailings for retreatment will show whether the tailings need fine grinding or whether only a system of screening will be necessary.

Chemical analyses of various screen products of tailings taken from the interior of tailing piles have shown the presence of zinc in the form of sulphate. This is especially true where the mine waters have been acid, containing free sulphuric acid, and the tailing piles have been standing for several years. Although not enough tests were made to determine the proportion of the total zinc in the tailings as sulphate or as oxide, it would be well to determine by thorough sampling and assaying the form in which the zinc is present, especially for tailings from mines where the mine water has been acid and for tailing piles which show marked oxidation on the surface. Zinc sulphate is not only soluble in water but its specific gravity is less than that of flint. Some of the zinc may have been leached out of the tailing pile. Such preliminary precautions might save an operator the investment of a tailing mill.

Plates XV, *B*, and XVII show two methods of bringing the tailings into the mill for retreatment. The method shown in Plate XVII is the one most commonly used.

Screen analyses and assays of screen products of samples taken from fairly rich and relatively old tailing piles are shown in Tables 44 to 46 following. The tailing piles represented in Tables 44 and 45 were being retreated, but not more than 1.5 tons of concentrate, on the average, per 100 tons of tailings treated was being produced in either mill, which shows that poor zinc recoveries were being made. Finer grinding should have been done and the mills should have been properly equipped for treating the fine material. Table 46 represents a pile that would not pay to retreat under present conditions.

TABLE 44.—*Screen analyses of representative samples taken from tailing pile in West Joplin district.*

A. SAMPLES TAKEN FROM TOP OF PILE.

Size of mesh.	Size of screen openings.		Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc.
	Mm.	Inch.			
3	6.680	0.263	2.85	2.4	1.9
10	1.651	.065	64.9	3.4	62.1
35	.833	.0328	12.65	2.9	10.3
200	.074	.0029	13.95	4.6	18.2
<200	.074	.0029	5.65	4.7	7.5
Total or average			100.00	3.55	100.0

B. SAMPLES TAKEN FROM MIDDLE OF PILE.

3	6.680	0.263	6.0	1.7	2.5
10	1.651	.065	70.4	4.5	78.25
35	.833	.0328	10.55	2.7	7.05
200	.074	.0029	9.15	3.0	6.75
<200	.074	.0029	3.9	5.7	5.45
Total or average			100.0	4.05	100.00

C. SAMPLES TAKEN FROM BOTTOM OF PILE.

3	6.680	0.263	12.9	2.2	13.1
10	1.651	.065	56.25	2.3	59.4
35	.833	.0328	12.1	1.7	9.45
200	.074	.0029	14.45	1.8	11.95
<200	.074	.0029	4.3	3.1	6.1
Total or average			100.00	2.17	100.00

TABLE 45.—*Screen analysis of representative sample taken from tailing pile in Galena district.*

Size of mesh.	Size of screen openings.		Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc.
	Mm.	Inch.			
3	6.680	0.263	22.95	2.0	18.65
10	1.651	.065	48.7	2.2	43.6
35	.833	.0328	10.5	2.5	10.7
200	.074	.0029	14.65	3.6	21.45
<200	.074	.0029	3.2	4.3	5.6
Total or average			100.00	2.45	100.00

TABLE 46.—*Screen analysis of representative sample taken from tailing pile in Prosperity district.*

Size of mesh.	Size of screen openings.		Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc.
	Mm.	Inch.			
3	6.680	0.263	24.7	0.9	24.9
10	1.651	.065	45.2	.9	45.5
35	.833	.0328	12.15	.4	5.45
200	.074	.0029	14.9	1.0	16.7
<200	.074	.0029	3.05	2.2	7.45
Total or average			100.00	0.89	100.00

DRAG-LINE METHOD OF BRINGING TAILINGS TO MILL FOR RE-TREATMENT.

LABOR CONDITIONS.

