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THE METALLURGY OF QUICKSILVER

BY

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THE METALLURGY OF QUICKSILVER.

By L. H. Duschak and C. N. Schuette.

INTRODUCTION.

In the years 1850 to 1923, the United States produced 2,426,000 flasks[1] (73,600 metric tons) of quicksilver worth $120,500,000. California yielded 2,195,000 flasks of this total; the remainder came from Texas, Oregon, Nevada, and Arizona. Most of this metal has been extracted from low-grade ores, those containing less than 0.5 per cent mercury or 10 pounds per ton. The finished product or " virgin metal " is made at the mine and shipped in flasks.

Quicksilver is unique in being the only metal that is liquid at ordinary temperature. Because of this and other physical and chemical properties, it is perhaps more indispensable to industry than any other metal. On the other hand, the quicksilver industry of the world is of vanishing significance[2] when compared to the major mineral industries with respect to quantity and value of product, capital invested, or the number of men employed. The peculiar value of quicksilver is due to the fact that in some of its applications no substitute is available and in others the substitutes would be unsatisfactory or extremely expensive. Scarcely a branch of science or industry fails to make some use of mercury or its compounds. As a detonator for explosives, mercury fulminate holds first place and in safety and reliability could be replaced only by the highly expensive silver fulminate.[3] Through its use in detonators and in the metallurgy of the precious metals quicksilver is of special importance to mining. In medicine, in the manufacture of electrical apparatus, the production of pigments and antifouling paints, and the general field of experimental science quicksilver is equally indispensable.

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[1] Statistics from U. S. Geological Survey. One flask now contains 75 lbs.; formerly, up to 1904, 76½ lbs.

[2] If the new mercury boiler should prove as successful in practice as its sponsors believe, it will cause an increased demand for quicksilver, but this demand is not likely to be great in the immediate future: Quicksilver production for 1923, U. S. Geol. Survey, June 23, 1924.

[3] According to S. P. Howell and J. E. Crawshaw, explosives engineers of the Bureau of Mines.

Because of its small commercial importance and the lack of a stable market and price for the metal, the quicksilver industry, as a whole, has not had the benefit of the same metallurgical and business direction that has been given to the winning of the major metals. The unique relation of mercury to national health (certain drugs) and security (fulminate for defensive purposes), and the indispensability of the metal and its compounds in science and industry seem to justify investigation of the quicksilver industry by the Bureau of Mines. In Europe, government interest has been direct. The rich deposits at Almaden, Spain, are owned and the product is marketed by the Spanish Government; the mines at Idria, formerly belonging to the Austro-Hungarian Government, have now passed under Italian control; and the most productive mines of the Monte Amiata district, Italy, which were largely owned by German interests, were taken over by the Italian Government after Italy's entry into the World War. While under the control of the Austro-Hungarian government, the mines and reduction works at Idria employed a number of able engineers, and as a result, notable advances in the metallurgy of quicksilver were made there.

In the United States the quicksilver deposits which can supply domestic needs for many years to come are of course privately owned; and the industry has so far received little, if any, assistance from the Federal Government, except in the matter of tariff protection.[4] An opportunity to be of assistance was offered the Bureau of Mines in 1917, when certain quicksilver operators requested the aid of the bureau in making a study of quicksilver condensers, with particular reference to eliminating sources of loss that might exist in the treatment of low-grade ores. The matter was investigated by the bureau jointly with several operating companies in California and the results have been published.[5]

GENERAL CONCLUSIONS.

The main conclusions to be drawn from the work discussed in this bulletin may be summarized as follows:

In the direct-furnace treatment of quicksilver ores the major problem in the extraction of quicksilver has been solved. Methods are available whereby low-grade ore can be treated with a remarkably high recovery and at low cost in view of the small scale of operations at most plants. It does not follow that present practice

[4] The tariff act of 1913 carried a 10 per cent ad valorem duty. The present act, operative since October, 1922, imposes a tariff of 25 cents a pound on quicksilver, equal to $18.75 a flask, with corresponding duties on mercurials.

[5] Duschak, L. H., and Schuette, C. N., Fume and other losses in condensing quicksilver from furnace gases: Tech. Paper 96, Bureau of Mines, 1918, 29 pp. Reprinted in Min. and Sci. Press, vol. 117, 1918, p. 315.

at all or even at the majority of the quicksilver reduction works in this country has reached the highest possible point of efficiency. Improvements can be made at many plants mainly by correcting minor defects rather than by making fundamental changes in the process used.

In the past the quicksilver industry has suffered from lack of competent technical supervision, and some time and effort have been wasted through attempts to devise improvements in process and equipment without adequate regard for developments in other branches of metallurgy. Now knowledge of the metallurgy of quicksilver has advanced so far that adequate information is available for the design, construction, and operation of a plant for the treatment of any ordinary mercury-bearing ore. A few unusual cases which require further investigation are mentioned in this bulletin, but generally speaking there is no difficulty in recovering the metal from its ores effectively, and improvements in practice will consist mainly in applying available information more efficiently.

Mining of the ore offers the greatest opportunity for reducing production costs. To bring the ore to the reduction works generally costs two or three times as much as to treat it. This expense is due partly to the mode of occurrence of quicksilver deposits, and partly to the fact that mining operations are seldom conducted with maximum efficiency. Fluctuations in the price of and demand for quicksilver have tended to prevent operators from carrying on development in advance of actual mining; in consequence, the planning of a systematic mining program has been impossible. As an outcome of activities during the World War, however, a number of properties now have fair quantities of ore definitely blocked out. The authors believe that the greatest opportunity for increasing the economy in quicksilver production lies in giving more attention to the geology of the deposits and the improvement of mining methods.

ACKNOWLEDGMENTS.

Acknowledgment is gratefully made of the assistance of a number of quicksilver producers. Those in California are: H. W. Gould, former general superintendent of the New Idria Quicksilver Mining Co.; Murray Innes, manager of the Oceanic and Oat Hill quicksilver mines; and Clifford G. Dennis, president and manager of the St. John's Mining Co. Similar acknowledgment is made to Andrew Rocca, general superintendent of the Cloverdale and Helen mines; W. D. Burcham, owner and operator of several mines in the Terlingua district of Texas; and to the many officials and employees at the plants visited. The authors received uniform

courtesy and at all plants except one have been given every facility for investigation, including access to cost and confidential records.

The authors are particularly indebted to H. W. Gould for a number of photographs showing European quicksilver operations, and for first-hand information which enabled them to confirm and revise data obtained from various sources in regard to present practice on the Continent.

In preparing the section on the concentration of quicksilver ores, the authors were assisted by Thomas Varley, superintendent of the Intermountain experiment station of the Bureau of Mines. Dr. R. R. Sayers, chief surgeon of the Bureau of Mines, has collaborated in preparing that part of the text dealing with health hazards in quicksilver operations. The following members of the Bureau of Mines rendered aid during the investigation: G. N. Libby, assistant metallurgist; W. C. Riddell, chemical engineer; and C. M. Bouton, now associate research chemist at the Pittsburgh experiment station. The investigation covered by this report was carried out through the Pacific experiment station at Berkeley, Calif.

HISTORICAL DEVELOPMENT OF QUICKSILVER METALLURGY.

Of all the metals, quicksilver is probably the most easily recovered from the ore, as it can be volatilized at a comparatively low temperature, and thus separated from nearly all other substances that might be present in the ore. For this reason, the principle on which the extraction of quicksilver is based, a simple roasting operation, is the same to-day as in early times. The oldest records indicate that quicksilver was first recovered by crude heap roasting, and later by distillation in clay retorts. The earliest comprehensive description of the treatment of quicksilver ore, cinnabar, is given in Agricola's De Re Metallica,[6] first published in 1556. Agricola mentions five methods of retorting ore, which differ only in the size, shape, and method of heating the clay retorts. The quicksilver vapor was condensed on the cover of the retort or in a second vessel, and sometimes dropped into ashes or sand from which it was subsequently panned.

Spirek[7] has comprehensively reviewed the development of quicksilver metallurgy up to 1906. At Idria, formerly in Austria but now in Italy, clay retorts were used from 1530 until 1641, when the first iron retorts were installed. The practice of mixing lime with

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[6]Agricola, Georgius, De re metallica. Translated by H. C. and L. H. Hoover, London, 1912, book 9, p. 426.

[7] Spirek, Vinzenz, Über der Quecksilber Hüttenwesen, seine Geschichte und Entwickelung und über die Bergwerke in dem Quecksilbergebiete am Monte Amiata: Chem. Ztg., Jahrg. 30, Bd. 1, 1906, p. 452.

the ore was begun in 1850. The arrangement of retorts in benches and grate firing were introduced in 1696.

Lopez Saavedra Barba at Huancavelica, Peru, invented the shaft furnace in 1633; Bustamente introduced it at Almaden, Spain, in 1646. Plate II, *A*, shows the first of these furnaces, named the San Juan and Plate II, *B*, a general view of the furnaces. Between the years 1646 and 1654, 10 Bustamente furnaces were built at Almaden, all of which are still in use. Plate II, *C*, shows a furnace built in 1654.

Reverberatory furnaces were first used about 1842 at Idria, and the shaft furnace was introduced there by Hähner in 1849. Muffle furnaces were tried from 1869 to 1882. In 1872 the Exeli continuous shaft-furnace was invented at Idria; in 1874 it was introduced in California, where externally fired intermittent shaft-furnaces had been used.

In 1874, the Knox continuous shaft-furnace for the treatment of both fine and coarse ores was first used in California. A year later came the invention of the Scott furnace. The development of this furnace, in which fine ore could be continuously and economically treated, marked an important advance in quicksilver metallurgy. Cermak, at Idria, Austria, later took up the principle of the Scott furnace and incorporated it in his design, erecting his first furnace in 1880.

Goodyear,[8] Egleston,[9] and Schnabel,[10] have discussed early quicksilver practice in California. The last named gives a full review of the metallurgy of quicksilver.

Recent developments in quicksilver metallurgy have been determined largely by the need of economic methods of handling large quantities of low-grade ore. Except as a means for developing prospects and for treating small isolated deposits of high-grade ore, the retort has come to be an adjunct for the treatment of soot at large furnace plants. The use of coarse-ore furnaces also has been almost entirely discontinued.

The problem of handling large quantities of low-grade ore has naturally directed attention to mechanical furnaces, and in 1916 a multiple-hearth furnace of the McDougall type was successfully used for roasting quicksilver ore. The use of the rotary cylindrical furnace of the familiar cement-kiln type was first suggested [11] as far back as 1876. Although several attempts were made to use a

[8] Goodyear, W. A., Geology of California: California Geol. Survey: vol. 2, Appendix G, 1882, pp. 91–135.

[9] Egleston, Thomas, The metallurgy of silver, gold, and mercury in the United States. New York, 1890, vol. 2, pp. 799–901.

[10] Schnabel, Carl, Handbook of metallurgy. (Translated by Henry Louis.) New York, 1907, pp. 329–443.

[11] Purnell, Samuel, Remarks on quicksilver: Min. and Sci. Press, vol. 33, Oct. 28, 1876, pp. 240–276; 289–321; 352.

furnace of this type, the first real success was not achieved until 1917. This furnace bids fair to be decidedly important in the future of the industry.

Attempts to concentrate cinnabar ore were made as early as 1871, and sporadic efforts have continued to the present time. With a few exceptions, however, concentration has not proved successful on a large scale. The hydrometallurgic treatment of quicksilver ores is still in the experimental stage, with little prospect of any practical development in the near future. Bradley [12] has discussed at some length the concentration and wet treatment of quicksilver ores, also the development of general metallurgical practice in California up to 1918.

ORES OF QUICKSILVER.

From the standpoint of the metallurgist, the only important quicksilver minerals are cinnabar and native quicksilver. Pyrite and marcasite often are associated with the cinnabar, as are small quantities of other sulphides, including those of arsenic and antimony. Native sulphur and bituminous matter also occur in some ores. Ransome [13] has described the various minerals containing mercury.

Quicksilver ore is found in rocks of all geologic ages and all classes; hence the gangue material of the ore differs in different mines and often in the same mine. The common gangue rocks are limestone, calcareous shales, sandstone, serpentine, chert, andesite, basalt, and rhyolite. The great variety of gangue minerals means a corresponding variation in the physical characteristics of the ore. Two general types of cinnabar ore can be distinguished, namely: (1) Disseminated ore, in which the cinnabar has impregnated a more or less fine-grained or highly brecciated gangue; and (2) ores deposited in fissures and cracks of the country rock. In at least one body of disseminated ore the cinnabar was precipitated simultaneously with the amorphous silica cementing a brecciated rock. The second type of ore merges into the first as the fissures and cracks become very minute; at its other extreme are large veins and bodies of almost pure cinnabar, sharply separated from the inclosing gangue.

Both types of ore may form high-grade or low-grade ore bodies, may be hard or soft, may break into coarse or fine fragments, may show the red cinnabar plainly or may not, and may contain any of the associated minerals already mentioned.

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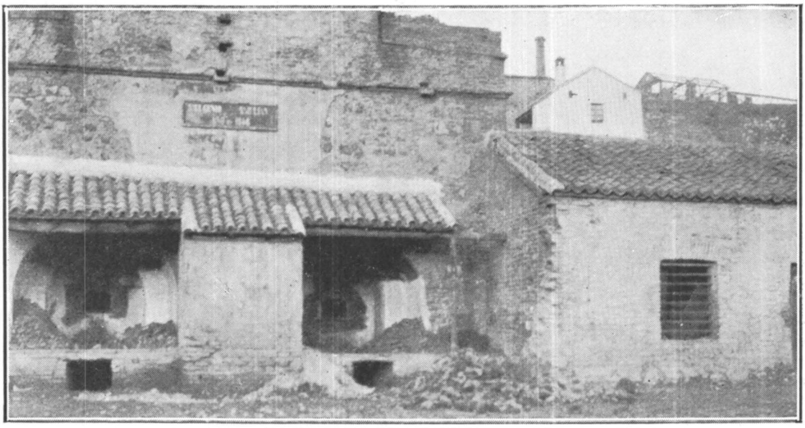
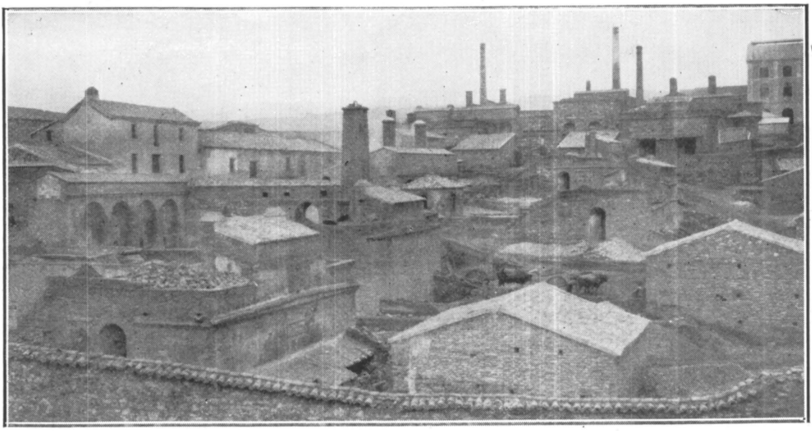
[12] Bradley, W. W., Quicksilver resources of California: California State Min. Bur. Bull. 78, 1918, pp. 209–352.

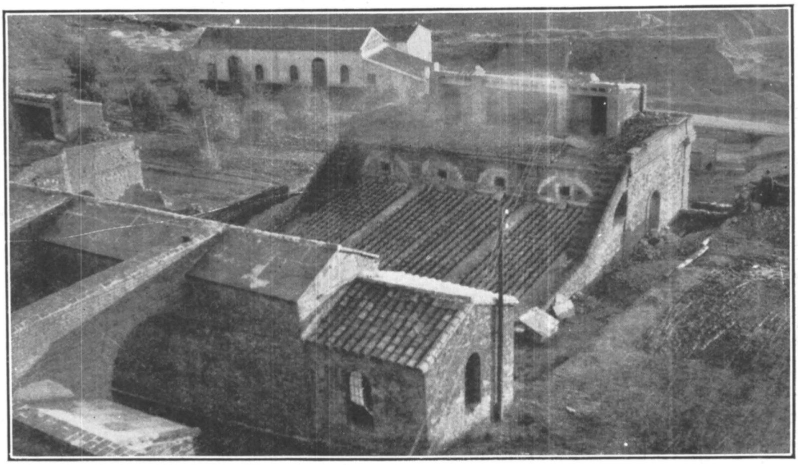
[13] Ransome, F. L., Quicksilver in 1917: U. S. Geol. Survey Mineral Resources, 1917, pt. 1, 1919, pp. 390–393.



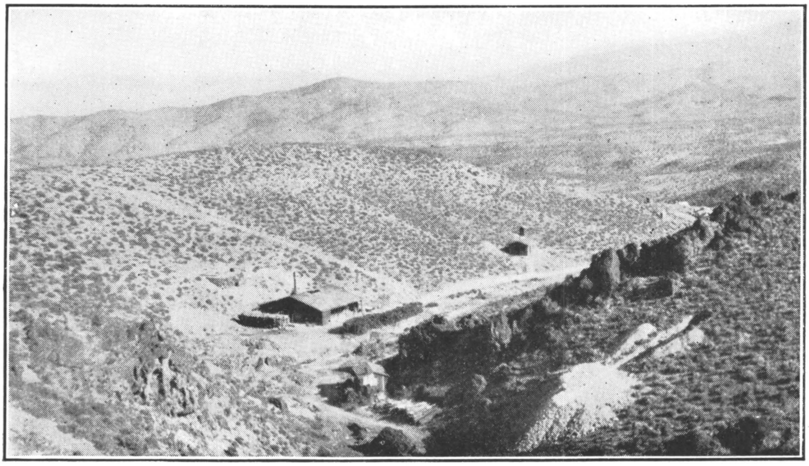
A. ORIGINAL BUSTAMENTE FURNACE BUILT AT ALMADEN, SPAIN, IN 1646 AND STILL OPERATING



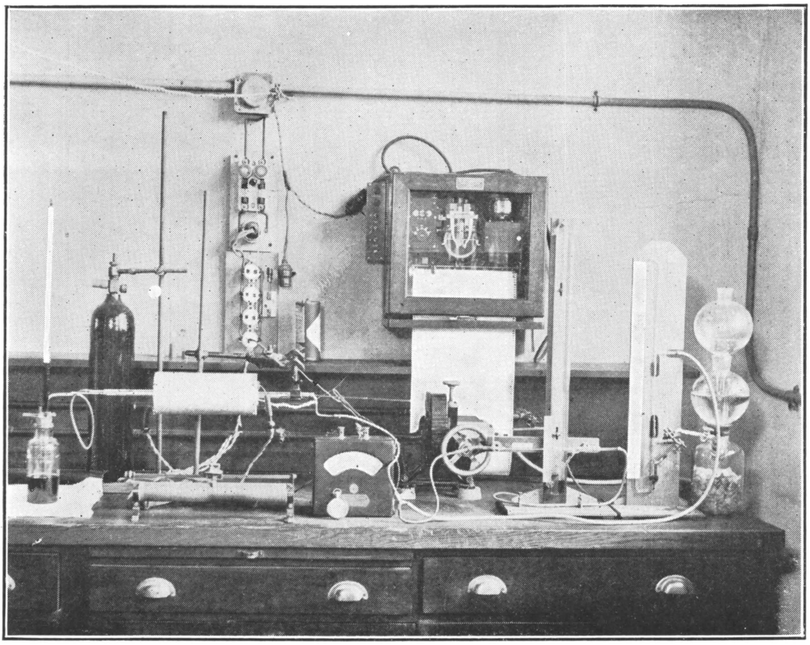
B. GENERAL VIEW OF FURNACES SHOWN IN *A.* BULLOCK CARTS ARE REMOVING ROASTED ORE



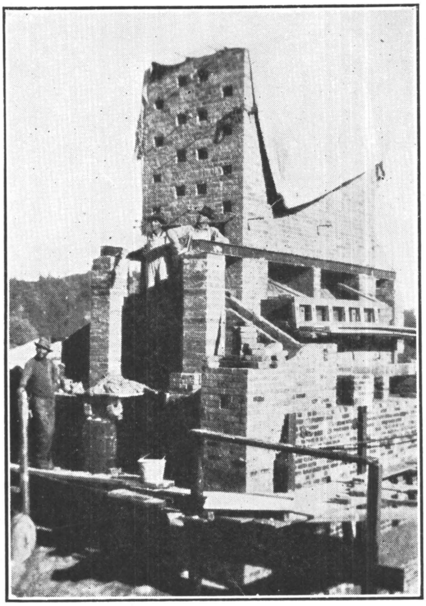
C. NEWEST BUSTAMENTE FURNACE AT ALMADEN, SPAIN, BUILT IN 1654 AND STILL OPERATING



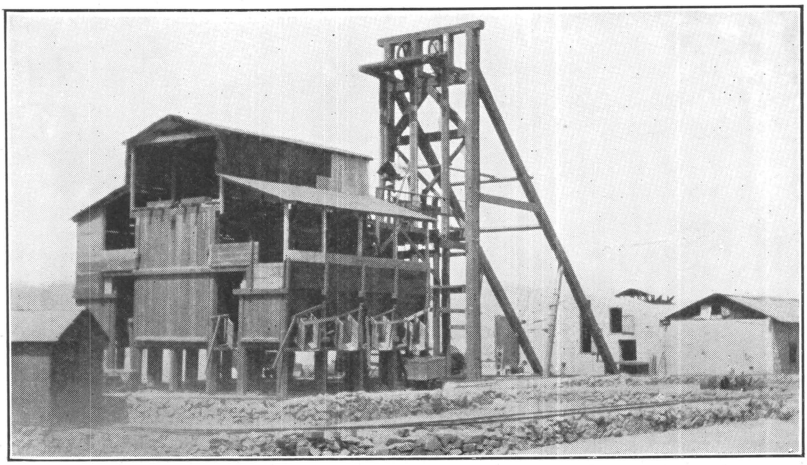
A. TYPICAL QUICKSILVER PROSPECT IN NEVADA. THE OUTCROP, TUNNEL, AND RETORT BUILDING ARE SHOWN



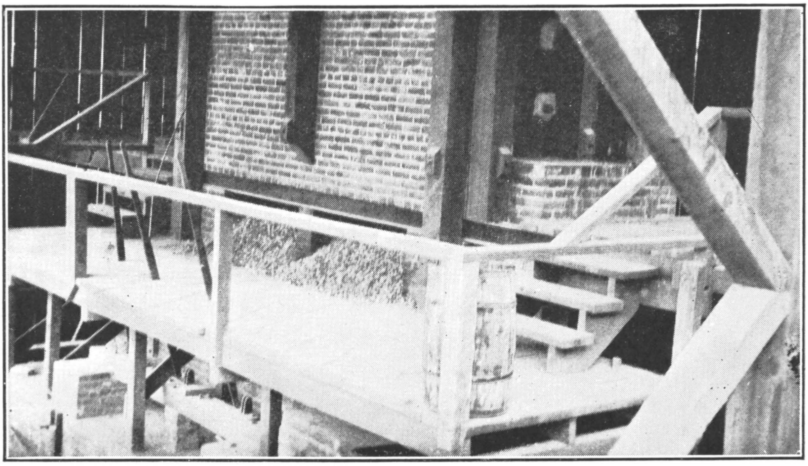
B. APPARATUS FOR ORE-ROASTING EXPERIMENTS



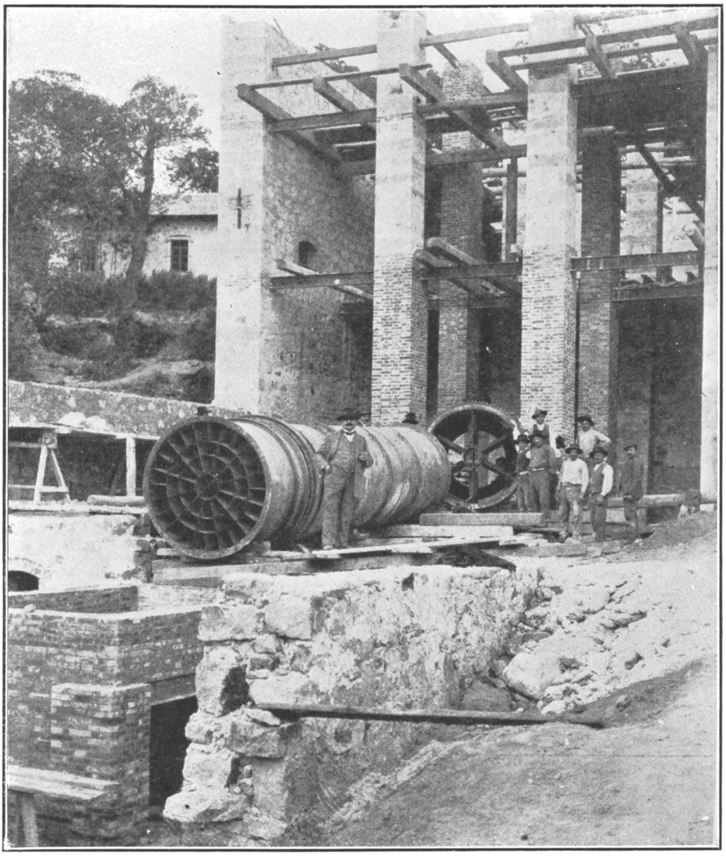
A. ORE DRIER UNDER CONSTRUCTION ON TOP OF SCOTT FURNACE



B. ROCK HOUSE AND HEADFRAME OF BIG BEND MINE, BREWSTER COUNTY, TEX.



A. DRIER DISCHARGE AND CHARGING HOPPER OF SCOTT FURNACE, OAT HILL MINE, CALIFORNIA



B. CELLULAR ROTARY ORE-DRIER BEING ERECTED IN MONTE AMIATA DISTRICT, ITALY

Metallurgically, quicksilver ores may be classified with reference to their crushing behavior and their texture, as these physical characteristics affect the ease with which the vapors of mercury and mercuric sulphide are released. Important features of chemical composition are the amount of sulphur, both free and combined, and the amount of bituminous material.