Nearly all the miners and surface men in the Joplin district are American born. As a rule they are efficient and intelligent, and readily pick up new ideas whenever they are given the opportunity.

The shift for underground work in the mines is 8 hours, whereas the mill shifts are 10 hours each. When a mine has been well developed a single underground shift can often break enough ore to keep the mill running during two 10-hour shifts. A few of the larger companies are now running their mills three 8-hour shifts rather than two 10-hour shifts. It is believed that this plan, if followed, will in many cases make for greater efficiency in both mining and milling.

In 1916 the average scale of wages paid to the miners in this district was based on a sliding-scale basis of a 25-cent increase or decrease for every $10 increase or decrease in the average base price for 60 per cent zinc concentrates covering a period of one month. The following table shows the wages paid to the men, according to their occupation, based on the average price for 60 per cent zinc concentrates between $80 and $90 per ton.

TABLE 47.—*Sliding scale of wages based on an $80 to $90 base price for 60 per cent zinc concentrates.*[a]

Underground men:	
Powder man	$4.00
Powder man helper	3.50
Tub hooker	4.00
Drill man	3.75
Drill man helper	3.25
Drill man, tripod	4.00
Drill man helper, tripod	3.50
Trackman	3.75
Trackman helper	3.25
Tub runner	2.75
Bruno man[b]	2.75
Roof man	3.75
Shovelers, 9 to 10 cents per tub (1,000 pounds), average wage per day	4.00–4.50
Pump man	3.25
Surface men:	
Hoist man, shaft	4.00
Hoist man, tram	2.75
Cull hand	2.80
Crusher feeder	3.20
Jig-man helper	3.75
Sludge man (running tables):	
10-hour shift	3.20
8-hour shift	2.95
Engineer	3.75
Roustabout	2.80

[a] Based on figures given in the Engineering and Mining Journal, vol. 101, 1916, p. 751.
[b] Term for miner who pulls down fine ore piled in a stope.

COST DATA.

To obtain average detail costs of mining and milling operations in this district would be difficult, as each company has its own system of keeping costs, and the mining conditions differ somewhat. Most of the larger companies keep detailed cost accounts, whereas others, usually at the smaller mines, keep only general costs. Three examples of itemized cost data are given in Tables 48 to 50. These cost data are for relatively large mines that have been operating for some time and can, therefore, be considered somewhat lower than the average throughout the district. In the new zinc fields that have recently been opened up near Tar River and Picher, Okla., the mining and milling costs are considerably higher than those in the tables, ranging from $1.50 to $2.50 per ton; but due to the higher lead and zinc content the costs per ton of concentrates produced are much lower, at present averaging $20 to $30 per ton.

TABLE 48.—*Mining and milling costs of a representative sheet-ground mine in the Webb City-Carterville district.*

Item.	1914	1915	1916
Mining costs [costs per ton of ore mined]:			
Ground boss	$0.0102	$0.0117	$0.0119
Drilling	.1927	.2608	.2735
Blasting	.1721	.1857	.1792
Roof trimming	.0111	.0160	.0182
Loading	.1509	.2125	.2317
Conveying to shaft	.0705	.1106	.1427
Hoisting	.0367	.0405	.0487
Lighting	.0070	.0062	.0052
Miscellaneous (superintendence, insurance, etc.)	.0209	.0403	.0711
Total mining cost	.6721	.8843	.9822
Milling costs [costs per ton of ore milled]:			
Culling	.0195	.0120	.0107
Crushing	.0448	.0322	.0376
Elevating	.0402	.0430	.0392
Jigging	.0318	.0358	.0387
Tabling	.0364	.0420	.0372
Water	.0070	.0059	.0063
Power	.0279	.0291	.0306
Miscellaneous (superintendence, insurance, etc.)	.0202	.0226	.0257
Total milling cost	.2278	.2226	.2260
Total mining and milling costs [costs per ton of ore mined]:			
Mining	.6721	.8843	.9822
Milling	.2249	.2214	.2255
Pumping	.0587	.0507	.0443
Miscellaneous	.0523	.0307	.0240
General administrative	.0398	.0252	.0435
Total mining and milling	1.0478	1.2123	1.3195
Total cost per ton of concentrates produced	33.3750	44.9813	51.9855
Quantity of concentrates produced, per 100 tons of ore mined and milled:	*Tons.*	*Tons.*	*Tons.*
Lead concentrates	1.0109	0.9423	0.9686
Zinc concentrates	2.1285	1.7528	1.5886
Total concentrates	3.1394	2.6951	2.5572