In general, the quicksilver ores of the United States do not carry other metallic compounds, such as those of arsenic and antimony, in sufficient quantity to influence their metallurgical treatment. A pyritic ore body recently developed in Oregon seems to be an exception, as its content of arsenic is comparable to its content of quicksilver. Plate III, *A*, shows a quicksilver prospect in Nevada.

BEHAVIOR OF QUICKSILVER ORES IN ROASTING.[14]

INTRODUCTION.

As already indicated, quicksilver ores are almost universally treated by a simple roasting process, the net result of which is the expulsion of the mercury, in the form of vapor, from the ore. In furnace treatment, with any of the several types of direct-fired furnace, an excess of air is usually present; in retorting, however, oxygen is likely to be largely or entirely absent. Thus the behavior of quicksilver ore must be considered under these two sets of conditions. In discussing this subject it is well to have in mind, first of all, some of the physical and chemical properties of mercury and the crystalline mercuric sulphide, cinnabar, which is the principal ore mineral.

VAPOR PRESSURE OF MERCURY AND CINNABAR.

The vapor pressure of mercury, expressed in terms of the mercury manometer, is roughly 0.28 mm. at 100° C., 17.5 mm. at 200° C., and 760 mm. at 357.3° C., the boiling point. At any temperature above the boiling point, mercury vapor will exert a pressure greater than one atmosphere. When ore containing free mercury is heated above this temperature, the mercury vapor will tend to force its way out through crevices in the ore. Below this temperature, evaporation will take place, the rate depending on the temperature and the speed with which the mercury vapor is removed by diffusion from the surface of the evaporating metal.

The temperature at which the vapor pressure of the crystalline mercuric sulphide, cinnabar, reaches one atmosphere was determined by Allen[15] and Crenshaw to be about 580° C. Some approximate

[14] Except where otherwise stated, the experiments described in this chapter were made by C. M. Bouton of the Bureau of Mines.

[15] Allen, E. T., and Crenshaw, J. T., The sulphides of zinc, cadmium, and mercury: Am. Jour. Sci., 4th ser., vol. 34, 1912, pp. 341-396.

determinations of the vapor pressure of mercuric sulphide at low temperatures were made by the authors[16] of this bulletin. Artificial cinnabar, prepared by sublimation of the black sulphide in evacuated glass tubes, was used, also selected samples of the natural mineral. The two materials gave identical results. The method used depended on determining the weight of sulphide needed to saturate a measured volume of nitrogen at a given temperature. The results of these determinations, together with the above value for the subliming point of the sulphide, are shown in Figure 1. During these vapor-pressure determinations some evidence of the dissociation of the vapor was observed, but no quantitative data as to the extent of the dissociation are available.

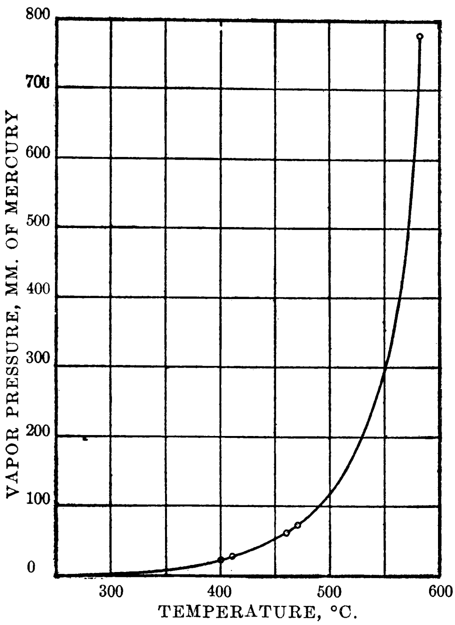


FIGURE 1.—The vapor-pressure curve of cinnabar.

The melting point of cinnabar is not known; according to the observation of Allen[17] and Crenshaw, it lies above 580° C., the subliming point. If sublimation alone is to be depended upon to extract the mercuric sulphide content of ore, evidently a temperature somewhat above 580° C. is required, although some evaporation of the sulphide will take place below this temperature.

REACTIONS BETWEEN CINNABAR AND OXYGEN.

When oxygen is in contact with the ore a considerable amount of mercury is liable to be released through the oxidation of the sulphide; sulphur dioxide and mercury vapor are formed. Friedrich[18] has shown that cinnabar begins to oxidize in air at as low a temperature as 230° to 300° C., and that at 350° to 400° C. the reaction is fairly rapid. These conclusions were confirmed by experiments made in a small tube furnace with natural cinnabar of high purity. (See Pl. III, *B*.) A weighed quantity of cinnabar was placed in a glass boat within a glass tube, and a measured stream of air was

[16] Schuette, C. N., Thesis for B. S. degree, College of Mining, University of California, Berkeley, May, 1917.

[17] Allen, E. T., work cited, pp. 341–396.

[18] Friedrich, K., Thermische Daten zu den Röstprozessen. Metallurgie, vol. 6, 1909, p. 169.

passed over it for various periods and at various temperatures. The condensed products of the reaction were collected in the extended end of the glass tube, which served as a condenser.

Oxygen reacts with mercuric sulphide as follows:

$$HgS+O_2=SO_2+Hg$$

At 400° C., half of the cinnabar (0.250 gram) was decomposed in one hour; whereas at 450° C., with the same air flow, the same amount was decomposed in 10 minutes. In all tests, oxygen was present in the escaping gas; the amount was greater in the experiments at 400° C. than in those at 450°. The greatest utilization of oxygen, amounting to nearly 90 per cent, was observed in an experiment at 450° with a low air velocity. The failure of the oxygen to react completely is not interpreted as due to the establishment of any equilibrium, but rather to the lack of enough contact between the gas stream and the cinnabar.

The question arises whether the reaction took place chiefly between the oxygen and the solid mercuric sulphide, or in the vapor phase between the oxygen and the mercuric sulphide vapor. It was observed that at any given temperature, the amount of cinnabar reacting in unit of time increased nearly but not quite in proportion to the rate of flow of air; in other words, as the rate at which the air was passed increased, the quantity of cinnabar reacting in unit quantity of air decreased slightly. Such variation is usually observed when an inert gas is passed over a volatile substance. The experiment indicates that the reaction between the oxygen and the cinnabar takes place largely in the vapor phase; that the cinnabar evaporates at a rate proportional to its vapor pressure, which depends upon the temperature; and that the rate of the resulting vapor reactions probably varies with the oxygen present.

At 350° and 400° C., about 95 per cent of the sulphur, corresponding to the weight of cinnabar that disappeared, was collected in the form of sulphur dioxide. At 450° C., however, approximately 50 per cent of the sulphur was collected in the form of sulphur trioxide.

In addition to the simple reaction given, the oxidation of mercuric sulphide by air may also result in the formation of a sulphate of mercury. In the experiments just referred to, a little white crystalline material was always found associated with the mercury in the condensing tube. There was not enough for accurate analysis, but tests indicated that it was mercurous sulphate. In the experiments at 350° to 400° C., the residue of cinnabar in the boat was usually covered with a frosting of white stellate crystals. This was not observed at 450° but sulphate was always deposited with the mercury in the condensing tube. This material could not be separated from the free mercury, but, in general, at 400° not more than 5 per cent of the total mercury collected in the condenser was combined as sulphate. At 450°, the limit was about 3 per cent. Part, at least, of the mercury so combined was present as mercurous sulphate. The following reactions were involved:

$$HgS+2O_2=HgSO_4 \text{ (ic)}$$
$$2HgS+3O_2=Hg_2SO_4+SO_2 \text{ (ous)}$$

These reactions are of interest in connection with incrustations of the sulphates of mercury often found in the upper part of the Scott furnace, and in that part of the condenser system adjacent to the furnace. The fact that in the above experiments more sulphates were formed at 350° and 400° C. than at 450°, suggest that there is an optimum temperature for this reaction. Further, the fact that at the two lower temperatures some of the sulphates appeared as an incrusation on the cinnabar, suggests that their formation may have been due to a reaction taking place on the surface of the cinnabar. It is conceivable that this surface acts as a catalyzer in promoting the oxidation of sulphur to sulphur trioxide. If this is true, the decrease of this reaction at 450° may be in part due to the fact that the cinnabar evaporates more rapidly owing to its higher vapor pressure at this temperature and the vapor so formed tends to prevent contact between the oxygen and the solid cinnabar.

This suggests that the rapid heating of the ore with the consequent rapid evaporation of the cinnabar would reduce the amount of sulphates formed. The formation of sulphates of mercury in the reaction between the atmospheric oxygen and cinnabar has its parallel in the "sulphatizing roast," which has been developed in the treatment of silver, copper, and zinc ores. In ordinary quicksilver furnace practice, enough sulphates do not form to be metallurgically important.

The apparatus on page 7 used to study the reaction between cinnabar and atmospheric oxygen, was also used to determine the effect of carbon dioxide and sulphur dioxide on the roasting operation. Carbon dioxide appeared to exert little influence. Twelve per cent by volume of sulphur dioxide slightly retarded the reaction between the cinnabar and the oxygen. There was no observable effect of either gas on the quantity of sulphates formed.

CHEMICAL REACTIONS IN RETORTING CINNABAR ORE.

When cinnabar or artificial mercuric sulphide is roasted with very little or no air, as when ore, concentrate, or soot is retorted, chemical changes other than those involving the oxidation of the sulphide are depended upon to separate the mercury from the sulphur. In the complete absence of oxygen, very little mercury will be released by the charge until the subliming point of cinnabar, 580° C., is reached, at which point the evolution of mercuric sulphide vapor will begin more or less rapidly, depending upon the porosity of the material. In retort practice mercury is usually released from its combination with sulphur by adding lime to the charge. The chemical reaction is as follows:

$$4HgS+4CaO=3CaS+CaSO_4+4Hg$$

Laboratory tests showed that in the absence of oxygen the reaction occurs essentially as indicated by the above equation. Other sulphur compounds of calcium may also be formed.

Scrap iron is sometimes used in the retort charge; the reaction is essentially as follows:

$$HgS+Fe=FeS+Hg$$

Unless enough lime or other material is present in the charge to react with the mercuric sulphide vapor, the walls of the retort will be attacked according to the reaction just given.

Certain common constituents of quicksilver ore, such as ferric oxide, can also remove the sulphur from mercuric sulphide. The following reaction would take place largely between the mercuric sulphide vapor and the solid oxide of iron:

$$7HgS+2Fe_2O_3=4FeS+3SO_2+7Hg$$

The admixture of magnetic oxide with an ore sample containing sulphur has been recommended in connection with the Eschka method of assaying quicksilver ores.

ROASTING EXPERIMENTS.

In order to obtain data to help interpret observations made in running large quicksilver furnaces, a series of roasting experiments was carried out in the laboratory. A summary of the results obtained is given in Table 1.

TABLE 1.—*Roasting experiments with quicksilver ores.*

Test No.	Ore sample.	Diameter of grains.	Quantity.	Time.	Temperature.	Hg eliminated.
		Mm.	*Grams.*	*Min.*	*°C.*	*Per cent.*
1	L125	10–14	50	15	435	74
2	do	1.5–5	50	15	435	78
3	do	1.5 or less	50	15	435	78
4	do	10–14	50	25	440	92
5	do	1.5–5	50	25	440	98
6	do	1.5 or less	50	25	440	99
7	do	10–14	50	8	550	96
8	do	1.5–5	50	8	550	99
9	do	1.5 or less	50	8	550	99
10	do	10–14	50	12	545	99.4
11	do	1.5–5	50	12	545	99.8
12	do	1.5 or less	50	12	545	99.8
13	L139	10–14	50	25	450	83
14	do	1.5–5	50	25	450	96

TABLE 1.—*Roasting experiments with quicksilver ores*—Continued.

Test No.	Ore sample.	Diameter of grains.	Quantity.	Time,	Temperature.	Hg eliminated.
		Mm.	*Grams.*	*Min.*	° *C.*	*Per cent*
15	L139	1.5 or less	50	25	450	100
16	do	10–14	50	10	545	88
17	do	1.5–5	50	10	545	100
18	do	1.5 or less	50	10	545	100
19	L138	10–14	50	25	450	49
20	do	1.5–5	50	25	450	58
21	do	1.5 or less	50	25	450	90
22	do	10–14	50	25	550	34
23	do	1.5–5	50	25	550	83
24	do	1.5 or less	50	25	550	96
25	do	7	12+	10	450	25
26	do	7	12+	30	450	40
27	do	7	12+	90	450	44
28	do	7	12+	30	600	63
29	do	7	12+	90	600	68
30	do	7	12+	360	600	92
31	L125	18 by 32 by 38	Lump	30	450	66
32	do	22 by 25 by 25	do	60	450	92
33	do	18 by 25 by 38	do	15	500	29
34	do	28 by 28 by 38	do	30	500	99
35	do	22 by 25 by 28	do	15	550	97
36	do	18 by 25 by 28	do	30	550	93
37	do	25 by 25 by 44	do	60	550	99.5
38	L180	25 by 38 by 38	do	10	450–485	67
39	do	18 by 25 by 25	do	15	500	100

Experiments 1 to 24, inclusive, were made in a cast-iron muffle; the ore was spread in a layer about 10 mm. deep in thin sheet-iron trays. The time is counted from the introduction of the cold charge into the already heated muffle. The temperature is that indicated by a thermocouple embedded in the ore, after practically constant conditions had been established. The time required to reach this constant temperature was from six to eight minutes.

Sample L125 was a friable sandstone with cinnabar particles distributed with fair uniformity. Sample L139 was similar in character to L125, but contained a small percentage of pyrite. In sample L138 the cinnabar was largely disseminated through silica, probably chalcedony, with which the original breccia had been recemented. This ore was very hard and fine grained.

The samples picked for the experiments were crushed in a small jaw crusher and screened into three sizes; they had the following mercury content:

Mercury content of ores tested.

Sample No.	Size and mercury content of material.		
	10–14 mm.	1.5–5 mm.	−1.5 mm.
	Per cent.	*Per cent.*	*Per cent.*
L125	1.73	2.12	2.66
L139	.21	.24	.22
L138	1.48	1.57	1.56

Experiments 25 to 30, inclusive, were made with selected pieces of sample L138 in a small electrically heated crucible furnace equipped with a special iron muffle, which permitted rapid heating of the charges and close temperature control. The average mercury content of the selected material was 1.17 per cent. Experiments 31 to 39, inclusive, were made in the same apparatus with single lumps of ore. Rectangular pieces, in which the cinnabar appeared to be uniformly distributed, were selected. Each piece was cut in two; one half was assayed and the other half used for the roasting experiments. The lumps of sample L125 had a mercury content of approximately 2 per cent. Sample L180, used for experiments 38 and 39, was a friable sandstone carrying about 0.5 per cent of mercury. In all of these experiments a slow current of air circulated over the roasted ore.

With all of the samples of sandstone ore, including even the large lumps, considerably more than half the mercury content of the ore was expelled in a comparatively short time, even at 440° to 450° C. At this temperature the vapor pressure of cinnabar is about 50 mm., and therefore the release of mercury from the ore seems likely to have taken place to a greater extent through the diffusion of oxygen into the crevices of the ore, followed by oxidation of the cinnabar, than by the diffusion of mercury sulphide vapor to the surface of the lumps. As would be expected, the larger sizes of ore give up their mercury somewhat slower than fine material; but with the sandstone ores, as will be seen by comparing experiments 35, 36, 37, with 7 to 12, inclusive, the effect of the size of the ore particles is not particularly marked.

The results obtained with sample L138, a dense siliceous ore, are strikingly different from the friable sandstone, especially as regards the rate of freeing the mercury. It will be noted that the influence of the size of ore particles is much more marked than with the sandstone ores. As will be seen from experiment 25, a certain amount of the mercury is released rather readily by the ore, but a long time and relatively high temperature are required for anything like complete liberation. The rate of diffusion of the mercuric sulphide vapor through the dense silica is apparently the controlling factor. The escape from the surface layers takes place readily, but even at 600° C., where the vapor pressure of the cinnabar exceeds one atmosphere, the diffusion of the vapor through a layer of dense silica 2 or 3 mm. thick is a matter of hours. For furnace treatment, the requirements of this ore are evidently quite different from the sandstone ores, and in actual practice, with the McDougall type of mechanical furnace, the ore had to be crushed to ¼-inch size.

In order to gain some further insight into the process of liberation of the mercury from lumps of ore, small cylindrical briquets about 28 mm. in diameter and 25 mm. thick were made of Portland cement and crushed granite, with a weighed quantity of mercuric sulphide in the center of each. The briquets weighed about 30 grams apiece, and contained from 0.2 to 0.3 grams of mercury, corresponding to an ore carrying from 1 to 1.5 per cent. The briquets of series A and B were identical, except that series B contained about 0.3 gram of pyrite mixed with mercuric sulphide. The briquets of series C were made of a mixture of granite sand about 0.5 mm. in diameter, mixed with 20 per cent of pyrite of the same size. The absence of fine material made these briquets more porous than those of series A and B. The briquets were heated in a small iron muffle, through which a current of air passed at the rate of approximately 100 c. c. a minute. A few of the results are given in Table 2.

TABLE 2.—*Roasting experiments with briquets of cement and granite containing mercury.*

Temperature.	Time.	Quantity of mercury freed.		
		Series A.	Series B.	Series C.
° C.	*Min.*	*Per cent.*	*Per cent.*	*Per cent.*
500	10	4	6	2
500	20	14	24	12
500	30	24	96	26
600	5	.3	8	3
600	10	14	33	15
600	15	39	87	57

The temperature of the muffle was kept as constant as possible throughout the experiment. A thermocouple imbedded in one of the briquets showed that from 10 to 14 minutes elapsed before the temperature became approximately uniform throughout the briquet.

A comparison of the results with series A and B shows that the briquets of series B, which contained a small amount of pyrite mixed with the mercuric sulphide at the center of the briquets, gave up their mercury much more readily. Apparently the sulphur vapor from the pyrite must have aided the escape of the mercuric sulphide vapor. This explanation is not entirely satisfactory, as the vapor pressure of sulphur from pyrite does not become equal to one atmosphere until a temperature of 675° C. is reached. The temperatures of these roasting experiments were well below this but were still within the range where some sulphur vapor is evolved by pyrite. The results with series C are not significantly different from those of series A.

An examination of the briquets of series C after roasting showed that the pyrite has been but slightly decomposed, which indicates that very little oxygen had diffused into the briquets. This would argue that the escape of the mercury took place largely through the diffusion of mercuric sulphide vapor to the surface of the briquet, which agrees with Oschatz's [19] conclusion.

In this connection some roasting experiments with an ore containing a considerable amount of pyrite are of interest. The analysis is as follows: Mercury, 1.76 per cent; sulphur, 22.3 per cent; iron (Fe), 19.5 per cent; and insoluble, 49.6 per cent. The ore was crushed to pass a 1-inch ring; material finer than one-half inch was rejected. The experiments were made in an electrically heated muffle, the ore being spread out in a layer 25 mm. deep in shallow iron trays. Material roasted for a half-hour at 700° C. with a limited supply of air had the following composition: Test No. 1, mercury, 0.02 per cent, and sulphur, 11.6 per cent; and test No. 2, mercury, 0.03 per cent, and sulphur, 14.5 per cent.

If allowance is made for the loss of weight of the material in roasting, evidently only a little more than half of the sulphur was expelled from the ore. This corresponds to the distillation of the major portion of the so-called first atom of the sulphur from the pyrite, with the formation of pyrrhotite. The results show what can be accomplished by a quick roast of an ore of this character.

In all of the experiments described above, the material contained at most only a small percentage of mercury, corresponding to a relatively low-grade ore. With large nuggets of cinnabar, the time required to transmit the heat absorbed as latent heat of evaporation of the sulphide would become an important factor and a longer roasting period would be required.

The above experiments show that there is a wide variation in the roasting behavior of quicksilver ores, depending upon the physi-

[19] Oschatz, K., Die Verhüttung der Zinnobererze am Monte Amiata: Glückauf, Jahrg. 54, 1918, p. 547.

cal characteristics, and, to a lesser extent, upon the chemical composition of the ore. A low-grade porous sandstone ore in the form of 1-inch pieces will evidently give up its mercury in from a half to one hour, at a temperature of 500° to 600° C. With a dense and mechanically strong ore, the roasting period would be several times greater.

BEHAVIOR OF ARSENIC AND ANTIMONY IN QUICKSILVER ORES

Compounds of arsenic and antimony are about the only volatile materials other than sulphur and bituminous matter that are liable to influence the metallurgical behavior of the ore. Of the two metals, arsenic is the more liable to cause complications, as vapor-pressure curves of arsenic trioxide and mercury are not widely different. Vapor pressures for a few temperatures are given in the following table; those for arsenic trioxide are taken from the work of Welch and Duschak.[20]

Vapor pressure of mercury and arsenic trioxide.

Temperature, ° C.	Vapor pressure, mm.	
	As_2O_3.	Hg.
160	0.05	4.2
200	.65	17.5
300	89.1	246

The boiling point of mercury is about 357° C., and that of arsenic trioxide, as obtained from the work of Welch and Duschak, about 355° C. Evidently when arsenical quicksilver ore is roasted in the presence of oxygen, the arsenic trioxide will distill over and condense with the mercury in the condenser system.

At certain Spanish mines[21] the ore contains enough arsenic to require special treatment. In general, quicksilver ores in the United States contain little or no arsenic, but recently in Oregon a deposit of pyritic ore has been developed that contains enough arsenic to complicate the metallurgy. An attempt was made to treat this ore in a small Scott furnace. The condenser product consisted of a gray soot, containing little free-running mercury and as much as 70 per cent arsenic trioxide. Attempts to recover the mercury

[20] Welch, H. V., and Duschak, L. H., The vapor pressure of arsenic trioxide: Tech. Paper 81, Bureau of Mines, 1915, 22 pp.

[21] Mineral Industry, vol. 4, 1895, p. 524.

by treating this soot in a retort were not successful, which might have been foreseen from the vapor-pressure relationships mentioned. Some experiments made under the author's direction indicated that separation was possible by pulping the soot with 5 to 10 parts by weight of water, and gently agitating the mixture. The free mercury gradually coalesced and settled to the bottom, the arsenic trioxide remaining in suspension.

The vapor pressures quoted indicate that when arsenic trioxide and mercury vapors are proportionally equal in the furnace gas, the arsenic trioxide begins to condense at a higher temperature than mercury. This difference might make possible a part separation of these substances by suitably controlled condensing. A condenser would be needed with which the rate of cooling could be closely controlled, and the condensed material rapidly settled. The tile-pipe condenser, described on page 129, would probably best serve this purpose.