TABLE 49.—*Mining and milling costs of a sheet-ground mine in which mining consists principally in "taking up stope."*

[Cost per ton of ore mined.]

Item.	1914	1915	1916
Mining costs:			
Labor, without shoveling	$0.2621	$0.1657	$0.2558
Labor, shoveling	.1362	.1713	.1802
Salaries	.0215	.0117	.0132
Explosives	.2036	.0920	.1464
Fuel	.0537	.0348	.0471
Liability insurance	.0159	.0146	.0196
Compensation for injuries to employees	.0028	.0111	.0020
Water	.0077	.0043	.0047
Lubricating oil	.0073	.0041	.0046
Repairs, supplies, and sundries	.0410	.0315	.0468
Total mining cost	a.7518	.5411	.7204
Milling costs:			
Labor	.0825	.1062	.1298
Salaries	.0215	.0117	.0132
Hard iron	.0327	.0266	.0297
Belting	.0228	.0083	.0112
Fuel	.0258	.0190	.0206
Liability insurance	.0033	.0046	.0058
Compensation for injuries to employees	.0001	.0008	.0005
Water	.0077	.0044	.0047
Lubricating oil	.0028	.0022	.0017
Repairs, supplies, and sundries	.0309	.0295	.0436
Total milling costs	.2301	.2133	.2608
Total mining and milling costs	a.9819	.7544	.9812
Total cost per ton of concentrates produced	26.360	50.1940	55.3083
Quantity of concentrates produced per 100 tons of ore mined and milled:	*Tons.*	*Tons.*	*Tons.*
Lead concentrates	0.5922	0.0785	0.1014
Zinc concentrates	3.1326	1.4244	1.6726
Total concentrates	3.7248	1.5029	1.7740

a In 1914 both heading and stope were being mined with a much smaller tonnage, hence costs were relatively higher than in 1915.

TABLE 50.—*Mining and milling costs of a representative mine in the Commerce district, Okla.*

[Costs per ton of ore mined.]

Item.	1914	1915	1916
Ground labor	$0.2885	$0.4001	$0.4667
Power	.0725	.0929	.1368
Timber	.0131	.0028	....
Powder	.1267	.1591	.1513
Mill labor	.1328	.1896	.2451
Mill fuel	.0218	.0222	.0362
Mill supplies	.1117	.1078	.1178
Shop labor	.0427	.0478	.0568
Shop supplies	.0594	.0633	.0645
Top labor	.0758	.0841	.1161
Pumping	.0140	.0202	.0291
General expenses (including overhead)	.0475	.1265	.1599
Total mining and milling cost	1.0065	1.3164	1.5803
Total cost per ton of concentrates produced	18.8052	30.4910	39.6203
Quantity of concentrates produced per 100 tons of ore mined and milled:	*Tons.*	*Tons.*	*Tons.*
Lead concentrates	1.6450	1.6582	1.2078
Zinc concentrates	3.7066	2.6588	2.7757
Total concentrates	5.3516	4.3170	3.9835

MARKETING LEAD AND ZINC ORES.