With reference to the behavior of antimony compounds, Hofman and Blatchford [22] have shown that stibnite begins to roast in an oxidizing atmosphere at a little above 300° C., also that the volatilization of the oxide is not appreciable until a temperature of 500° C. is reached. This would indicate that, as in the case of arsenic, any antimony present in the cinnabar would probably pass to the condenser system along with the quicksilver vapor. As distinguished from arsenic, however, the oxides of antimony have distinctly lower vapor pressures. Therefore they would tend to condense long before the mercury vapor, and a fairly good separation in the condenser should be possible.

Antimony in small quantities has been found in several quicksilver ores in the United States. A small percentage of this material was found in a sample of mercurial soot; but, as far as known, not enough has been present to cause trouble.

The oxides of antimony have a very low vapor pressure at the boiling point of mercury, therefore successful treatment in a retort of any condenser residue containing oxides of antimony should be possible.

The contamination of mercury by metallic arsenic or antimony has not so far been reported. It would not be expected that the oxides of either arsenic or antimony would be reduced by mercury vapor at ordinary furnace temperatures, and, as long as the ore is roasted in the presence of any excess of oxygen, no metallic arsenic or antimony should form.

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[22] Hofman, H. O., and Blatchford, J., The behavior of stibnite in an oxidizing roast: Trans. Am. Inst. Min. Eng., vol. 54, 1917, p. 671.

PRELIMINARY TREATMENT OF QUICKSILVER ORES.

INTRODUCTION.

The usual method of treating quicksilver ore by roasting in either a direct-fired furnace or retorts is so simple that comparatively little preliminary treatment is needed. The discussion which follows particularly refers to the preparation of ore for roasting either in the Scott furnace or in some form of mechanical furnace.

SORTING.

Ore is seldom hand sorted unless it breaks largely into lumps more than 3 inches in diameter. Generally speaking, quicksilver ore is easy to sort owing to the bright red color of the cinnabar. Some impure cinnabar has a brownish color, almost identical with that of

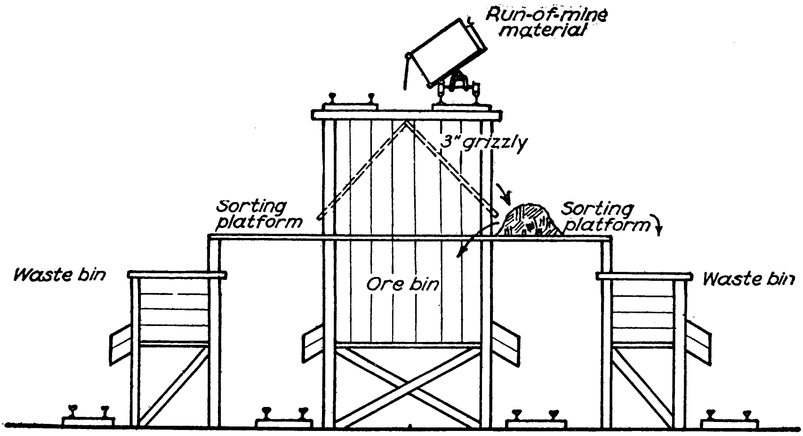

FIGURE 2.—Ore-sorting bin at the Big Bend mine, Brewster County, Tex.

certain forms of iron oxide, also in the ore. If the surface is wetted the cinnabar at once shows bright red and can be readily distinguished. That class of ore in which the cinnabar has been deposited in cracks and fissures lends itself most readily to hand sorting, whereas, with certain types of disseminated ore, selection is difficult. Sorting is done underground sometimes, but usually it is done at the surface either in specially equipped rock houses or on sorting platforms.

One of the best arrangements for hand sorting was at the Big Bend mine at Study Butte in the eastern end of the Terlingua district, Brewster County, Tex.; it is shown in Figure 2 and Plate IV, *B* (p. 6). As the sketch shows, all minus 3-inch material passes through the gable grizzly to the ore bin; the oversize is sorted and hand cobbed. The sorted ore is trammed to the crusher and the waste to the dump.

Four to six men are used for the sorting, and two men for tramming ore and waste. The oversize from 80 tons of run-of-mine material is sorted in about six hours, with a rejection of about 50 per cent of the total. The cost, based on the original tonnage, is 8 to 9 cents a ton, or 16 to 18 cents a ton on the 40 tons of sorted material. Mexican labor is used, costing $1.25 a day.[23] Therefore, the cost is 16 to 18 cents per ton to reject 40 tons of waste. This reject carries very little mercury, and, as the furnace treatment would cost several times as much as sorting, a big saving is made. Even though a considerably higher wage were paid, the sorting would be profitable. The advantage of sorting depends largely on the amount of quicksilver lost in the waste, as shown by the following example:

Comparison of profit in straight reduction and in sorting quicksilver ore.

Assumptions:	
Mercury content of ore, per cent	0.25
Cost of mining and development, per ton	$3.00
Cost of sorting, per ton	.10
Cost of furnacing, per ton	.75
Depreciation of plant per ton capacity, per ton	.50
Furnace recovery, per cent	95
Loss in sorting, per cent	10
Sorting ratio	50:50
Straight furnace reduction:	
Cost of mining 80 tons, at $3	$240.00
Cost of furnacing 80 tons, at 75 cents	60.00
Depreciation, 80 tons, at 50 cents	40.00
Total	340.00
Mercury recovered, 80 tons × 5 pounds × 95 per cent	380 lbs.
Value at $75 per flask	$380.00
Profit per ton	.50
Sorting and furnace reduction:	
Cost of mining 80 tons, at $3	240.00
Cost of sorting 80 tons, at 10 cents	8.00
Cost of furnacing 40 tons, at 75 cents	30.00
Depreciation, 40 tons, at 50 cents	20.00
Total	298.00
Mercury recovered, 40 tons × 10 pounds × 0.90 × 95 per cent	342 lbs.
Value at $75 per flask	$342.00
Profit per ton	.55

If the content of the ore is assumed to be 0.5 per cent, other factors remaining the same, sorting would not be profitable.

It may be mentioned in passing that the limestone-cinnabar ores of Texas lend themselves particularly well to close sorting.

[23] Hunley, J. B., Private communication from superintendent for the Study Butte Mining Co.

PICKING BELTS AND TABLES.

As far as is known, picking belts have not been used in connection with quicksilver operations in the United States, although judging from their extensive use with other ores, some of the larger plants might find them advantageous. At Idria, in Austria, an annular revolving picking table is used. Ore is fed onto this table intermittently from a chute. The waste is picked out, and the ore is then scraped off by an adjustable deflector. When the waste in the ore is so little that it can all be rejected in one revolution of the table, the deflector is kept on the table. If the amount of waste is large, the deflector is raised until the picking is done.

At Almaden, Spain, the ore is broken by hand and sorted according to size, but not grade.

CRUSHING.

The amount of crushing required by quicksilver ore varies considerably. For ordinary furnace treatment, 1 or 2 inch ore is needed. For an ore that breaks readily in mining, only hand spalling of the larger pieces is necessary. Ore is crushed extremely fine at the Goldbanks mine near Winnemucca, Nev. Here the ore is unusually hard and fine grained. The cinnabar is disseminated throughout the silica which cements the original breccia. Crushing to one-fourth inch was necessary in order to roast the ore within a reasonable time.

In general, particularly where a mechanical furnace is used, it is desirable to produce a minimum of fine material. For this reason a Blake type of crusher is to be preferred to a Dodge, and the ore should be handled as little as possible.

SCREENING.

Except by the use of grizzlies, quicksilver ore is seldom screened. At the Chisos mine in Texas the feed for the rotary furnace passes over an inclined screen with $\frac{1}{8}$-inch openings placed just below a grizzly.

At New Idria, Calif., a trommel is used for sizing part of the ore, the material passing a $2\frac{1}{2}$-inch grizzly being separated into the following sizes: Minus $\frac{1}{8}$ inch, $\frac{1}{8}$ inch to $\frac{5}{8}$ inch, $\frac{5}{8}$ inch to $\frac{7}{8}$ inch, and $\frac{7}{8}$ inch to $2\frac{1}{2}$ inches. These several grades of fine material were combined with other ore in making up the feed to the Scott furnace. The furnace was easily controlled by adjustment of the furnace feed which the choice of these several sizes of ore made possible. Cinnabar is so brittle that the finer sizes of ore are liable to be of somewhat higher grade. This is shown by the following assays of a sample of friable sandstone with cinnabar disseminated through it, which was crushed in the laboratory by a small jaw crusher set at about one-half inch, and was screened with a set of standard Tyler screens.

Variation of grade of ore with size.

Size of ore, meshes per inch.	Mercury, per cent.
2–3	2.26
6–10	2.79
48–65	3.24
100–150	4.36
150	4.92

These results show that screening is somewhat a concentrating process. With a hard, well broken ore, containing the cinnabar in cracks and fissures, the attrition from a revolving trommel might clean the coarser material so much that it might be rejected.

BRIQUETTING.

Before the development of the continuous fine-ore furnaces it was the practice to briquet the fine ore into rather large bricks known as "adobes," which were bonded with clay and made by hand in wooden molds. This practice has been described by Christy.[24]

At the present time, as far as is known, ore is briquetted in this country at only one plant—namely, that of the Chisos Quicksilver Mining Co. in Texas. The minus $\frac{1}{8}$-inch material which is removed from the feed to the rotary furnace is briquetted in a small machine of the tangential or Belgian type. A little water is added to the dust in a screw conveyer feeding a press. The material itself contains enough bonding material to form briquets, which can be roasted in the Scott furnace without undue dusting. At Almaden, in Spain, some of the fine ore is briquetted in hand molds, and then dried in the sun for treatment in Bustamente furnaces.

DRYING.

The drying of the quicksilver ore prior to furnace treatment is not ordinarily regarded as essential in furnace practice and has not received the consideration that it deserves. Usually drying is done because the physical characteristics of the wet ore require it, not because of effort to increase the efficiency of the furnace and the condenser.

[24] Christy, S. B., Quicksilver reduction at New Almaden: Trans. Am. Inst. Min. Eng., vol. 13, 1885, p. 552.

The various types of roasting furnaces differ as to the permissible moisture content of the ore. The rotary furnace is capable of handling a very wet and sticky ore, the limitation in this case being in the ore feeder rather than in the furnace itself. On the other hand, the Scott furnace requires a feed which will flow by gravity through the throat and over the tiles. A wet and sticky ore is inclined to hang up in the upper part of the furnace, thus upsetting the internal balance of the furnace, with the attendant dangers of overheating sections thereof, imperfectly roasting part of the ore, and reducing the furnace capacity. More serious than these, however, is the danger of mechanical injury to the internal furnace structure by the barring done to move the ore. For the Scott furnace, therefore, the drying of wet and sticky ore is essential.

EFFECT OF MOISTURE ON THERMAL EFFICIENCY OF FURNACE.

The preliminary drying of the ore also affects the thermal efficiency of the furnace and condenser, matters which have received scant consideration. A few simple calculations will bring out the points in question. If the ore enters the furnace at 20° C., and the hot gases leave the furnace at 200° C., the heat that will be removed from the furnace for each kilogram of water in the ore is as follows:

	Kg.-cal.
Heating water from 20° to 100° C	80
Latent heat of evaporation	537
Heating water vapor from 100 to 200° C	43
Total heat absorbed	660

Consider now the relation which this bears to the heat needed to roast dry ore. If the ore enters the furnace at 20° C., if the roasted ore leaves the furnace at a temperature of 500° C., and if the mean specific heat of the ore between 20° and 500° C. is 0.25, then the net quantity of heat required to roast 1 kilogram of ore is $(500-20) \times 0.25 = 120$ calories. The ore as drawn is cooler, but this heat is largely lost through conduction into the foundation.

For simplicity, the ore is assumed to be free from substances such as magnesium and calcium carbonates, the decomposition of which would absorb heat while they were passing through the furnace. According to the above calculation, the heat required by 1 kilogram of water in the ore is equal to that removed from the furnace by 5.5 kilograms of dry rock. Therefore, the presence of 9 per cent of moisture in the furnace feed will absorb as much heat as is required to roast an additional one-half ton of dry ore. If this ore containing 9 per cent moisture were sun dried to a moisture content of 4.5 per cent, theoretically 25 per cent less fuel would be needed, or 25 per cent more ore could be roasted. In practice, probably both of these advantages would be partly realized.