The schedules for the purchase of lead and zinc ore in the Joplin district and the methods of marketing the ores differed somewhat from the schedules and methods employed in the Western States. The producer is paid by the ton of concentrate, the ore being purchased outright at the mine. The hauling and freight charges are paid by the smelter. The price paid for the various grades of concentrates varies according to a sliding scale based on the price paid for concentrates of a standard grade, which is known as the base price. As the concentrates are sold on the open market, there is more or less competition among the smelter representatives in the district who bid on the various lots offered for sale by the producers.

The schedules or basis on which the different kinds of ore are purchased are (1) 60 per cent metallic zinc for sphalerite or zinc-blende concentrates, (2) 40 per cent metallic zinc for concentrates of the oxidized zinc ores, the carbonate and silicate, and (3) 80 per cent metallic lead for the galena concentrates. The base price depends largely upon the state of the spelter market and is determined during the latter part of each week.

As the different grades of concentrates produced from the many mines throughout the district vary, the prices received per ton of concentrates, dry weight, are determined by the grade of concentrates and the content of impurities. Thus for zinc-blende concentrates, if the assay varies from the basis, $1 is added to or deducted from the base price for each 1 per cent of zinc above or below 60 per cent. Penalties are imposed for certain impurities if present. Usually for each per cent of iron above 1 per cent, $1 is deducted from the base price. Where the percentage of iron is high this penalty is often greatly reduced, if imposed at all. Lead and lime are penalized, usually by deducting a certain amount from the base price per ton of concentrates. This results in two or three different grades of zinc concentrates, each with its base price. For example, a zinc-blende concentrate containing 0.5 per cent or more lead would no longer be classed under the standard grade, but under a grade with a lower base price.

Because of the great demand for the better grades of spelter since the war began, the smelters have been somewhat more exacting as to impurities in zinc concentrates and have lowered the allowable percentages of impurities for the different grades. Other undesirable impurities in zinc concentrates are barite, fluorspar, and bitumen, but these are not frequently found with the ores of the Joplin district. Sorted ore, such as lump galena ore and the oxide ores of zinc, is usually sold on a flat bid, the ore buyer inspecting the ore and making an offer of so much per ton for it.

HEALTH CONDITIONS IN THE MINES.

In a metal mine the conditions affecting the health of the miner depend largely on the material mined, the depth of the mine, the gases present, and the ventilation. In the Joplin district the mines are shallow, injurious gases such as are found in coal mines or deep metal mines are lacking, and the ventilation is generally good.

VENTILATION.

The adequate ventilation is largely a result of thorough inspection by the State mine inspectors, who zealously enforce the law compelling operators to sink shafts, drill air holes, or cut drifts to openings, wherever needed to protect the health of the miners. Fans are used where natural ventilation is insufficient or can not be utilized.

WASH AND CHANGE HOUSES.

In many mines the ground underfoot is wet, so that the men, after the shift, leave the mine workings with clothing more or less wet. Most of the mines have change houses with individual lockers, a place for hanging and drying wet clothes, a large wash sink with running hot and cold water, and a few with shower baths. As many of the men drive or walk to and from the mines the change houses are necessary and are largely used. Although in general these buildings are small and not elaborate, they serve the purpose of allowing the clothes to dry after each shift, and guard the men against unnecessary exposure. At mines where there are no change houses the men are subjected, especially during the winter months, to a sudden change of temperature at the surface, with the consequent risk of severe colds. Although the men become hardened to these conditions, the constant exposure day after day gradually reduces their vitality, and precautions to prevent such conditions are highly desirable.

ABATEMENT OF ROCK DUST.

In the mines, especially the sheet-ground mines, that are practically free from water and dampness, more or less fine dust from the drilling and blasting may be carried in the air. This dust, if inhaled by the miner every day, may in time cause lung diseases. To quote from the preliminary report[a] on this district:

> The extent of harm done by inhaling the dust in metal mines is not definitely known, but it is believed that much of the lung trouble often known as "miners' consumption" has been caused by it. Some of the operators in the district lessen the injurious effect of rock dust by dampening the faces, but this is done at few mines.

[a] Wright, C. A., Mining and treatment of lead and zinc ores in the Joplin district, Missouri, a preliminary report: Tech. Paper 41, Bureau of Mines, 1913, p. 40

This wetting of the face, however, should be done at all mines where rock dust is prevalent, for it would without doubt prevent much sickness and spreading of disease. In this connection it may be stated that the Bureau of Mines, in cooperation with the Bureau of Public Health, is to investigate a number of problems relating to the hygiene of mines and to the mining industry, the investigations being conducted with regard to scientific as well as administrative problems and including a study of the prevalence and prevention of lung diseases among miners.

Following a preliminary report [a] of these investigations, published in 1915, a more detailed study was made of the harmful effect of the finely divided, siliceous rock dust on the miner's lungs when it is suspended in the air in the working drifts, and also with regard to other sanitary conditions at the mines as well as in the homes of the miners. Dr. Lanza, of the Public Health Service, spent several months making physical examinations of the miners and their families to determine the prevalence of pulmonary disease. The complete report of the investigations has been published in Bulletin 132[b] of the Bureau of Mines, to which the reader is referred for further details.

As a result of the investigations and the enforcement of the new mining laws and the careful supervision of the State mine inspectors, there has been a marked gain in abating the rock dust in the mines and in improving sanitary conditions. Also the miners themselves have shown willingness to cooperate with the mining companies.

RESUME.

In view of the fact that the main object of every mining operation is to obtain the greatest profits from the expenditures made, certain conditions must constantly be considered. The operator should not only attempt to obtain the most efficient working conditions with the equipment and labor on hand, but should also investigate the possibilities of improvements that will result in a greater saving and in increased profits, even if they involve changes in the methods of mining and milling at no small cost. Although there has been a marked improvement in the saving of mineral the past few years by improved mill construction and more efficient operation and equipment, at many properties a greater mineral recovery and increased profits could be realized. As the conditions at any two mines are not exactly alike, each mine or mill must be considered by itself. Likewise any improvements that could be made for increasing the working efficiency in labor, time, and production, must be determined under the existing conditions at each mine separately. Hence a few of the main points that might prove worthy of more thorough

[a] Lanza, A. J., and Higgins, Edwin, Pulmonary disease among miners in the Joplin district, Missouri, and its relation to rock dust in the mines: Tech. Paper 105, Bureau of Mines, 1915, 48 pp.

[b] Higgins, Edwin, Lanza, A. J., Laney, F. B., and Rice, G. S., Siliceous dust in relation to pulmonary disease among miners in the Joplin district, Missouri: Bull. 132, Bureau of Mines, 1917, 112 pp.

investigation by the operators at many of the mines and mills, as brought out more or less in detail in the preceding pages of this report, are given here.

The proper grade and quantity of explosive, and also the proper strength of cap, to be used in breaking ground, should be determined according to the character of the ore-bearing ground and the product to be obtained. The increased effectiveness of powder by the use of stemming should be determined as to the quantity of ore broken. A saving in the use of powder would be an important item in reducing the mining costs.

To follow a system of mining that leaves the working face more accessible and in the best condition for subsequent drilling and blasting is important. The drill man or mine foreman should consider and try to foresee as far as practicable the form in which the face will be left after each round of shots.

The best method of hauling the ore from the working face to the shaft for saving time, labor, and costs should be ascertained.

The only sorting in the mines of this district worthy of mention is the culling of large bowlders of waste rock. Because of the large quantities of waste mined with the ore, especially in the sheet-ground mines, where in many places the ore is in "runs" with a comparatively thick layer of barren rock between, it is reasonable to assume that some system of hand sorting, underground or in the mill, could be used successfully. The discarding of a larger proportion of waste rock would result in a relatively higher metal recovery in the mills, save power, reduce wear and tear on crushing machinery, elevator belts, and other equipment, and increase the relative capacity of the mill.