EFFECT OF WATER VAPOR ON THERMAL EFFICIENCY OF CONDENSER

The presence of water vapor in the gas leaving the furnace also adds to the work of the condenser system, as more heat has to be dissipated in cooling and condensing the water vapor. The following example shows how the moisture in the ore may increase the quantity of heat that the condenser must remove.

Let it be assumed that the fuel requirement for 1 metric ton is 8 gallons of fuel oil, weighing 7.9 pounds a gallon, corresponding to the consumption of 63.2 pounds or 28.7 kilograms of oil a ton. The weight of air theoretically required for the combustion of fuel oil is very close to 15 times the weight of the oil. If twice the theoretical quantity of air is used, the total air entering the furnace per ton of ore is $28.7 \times 30 = 861$ kilograms. With an ore essentially free from sulphur and carbonates, the weight of dry gases leaving the furnace for a ton of ore charged is approximately equal to the weight of the entering air. If it be assumed, further, that the gases leave the furnace at 200° C., and are discharged from the condenser system at 40° C., the heat which must be removed from the dry gases by the condenser for a ton of ore is $(200-40) \times 0.23 \times 861 = 31{,}680$ kg.-cal. (Specific heat of gas between 40 and 200° C.=0.23.)

Besides the water vapor that may come from the ore, this quantity of dry gas will have associated with it water vapor from other sources as follows:

	Kg.
Combustion of 28.7 kg. of oil	37
Steam for atomizing oil (0.6 kg. per kg. of oil)	17
Water vapor in air for combustion, one-third saturated at 20° C	4
Total	58

Of this vapor, about 42 kilograms are required to saturate the above quantity of dry gas coming from the condenser system at 40° C.; this leaves 16 kilograms of water vapor, which will remain in the condenser and free the following amount of heat:

	Kg.-cal.
Cooling water vapor, 200°–100° C. (specific heat, 0.43), per kg	43
Latent heat of liquefaction, at 100° C. per kg	537
Total heat released, per kg	580

If the cooling of the condenser water below 100° C. is disregarded, the heat given to the condenser system by 16 kilograms of water vapor is $580 \times 16 = 9{,}280$ kg.-cal.

54971°—25——3

The total heat imparted to the condenser system for a ton of each ore by the dry gas and the water vapor associated with it from the sources just discussed is then as follows:

	Kg.-cal.
Cooling 861 kg. dry gas, 200° to 40° C	31, 680
Cooling 42 kg. water vapor, 200° to 40° C	2, 890
Cooling and condensing 16 kg. water vapor	9, 280
Total	43, 850

Any moisture contained in the ore will be liquefied in the condenser system, as the water vapor from the combustion of the fuel and other sources is more than enough to saturate the gas discharged from the condenser. All of the heat associated with this water will, therefore, be given up to the condenser, even though some of the water may leave the condenser system in the form of mist. Again, if the quantity of heat involved in cooling the water below 100° C. is neglected, the heat imparted to the condenser system by a kilogram of water is 580 kg.-cal., as calculated above. If the ore were to carry 10 per cent of moisture—that is, 100 kilograms to a metric ton—the heat carried to the condenser by this would be $100 \times 580 = 58{,}000$ kg.-cal. It will be noted that this quantity of heat is greater than that given up by the dry gas and the water vapor from the combustion of the fuel. In other words, the presence of 10 per cent of moisture in the ore more than doubles the quantity of heat to be removed by the condenser. Condenser systems usually have large reserve capacities, as they are not scientifically designed. However, there is no economy in making a condenser system larger than needed in order to take care of a load which should not be put upon it.

TROUBLE FROM WATER IN CONDENSERS.

Apart from loss of thermal efficiency, the condensation of much water in the condensers is also objectionable, in that the water will take up sulphur dioxide from the gas, forming sulphurous and, later, sulphuric acid which is liable to destroy the material of the condenser. Also, mercury may be lost in the water flowing from the condenser system, although this can be prevented, as is discussed in detail on page 138. To sum up the advantages gained by drying the furnace feed, fuel is saved, furnace capacity is increased, and overloading the condenser system is avoided.

DRYING METHODS IN USE.

As previously stated, comparatively little attention has been given to drying quicksilver ores, and in many places nothing is done beyond storing the ore for a time in sheds or covered ore bins. Sometimes the ore is spread out on special platforms and sun dried.

At some Scott-furnace plants with brick condensers, the first section of the condenser was built with a sheet-iron top sloping at an angle of 45°, which served as an incline drier. The idea of using heat from the condenser for ore drying is good theoretically, but few massive brick condensers are now built, and drying in this way is done at only a few plants.

Incline driers have also been made to utilize the heat in exhaust steam and in the exhaust from oil engines. At the St. John's mine,

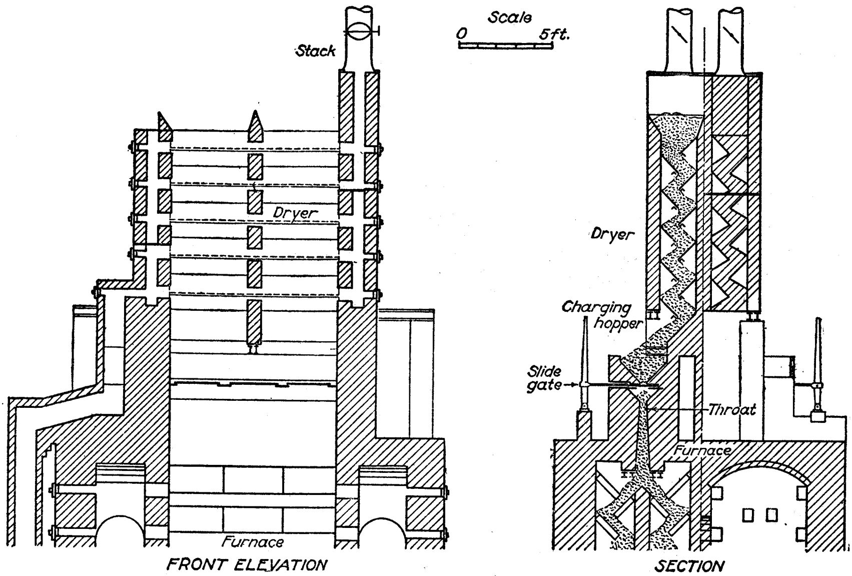

FIGURE 3.—Front elevation and cross section of ore drier on Scott furnace.

near Vallejo, Solano County, Calif., there is an incline drier built of sheet iron, with a system of brick flues beneath for heating with fuel oil. Rotary driers form part of the equipment at the Socrates mine, Sonoma County, Calif., also of the New Guadalupe mine, Santa Clara County, Calif.

A recent development in drier construction, designed by Murray Innes, of San Francisco, is that at the Oat Hill mine, near Middletown, Napa County, Calif. This drier, which is built on the top of the Scott furnace, is shown in Plate IV, *A* (see p. 6), and also in Figure 3. The hot air, which is brought up to the drier in two vertical brick flues, comes from the roasted ore just before it is discharged from the furnace. This air is heated in the base of the furnace, as is discussed on page 51.

As will be seen from Figure 3, the general principle on which the design of the drier is based is identical with that of the Scott furnace. The fire-clay tile of the Scott furnace are replaced by plates of sheet iron, which are supported by the brickwork and angle irons. The drier when visited was receiving about 30 cubic meters or 1,200 cubic feet of air a minute, at a temperature of 260° C. and a barometric pressure of 710 mm. The air entering the stacks had a temperature a little above 100° C. The drier was handling 36 metric tons of ore in 24 hours, and reducing the moisture from 8.5 to 0.3 per cent. The ore remained in the drier 2 to 2½ hours.

As will be seen from the sketch, the interior of the drier is readily accessible through the openings in the outer wall, and any tendency of the ore to hang up can be easily overcome. Unless native mercury is present in the ore, the escaping fumes can not injure the attendants while barring down hang-ups, as at the prevailing temperature the cinnabar has no measurable vapor pressure. The drier had been used continuously for over a year, and was regarded as successful in every way. One possible objection to the position of the drier on the top of the furnace is the excessive height of the combined furnace and drier, but as the Oat Hill plant is on a hillside, this height causes no inconvenience. One advantage of this particular arrangement is that the dried ore passes directly to the furnace, as shown on Plate V, *A*, but a drier of this type could be made an independent unit and heated with special fire boxes. (See p. 7.)

Calculation based on the performance of the Oat Hill drier indicates that if this drier were run separately from the furnace, about 16 kilograms (35 lbs.) of wood would be required to dry 1 ton of ore from 8.5 per cent of moisture to 0.5 per cent. The thermal efficiency of such a drier should be about 60 per cent.

DRYING PRACTICE AT MONTE AMIATA, ITALY.

In general, little attention to ore drying has been given in Europe. This is due partly to the character of the ore and to the natural drying that takes place incidental to the hand sorting, common at European plants. A notable exception to this is found in the Abbadia San Salvatore mine, Monte Amiata, in Tuscany, Italy. The ore is a mixture of about one part of limestone to two parts of clay and marl, and carries about 15 per cent of moisture as mined. The ore carries on the average 0.8 per cent of mercury in the form of cinnabar, the lowest economic mining limit being 0.03 per cent. High-grade ore carrying over 6 per cent of mercury is mined separately; it consists entirely of clay and marl with nuggets of cinnabar.

The clay makes the ore extremely sticky, and furnace treatment is impossible unless the moisture content is reduced below 7 per cent. A great deal of attention has been given to the drying problem at Monte Amiata, and a full description of the various efforts to solve this problem is given by Oschatz,[25] from whose account the material in this section is taken.

The drying and the sizing of the ore produce the following three grades of material:

Result of drying are at Monte Amiata, Italy.

Character of material.	Size.	Mercury content.	Type of furnace.
	Mm.	*Per cent.*	
Limestone	40–200	0. 3–0. 5	Shaft.
Limestone, ⅓; clay, ⅔	5– 40	0. 6–0. 9	Fine ore.
Clay and cinnabar nuggets	0– 5	1. 0–1. 4	Rotary.

Up to 1911 five different methods of ore drying had been used—namely, (1) sun-drying on open platforms; (2) covered drying floors; (3) heated drying floors; (4) Fantoni shaft drier; (5) drying tunnels. Since that year rotary drying kilns have been used extensively. The heated drying floors are still used for ore brought out from the mine on a level below the rotary drier. Some ore is also sun-dried in storage piles. Evidently much experience in ore drying has been had at Monte Amiata, probably more than at any other quicksilver plant. In view of the variety of methods which have been used, it seems worth while to trace the development of the drying practice in some detail.

SUN-DRYING PLATFORM.

The sun-drying platform built upon the south slope of a hill is shown in Figure 4. It was 12 meters (40 feet) wide and 20 meters (66 feet) long, and had a slope of 30° from the horizontal. The floor was planking covered by sheet iron 2 to 3 mm. (0.08 to 0.12 inch) thick. At intervals of 2 to 3 meters (6.5 to 10 feet) removable cross boards 20 cm. (8 inches) high were placed. A charging track and small receiving bin for the wet ore were provided at the top, and a covered bin for the dried ore at the bottom.

COVERED DRYING FLOORS.

The ore was spread out to dry on covered floors, in a layer 20 cm. (8 inches) deep. To reduce the moisture content from 15 to 7 per cent 24 to 48 hours were required.

HEATED DRYING FLOORS.

The drying floor had an area of 120 square meters (1,290 square feet), and was made of cast-iron plates 12 mm. (0.5 inch) thick.

[25] Oschatz, K., Die Verhüttung der Zinnobererze am Monte Amiata: Glückauf, Jahrg. 54, 1918, p. 113.

These plates were laid over a system of six flues, a fire box being provided for each two flues. The ore was spread in a layer about 30 cm. (12 inches) deep. The drier had a maximum capacity of 10 tons in 24 hours. The fuel consumption was high as compared with other types of driers to be described later.

FANTONI DRIER.

The general construction of the Fantoni drier resembles that of the ore chambers of the Cermak-Spirek furnace. The drier was 10 meters (33 feet) long, 2 meters (6.6 feet) wide, and 7 meters (23 feet) high over all; the drying chambers were 5 meters (16.5 feet) high.

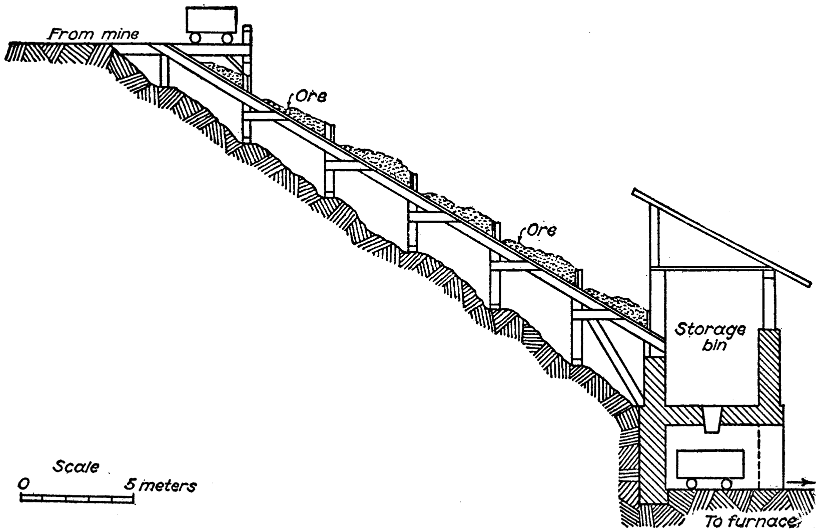

FIGURE 4.—Inclined ore-drying platform, formerly used at Monte Amiata, Italy.

The gables, which were supported by the side walls, were made of cast-iron plates 1.12 meters (3.6 feet) long, 0.6 meter (2 feet) wide, and 12 mm. (0.5 inch) thick. They were set at an angle of 60° from the horizontal; the distance between the lower edges of two neighboring gables was 20 cm. (8 inches). The drier was built in two sections, each with its own fire box. The flue system was so arranged in the walls that the hot gases passed back and forth under each successive tier of gables, and finally escaped by a stack placed on top of the drier in the middle. The ore was charged at the top through a screen having 10-cm. (4-inch) openings, and drawn out through iron doors at the bottom onto an iron plate, from which it was hoed into cars at the side of the drier.

The fuel used for this and other driers is described as "Reisig," no doubt being the smaller branches from trees felled for fuel, and brush from the genesta, a common European shrub.

The consumption was 60 kilograms (110 pounds) to 1 metric ton of ore. The moisture content of the ore was reduced from 15 to 3 per cent, corresponding roughly to an evaporation of 120 kilograms of water to a ton of ore. The ratio of the weight of water evaporated to the corresponding weight of fuel is therefore 2.4. The capacity of the drier for two shifts of 8 hours was 8 metric tons. The crew for each shift included a fireman, who also assisted at the draw, and two men who charged the ore and did what barring was needed to prevent a hang-up. It is stated that there was little trouble from hang-ups except on the uppermost row of gables, and that few repairs were required.

The capacity of this drier was rather small, only 12 metric tons for 24 hours. This, no doubt, was due partly to the fact that the drier was run to economize fuel, as is indicated by the statement that the escaping gases were cooled so much that only a moderate draft resulted. It will be noted that this drier ran on the counter-current principle, so that the heat was economically used.

COMPARISON WITH DRIER AT OAT HILL MINE

A brief comparison of this drier with that at the Oat Hill mine in California, described on page 25, is of interest, as they resemble each other. The Fantoni drier was approximately 33 feet long, 16.5 feet high, and 6.5 feet wide; the corresponding dimensions of the Oat Hill drier were 12 feet 9 inches by 12 feet by 6 feet. The volume of the ore chambers in the Fantoni drier was roughly 1,400 cubic feet as against about 300 cubic feet in the Oat Hill drier.

The Fantoni drier had a capacity of 12 metric tons in 24 hours and reduced the moisture approximately from 15 to 3 per cent; whereas the Oat Hill drier handled 36 metric tons in 24 hours, reducing the moisture from 8.5 to 0.5 per cent. Due allowance must, of course, be made for the difference in the amount of water to be evaporated per ton of ore, and also for the difference in the character of the ore. The Oat Hill ore is a friable sandstone which parts with its moisture readily, whereas the description of the Monte Amiata ore would indicate that the clay retains moisture tenaciously. A comparison of the two designs indicates that in the Oat Hill drier the average thickness of the layer of ore is not more than half of that in the Fantoni drier for the unit area exposed. The gases left the Oat Hill drier a little above 100° C., making a strong draft and correspondingly strong circulation of the air in the drier. This rapid circulation and higher temperature of the gases would, of course, enlarge its capacity, owing to the more rapid transfer of heat to the ore. The high temperature of the escaping gases involves a certain loss of recoverable heat. On the other hand, the much smaller superficial area of the Oat Hill drier per ton capacity makes for a lower radiation and convexion loss, and correspondingly better thermal efficiency.

TUNNEL DRIER.

The tunnel drier at Monte Amiata consisted of four single-track tunnels 60 meters (197 feet) long, arranged in two sections. Each tunnel held 40 cars, and a track for the return of the empty cars was placed between the two sections of driers. Each car carried 8 zinc-coated trays with perforated bottoms, the openings being 25 mm. (1 inch) in diameter. The trays were of different sizes, the smallest being placed at the bottom, so that the car carried an inverted pyramid of trays. The ore was spread on trays to the depth of 10 to 15 cm. (4 to 6 inches).

Each tunnel had two fire boxes, one under the discharge end of the drier, the other at the middle point of the tunnel. The hot gases entered the tunnel through openings between the rails. A 14-horse-power exhauster at the charge end of the tunnel withdrew the moist gases from all four tunnels.

The cars were loaded on a storage floor having a capacity of 500 tons, which was at the charge end of the tunnels. The large pieces of limestone were separated and the clay lumps broken up. A crew of 1 foreman and 22 men ran the drier, 16 men loaded the cars, 3 men ran a mechanical dumping device at the discharge end, 2 men attended to the fires, and 1 man with a mule returned the empty cars to the charging floor.

This drier had a capacity of 120 tons in two 8-hour shifts. For drying the ore from 15 to 3 per cent moisture, the fuel consumption was 90 kg. of brush per metric ton of ore, giving a water-fuel ratio of 1.3. The maintenance cost of this drier is reported to be high, mainly because of the repairs to trays and cars.

ROTARY DRIERS.

During 1911 to 1913, three rotary-drier kilns were erected at Monte Amiata. These have been found so satisfactory that the use of all other types of driers has been abandoned, except as previously noted. Driers of this type appear to be much used in Europe, but so far are seldom seen in the United States. This type of drier, known as the Möller and Pfeifer drying drum, has been described in detail by G. Franke.[26]

[26] Franke, G., Handbook of briquetting (translated by F. C. A. H. Lautsberry). Philadelphia, 1918, vol. 2, p. 98.

DESCRIPTION

The general design of the drier shown in Figure 5, and also in Plate V, *B*, is taken from the article of R. Sterner-Rainer.[27] The revolving kiln was 10 meters (33 feet) long and 1.6 meters (5.25 feet) in diameter, and was inclined at 1 to 13. The kiln made 2 revolutions a minute, and required about 12 horsepower, probably including the power required for the screw conveyer in the dust

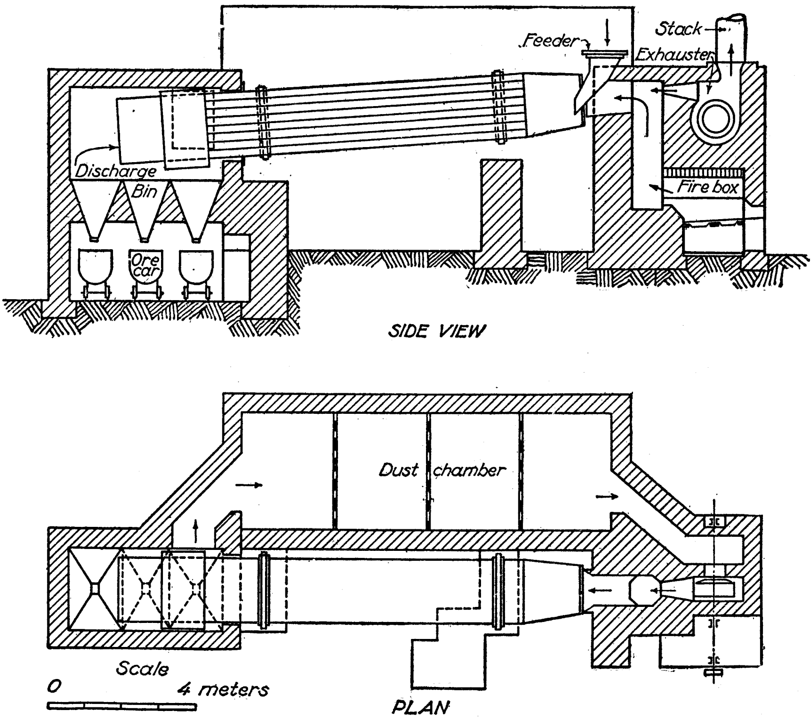

FIGURE 5.—Plan, side view, and sections of cellular rotary ore drier.

chamber and also for the exhauster. The interior of the kiln was filled by a cellular construction, shown in cross section in Plate V, *B*. The feeding device in the conical head is not described in detail, but apparently was spiral flanges that elevated the ore to the different openings. The construction is said to be surprisingly simple, and the distribution of the ore very uniform. On the charging hopper just above the feeder was a screen with 8 by 12 cm. (3 by 5 inch) openings. A classifying screen which separated the drier discharges into three sizes was attached to the lower end of the shell.

[27] Sterner-Rainer, R., Der derzeitige Stand des Quecksilber-hüttenwesens in Europa: Oest. Ztschr. Berg. Hut., Jahrg. 62, 1914, pp. 53–535. Translation by Schuette, C. N., Production of quicksilver in Europe: Chem. and Met. Eng., vol. 19, 1918, pp. 721–727, 770–775; and vol. 20, 1919, pp. 32–35, 82–84.

The kiln was heated directly by the hot gases from a fire box, the gases passing through the kiln in the same direction as the ore. Because dust was generated by the screening device at the end of the kiln, the escaping gases were passed to a dust chamber at the side of the kiln. This dust chamber contained three transverse screens, and was fitted with a hopper bottom and screw conveyor for the removal of the dust. Circulation through this dust chamber was induced by an exhauster at the feed end of the kiln. The discharge from the chamber was divided into two parts, the major part going to the stack and the other entering the hot gas flue from the fire box. The return gas apparently acted as an injector on the hot gases coming from the fire box, and also reduced the temperature of the gas stream entering the drier. The increase in the velocity of the gas passing through the drier owing to the return of part of the gas is, no doubt, one of the reasons for the large capacity shown by these driers. A temperature of 80° C. in the connection from the discharge bin to the dust chamber indicated the best running conditions.

In two 8-hour shifts, each drier reduced the moisture content of 100 metric tons from 15 to 5 per cent, corresponding to an evaporation of 100 kilograms (220 pounds) of water to a metric ton of ore, or an evaporation of 625 kilograms (1,355 pounds) of water an hour. The fuel consumption was 30 kilograms (66 pounds) of brush for a metric ton of ore, giving a ratio of water evaporated to fuel used of 3.3. The ore was in the kiln about 30 minutes.

TREATMENT OF DUST.

The amount of dust collected in the dust chambers in a week was about 3 tons, or about 0.5 per cent of the weight of the ore treated. The quantity of cinnabar in this dust varied from 0.4 per cent at the inlet end of the dust chamber to 0.2 per cent at the exit end, averaging about 0.3 per cent. This dust was treated on two concentrating tables, with a recovery of about 75 per cent of the cinnabar. To test whether gases leaving the exhauster carried away an appreciable amount of cinnabar dust these gases were sprayed with water in another chamber. The dust removed by the water spray was collected in a separate box, but contained too little cinnabar to justify treatment.

OPERATION OF THE DRIER.

Each drier had a crew of a foreman and five men; three men fed the ore and broke the large lumps on the charge screen, one man tended the discharge end of the kiln, and one was fireman.