At the mills sampling and mill testing should be done to find out what results are being obtained. To be of value, however, the sampling should be careful and systematic. The sampling of tailings without including the overflow water from settling tanks in determining the total metal loss of a mill, an oversight frequently observed, is of little value. Screen analyses of the samples to determine what sizes contain the valuable minerals are important, as a more definite idea can be obtained as to how and where further savings may be effected.

In jigging, it would seem advisable to deslime all material before it passes to the rougher jigs. Where the ore is chatty and a large quantity of middlings is produced, the middlings should be reground and treated separately over "chat" jigs rather than crushed and returned to the first rougher jigs, as practiced in some mills.

Desliming of the feed and separate treatment of the middlings can be applied to the treatment of sands on tables equally as well as in jigging. Roughing and cleaning tables could also be used to advantage in some of the larger mills of the district.

The possibilities of flotation are well worth investigation by every operator, as a small flotation unit might increase the mineral recovery 10 to 20 per cent.

In addition to efficiency in the mining and milling of the ores, health and safety conditions should be given more attention. Each operating company should see that in its mine or mill due effort is made to effect any improvement that will better the health and safety conditions. Such subjects as the safe use of explosives, mine timbering, fire protection, safety appliances, and sanitation in and about the mines, require careful study and should be just as vital to the mine operator as the number of tons of ore mined and treated each day.

At many of the mines and mills excellent results are being obtained. It is hoped, however, that every operator will endeavor to effect greater saving, so far as is profitable, and take advantage of any experiments made by others that would improve his own practice if applied.

MISCELLANEOUS DATA.

The number of zinc-smelting retorts in the United States at the end of 1916, with comparative figures for 1915, are given in Table 49. Monthly prices for lead and spelter for these years are shown in Tables 50 and 51.

TABLE 51.—*Zinc-smelting capacity of plants in the United States.*[a]

[Number of retorts at end of 1915 and 1916.]

Name.	Situation.	1915.	1916.
American Spelter Co	Pittsburg, Kans	896	[b] 896
American Steel & Wire Co	Donora, Pa	3,648	9,120
American Zinc and Chem. Co	Langeloth, Pa	3,648	7,296
American Zinc Co. of Illinois	Hillsboro, Ill	4,000	4,864
American Zinc, Lead and Smg. Co	Dearing, Kans	4,480	4,480
American Zinc, Lead and Smg. Co	Caney, Kans	6,080	6,080
Arkansas Zinc and Smelting Corpn	Van Buren, Ark		2,400
Athletic Min. and Smelting Co	Fort Smith, Ark		(c)
Bartlesville Zinc Co	Bartlesville, Okla	5,184	7,488
Bartlesville Zinc Co	Blackwell, Okla		8,800
Bartlesville Zinc Co	Collinsville, Okla	10,080	13,440
Chanute Spelter Co.[d]	Chanute, Kans	1,280	1,280
Cherokee Smelting Co	Cherokee, Kans	896	[e] 896
Clarksburg Zinc Co	Clarksburg, W. Va	3,648	3,648
Collinsville Zinc Co	Collinsville, Ill	[b] 1,536	1,984

[a] From Engineering and Mining Journal, vol. 103, Jan. 6, 1917, p. 25.
[b] No report received; entered the same as previous year.
[c] Under construction.
[d] Operated by American Metal Co. in 1915.
[e] Idle.

TABLE 51.—*Zinc-smelting capacity of plants in the United States*—Continued.

Name.	Situation.	1915.	1916.
Eagle-Picher Lead Co.	Henryetta, Okla.		3,000
Edgar Zinc Co.	Carondelet, Mo.	2,000	2,000
Edgar Zinc Co.	Cherryvale, Kans.	4,800	4,800
Fort Smith Spelter Co.	Fort Smith, Ark.		2,560
Granby Mining and Smltg. Co.[a]	Neodesha, Kans.	3,760	3,760
Granby Mining and Smltg. Co.[a]	East St. Louis, Ill.	3,220	4,864
Grasselli Chemical Co.	Clarksburg, W. Va.	5,760	5,760
Grasselli Chemical Co.	Meadowbrook, W. Va.	8,592	8,544
Grasselli Chemical Co.	Terre Haute, Ind.		(b)
Hegeler Zinc Co.	Danville, Ill.	3,600	5,400
Henryetta Spelter Co.	Henryetta, Okla.	600	3,000
Illinois Zinc Co.	Peru, Ill.	4,640	4,640
Iola Zinc Co.	Concreto, Kans.	660	[c] 660
Joplin Ore and Spelter Co.	Pittsburg, Kans.	1,440	[d] 1,792
J. B. Kirk Gas and Acid Co. [e]	Iola, Kans.	3,440	3,440
Kusa Spelter Co.	Kusa, Okla.	3,710	3,720
La Harpe Spelter Co.	Kusa, Okla.	(f)	4,000
Lanyon Smelting Co.	Pittsburg, Kans.	448	448
Robert Lanyon Zinc and Acid Co.	Hillsboro, Ill.	1,840	3,200
Lanyon-Starr Smelting Co.	Bartlesville, Okla.	3,456	3,456
Matthiessen and Hegeler Zinc Co.	La Salle, Ill.	6,168	6,168
Mineral Point Zinc Co.	Depue, Ill.	9,068	9,068
Missouri Zinc Smelting Co.	Rich Hill, Mo.		[d] 448
National Zinc Co.	Bartlesville, Okla.	4,260	4,970
National Zinc Co.	Springfield, Ill.	3,200	3,800
Nevada Smelting Co.	Nevada, Mo.	672	672
New Jersey Zinc Co. of Penn.	Palmerton, Pa.	6,720	7,200
Oklahoma Spelter Co.[d]	Kusa, Okla.	(f)	1,600
Owen Spelter Co.[g]	Caney, Kans.	1,280	1,920
Pittsburg Zinc Co.[h]	Pittsburg, Kans.	910	910
Prime Western Spelter Co.	Gas City, Kans.	4,868	4,866
Quinton Spelter Co.	Quinton, Okla.		1,340
Sandoval Zinc Co.	Sandoval, Ill.	896	672
Tulsa Fuel and Manufacturing Co.	Collinsville, Okla.	6,232	6,232
United States Smelting Co.	Altoona, Kans.	3,960	4,600
United States Smelting Co.	Checotah, Okla.		4,480
United States Smelting Co.	La Harpe, Kans.	1,924	1,926
United States Zinc Co. [i]	Sand Springs, Okla.	5,680	8,000
United States Zinc Co.	Pueblo, Colo.	2,208	1,984
United Zinc Smelting Corp.	Moundsville, W. Va.		(b)
Weir Smelting Co.	Weir, Kans.	(f)	448
Total		155,388	211,12

a Subsidiary of American Zinc, Lead & Smelting Co., 1916.
b Under construction.
c No report received; entered the same as previous year.
d Idle latter part of 1916.
e Taken over by United States Smelting Co. in July, 1915.
f Was under construction in 1915.
g Operated by American Zinc, Lead & Smelting Co.
h Operated by American Metal Co.. in 1915.
i Formerly Tulsa Spelter Co.

TABLE 52.—*Monthly prices of lead at New York City, N. Y., for the years 1905–1916.*[a]

[In cents per pound.]

	1905	1906	1907	1908	1909	1910	1911	1912	1913	1914	1915	1916
January	4.552	5.600	6.000	3.691	4.175	4.700	4.483	4.435	4.321	4.111	3.729	5.921
February	4.450	5.464	6.000	3.725	4.018	4.613	4.440	4.026	4.325	4.048	3.827	6.246
March	4.470	5.350	6.000	3.838	3.986	4.459	4.394	4.073	4.327	3.970	4.053	7.036
April	4.500	5.404	6.000	3.993	4.168	4.376	4.412	4.200	4.381	3.810	4.221	7.630
May	4.500	5.685	6.000	4.253	4.287	4.315	4.373	4.194	4.342	3.900	4.274	7.463
June	4.500	5.750	5.760	4.466	4.350	4.343	4.435	4.392	4.325	3.900	5.932	6.936
July	4.524	5.750	5.288	4.447	4.321	4.404	4.499	4.720	4.353	3.891	5.659	6.352
August	4.665	5.750	5.250	4.580	4.363	4.400	4.500	4.569	4.624	3.875	4.656	6.244
September	4.850	5.750	4.813	4.515	4.342	4.400	4.485	5.048	4.698	3.828	4.610	6.810
October	4.850	5.750	4.750	4.351	4.341	4.400	4.265	5.071	4.402	3.528	4.600	7.000
November	5.200	5.750	4.376	4.330	4.370	4.442	4.298	4.615	4.293	3.683	5.155	7.042
December	5.422	5.900	3.658	4.213	4.560	4.500	4.450	4.303	4.047	3.800	5.355	7.513
Average 5or year	4.707	5.347	5.325	4.200	4.273	4.446	4.420	4.471	4.370	3.862	4.628	6.858

[a] From Engineering and Mining Journal, vol. 83, Jan. 5, 1907, p. 10; vol. 85, Jan. 4, 1908, p. 11; vol. 89, Jan. 8, 1910, p. 67; vol. 93, Jan. 6, 1912, p. 15; vol. 99, Jan. 9, 1915, p. 58; vol. 103, Jan. 6, 1917, p. 16.

TABLE 53.—*Monthly prices of spelter at St. Louis, Mo., for the years 1905–1916.*[a]

[In cents per pound.]

	1905	1906	1907	1908	1909	1910	1911	1912	1913	1914	1915	1916
January	6.032	6.337	6.582	4.363	4.991	5.951	5.302	6.292	6.854	5.112	6.211	16.745
February	5.989	5.924	6.664	4.638	4.739	5.419	5.368	6.349	6.089	5.228	8.255	18.260
March	5.917	6.056	6.687	4.527	4.607	5.487	5.413	6.476	5.926	5.100	8.366	16.676
April	5.667	5.931	6.535	4.495	4.815	5.289	5.249	6.483	5.491	4.963	9.837	16.525
May	5.284	5.846	6.291	4.458	4.974	5.041	5.198	6.529	5.256	4.924	14.610	14.106
June	5.040	5.948	6.269	4.393	5.252	4.978	5.370	6.727	4.974	4.850	21.038	11.582
July	5.247	5.856	5.922	4.338	5.252	5.002	5.545	6.966	5.128	4.770	18.856	8.755
August	5.556	5.878	5.551	4.556	5.579	5.129	5.803	6.878	5.508	5.418	12.611	8.560
September	5.737	6.056	5.086	4.619	5.646	5.364	5.719	7.313	5.444	5.230	13.270	8.820
October	5.934	6.070	5.280	4.651	6.043	5.478	5.951	7.276	5.188	4.750	12.596	9.659
November	5.984	6.225	4.775	4.909	6.231	5.826	6.223	7.221	5.083	4.962	15.792	11.422
December	6.374	6.443	4.104	4.987	6.099	5.474	6.151	7.081	5.004	5.430	15.221	10.495
Average for year	5.730	6.048	5.812	4.578	5.352	5.370	5.608	6.799	5.504	5.061	13.054	12.634

[a] From Engineering and Mining Journal, vol. 83, Jan. 5, 1907, p. 13; vol. 85, Jan. 4, 1908, p. 15; vol. 89, Jan. 8, 1910, p. 71; vol. 93, Jan. 6, 1912, p. 22; vol 99, Jan. 9, 1915, p. 64; vol. 103, Jan. 6, 1917, p. 24.

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