The kilns are reported to have run satisfactorily in every way and to have saved much of the cost of drying. Expected difficulties, such as chemical deterioration of the metal work of the kilns and choking of the passageways, did not materialize. The passage remained clear largely because the hot gases entered the kiln along with moist ore. Exceptionally wet ore showed no tendency to adhere to the walls of the feeder or drier flues, probably because the metal parts were always a little hotter than the ore, so that any clay crusts which formed momentarily soon dried and contracted enough to scale off readily. Another advantage of the parallel-flow principle is the equal temperature along the length of the drier; the hot gas comes in contact with the cool damp ore and heats it as it travels through the drier, with a corresponding fall in the temperature of the gases. This prevents excessive heating of the ore, with attendant loss of mercury. No volatilization of mercury was observed in the drier.

GENERAL DISCUSSION OF DRYING METHODS.

The utilization of waste heat to dry quicksilver ores is an obvious economy wherever practicable. When this can not be done, there are, no doubt, many circumstances under which it will pay to dry the ore by the use of fuel. The impression prevails among practical quicksilver men that the use of fuel for ore drying increases fuel consumption. When all factors are considered, however, this is not necessarily so; at least the additional fuel consumption is much less than is supposed at first glance. The evaporation of a given amount of water involves the same expenditure of heat, irrespective of whether it is done in the upper part of a furnace or in a specially constructed drier. When a furnace is supplied with dry ore, it should show a larger capacity than when running with moist ore, provided the fuel consumption remains constant, or fuel consumption to a ton of ore treated should be less if the furnace tonnage remains constant. In other words, much of the heat that is used in the drier should be saved in one way or another when the furnace is running on dry ore. Furthermore, with uniformly dried ore it should be possible to run the furnace continuously at maximum capacity, a condition that makes for best efficiency in the long run.

The choice of drying equipment for a given plant will be determined largely by the same consideration that governs the choice of the furnace. In the United States, nonmechanical furnaces, such as the Scott furnace, are usually preferred for mines of moderate output, particularly where no convenient source of power is available. This type of furnace has the advantage of requiring little provision for mechanical attention, as it has no moving part and requires no power. For a Scott-furnace plant, the type of drier constructed at the Oat Hill mine seems particularly suitable. This drier can be built on the top of the furnace or as a separate unit heated by special fire boxes.

For a plant with a mechanical furnace a rotary drier is no doubt the most suitable, as under these conditions the objections mentioned above do not apply. The experience with rotary driers at Monte Amiata would indicate that these can be run at low cost for mechanical upkeep and have large capacity.

In the design and operation of drying equipment, these precautions must be observed: When the ore contains no free quicksilver, a maximum temperature of 200° or even 250° C. is permissible in the drier, as this is below the point at which cinnabar appreciably vaporizes. When free quicksilver is present, however, the drying becomes a more delicate operation, and the maximum safe temperature is in the vicinity of 100° C. With this limitation of temperature the drier would have to be much larger for a given capacity than when the higher temperature is allowable.

SAMPLING AND ASSAYING QUICKSILVER ORES.

INTRODUCTION.

Sampling and assaying must be considered together, as errors in both effect the calculation of the mercury content of a given quantity of ore or other material. Accurate assays are of little value unless the samples represent with corresponding exactness the mass of material under consideration. Generally speaking, present methods of assaying in the quicksilver industry are much more accurate than those of sampling. The question of sampling should receive more attention from quicksilver operators.

MINE SAMPLING.

The accurate sampling of ore in place is probably more difficult for quicksilver than for the majority of other metals. When quicksilver occurs native, the reason is obvious. The friability of cinnabar, and its tendency to occur in fissures and cracks of the country rock, make particularly difficult the securing of a true sample from a working face. Particles of cinnabar always tend to break off from the point where the cut is being made, and thus increase the mercury content of the sample. On the other hand, in the sample itself, much of the mineral is liable to be in the fine material; so care must be taken to see that none of this is lost. The sampling of ore in place should only be intrusted to some one who is familiar both with the peculiar characteristics of quicksilver ores in general and of the particular formation sampled.

Little development work in advance of actual mining is usually performed in quicksilver mines. For this reason, also because of the extremely irregular occurrence of the mineral in most places, the working face must be sampled after each blast so that ore can be properly selected with reference to the average grade that is to be delivered to the reduction works. This estimation of the grade of the ore can sometimes be made by inspection; frequently the ore is panned with a 6-inch pan for this purpose. Miners, after they become familiar with a particular deposit, show a remarkable expertness in estimating the mercury content of an ore in this way. Oschatz [28] says that a miner at Monte Amiata by simple panning consistently checked assay results within 0.1 per cent mercury. The subject of sampling and estimating quicksilver ore has been discussed at some length by one of the writers.[29]

PLANT SAMPLING.

The regulation of metallurgical treatment in the quicksilver industry by systematic sampling and assaying has not, in general, been nearly so well developed as in other branches of metallurgy. The relatively small scale of most quicksilver plants and certain inherent difficulties in accurate sampling are the main reasons for this neglect. Also the simplicity of quicksilver metallurgy makes elaborate sampling less necessary than for other ores. In the ordinary course of ore treatment there are no intermediate products to consider, and, in general, little retreatment of material, consequently relatirely few samples need to be taken. On the other hand, there can be no question as to the desirability of exact knowledge in controlling quicksilver reduction. The possession of data from which a metallurgical balance could be prepared would often save the quicksilver operator much guesswork and improve efficiency of operation.

GENERAL THEORY.

The general theory of sampling has been developed by Brunton.[30] A review of its application to the sampling of quicksilver ores and products at the reduction works will bring out one of the inherent difficulties referred to above. The accuracy with which the metallic content of any material can be determined rests ultimately upon the accuracy with which the final sample is weighed and analyzed. In order that the final result shall have the same probable accuracy as

[28] Oschatz, K., Die Verhüttung der Zinnobererze am Monte Amiata: Glückauf, Jahrg. 54, 1918, p. 609.

[29] Schuette, C. N., The sampling and estimation of quicksilver ore: Min. and Sci. Press, vol. 122, 1921, p. 293.

[30] Brunton, D. W., The theory and practice of ore sampling: Trans. Am. Inst. Min. Eng., vol. 25, 1895, p. 826.

that of the assay of the ultimate sample, the amount of the original sample taken must have a definite relation to the quantity which is weighed out for the final assay. This condition will be fulfilled if the same probability of error through the rejection or inclusion of high-grade particles is maintained throughout the entire sampling and subsequent reduction of the original sample to a size and fineness suitable for the final assay. According to Brunton, this is done when each successive sample from that originally cut from the ore down to and including that weighed out for the assay contains the same number of particles. Assuming then that a 1-gram sample is to be used for the final determination and that the fineness of this sample is such that it will pass the ordinary 100-mesh screen, it is possible to calculate the corresponding quantities of samples composed of particles of larger dimensions. The results of such a calculation are given in Table 3:

TABLE 3.—*Size-weight ratio for sampling quicksilver ores.*

Mesh.		Opening.	Wire diameter.	Contents of cube.	Weight of cube.	Weight of sample.
Per centimeter.	Per inch.					
		Millimeters.	*Millimeters.*	*Cu. centimeters.*	*Grams.*[a]	*Grams.*
39	100	0.14	0.116	$0.0_{5}2744$	$0.0_{4}225$	1.0
23	60	.25	.185	$.0_{4}1563$	$.0_{3}128$	5.7
12	30	.50	.33	$.0_{3}125$	.00103	45.7
8	20	.85	.40	$.0_{3}614$	.00504	224.0
3.9	10	2.00	.56	.008	.0656	2,915.0
						Kilograms.
2	5	4.00	1.00	.064	.525	23.33
1	2.5	8.00	2.00	.512	4.198	186.58
-------	---------	25.00 (1-inch)	---------	15.625	128.13	5,694
-------	---------	50.00 (2-inch)	---------	125.0	1,025.0	45,555
-------	---------	75.00 (3-inch)	---------	421.875	3,459.4	153,751

[a] Specific gravity of HgS=8.2. Weight of 1 c. c. HgS=8.2 grams.

Obviously, the strict application of this principle would often lead to the conclusion that the entire quantity of ore sent to the reduction works should be taken as the sample. This, in fact, is precisely what has been done in the past by certain quicksilver operators who assumed that the plant recovery equaled 100 per cent, and calculated the tenor of the ore from the amount of quicksilver produced.

Theoretically correct sampling clearly is impracticable in most quicksilver plants, but the more nearly it can be approached the more dependable the result will be. With low-grade ore, and particularly if the cinnabar is fairly well disseminated, the principle of sampling cited indicates that a representative sample should, in general, be 5 per cent of the material. The chances are small that a lump of high-grade material large enough to vitiate the sample will be included. If this happens it usually can be detected by comparing a set of assay results; the rejection of an occasional assay of unusually high metal content is justified. Mechanical rather than hand sampling is preferable, but the latter is almost universal today. The method of sampling and weighing ore at the Chisos mine, Brewster County, Tex., is shown in Plate VII, *A* (p. 52), and (except for mechanical sampling) it is perhaps as satisfactory and convenient a way as any. A shovelful of ore is taken from each 1-ton car when the car is balanced on scales which are set to a predetermined weight.

IMPORTANCE OF TRUE SAMPLE.

As has been shown, cinnabar, owing to its brittleness, tends to concentrate in the fines. It is, therefore, particularly important in hand sampling that the proportion of coarse and fine ore in the sample be the same as in the original lot. A striking example of an error from this source was observed where a sample was taken from each aerial tram bucket just after it had been filled. The ore had been reduced with a jaw crusher set at about 2 inches, and stored in a large bin from which it was drawn into the buckets. When each bucket was nearly filled, and the operator began to close the gate of the ore chute, the last material that flowed into the bucket was made up of relatively fine particles that passed the partly closed gate. This fine material accumulated on the side of the bucket next to the ore chute. The operator trimmed the bucket by scraping this pile of fine material to the center of the bucket, and immediately thereafter took the sample with a small scoop. Tests showed that the sample so taken contained an undue proportion of fine material, and also distinctly more mercury than the average tenor of the ore. Until this fact was recognized a large amount of mercury was supposed to be lost in some unknown way in the roasting.

The principle underlying mechanical sampling is that the whole ore stream should be intersected at regular intervals and at a uniform speed; only by strict adherence to this principle can truly representative samples be obtained. When a homemade mechanical sampler is used special attention should be given to the way in which the cutting device intersects the ore stream. The following details are essential: The cut-out device, whether a scoop, small chute, or whatever form it may have, should cross the ore stream in a fixed plane; the dividing edges of the scoop should not favor the fall of fragments in either direction; the velocity of the scoop should be uniform and the interval between cuts constant. If the scoop travels in a straight line, its edges should be parallel; and, similarly, if it moves in an arc the sides should be radial.

When a mechanical sampling device can not be conveniently installed the following simple procedure can be adopted where the ore is drawn from a bin into a furnace-charge car: A box having sides parallel to the charge car is suspended near the center of the car by supports resting on the sides. One end of the charge car is then brought under the ore chute, and when the ore is flowing freely the car is moved slowly forward at a uniform rate, so that during its passage under the ore stream the box will be evenly filled. If the box is allowed to overflow, an error will be introduced through the segregation of the finer and presumably richer ore. As mentioned above, about 5 per cent of the carload is a suitable quantity to take in this way.

The crushing and quartering of the sample should be carried out according to the figures given in Table 5. It is usually impracticable to keep the large samples in air-tight containers, therefore a small sample for the determination of moisture should be taken separately and placed at once in a tight receptacle.

When an ore in which the cinnabar shows freely is sampled by hand the sampler is liable to select high-grade pieces. With such ore it is therefore particularly important to adopt some method that will make the sampling impersonal as far as possible.

ACCURATE DETERMINATION OF THE WEIGHT OF CHARGE.

Accurate car sampling and assaying will not give reliable information regarding plant operation unless the weight of the ore charges is determined with equal accuracy. The error that can be caused by unreliable sampling and determination of the weight of material charged to the furnace is shown by the following observed example: The furnace charge was made up of four different materials, the bulk density of each of which was accurately determined from time to time. The quantity of each charge to the furnace was calculated from the number of cars charged; that is, from the volume. The charging was, in general, fairly regular, and the cars were uniformly filled. Samples were taken by hand, and the assaying was dependable. On the basis of the assay returns and calculated tonnage, the furnace input for a certain month corresponded to 770 flasks of mercury. The actual production during this same period was 642 flasks. During this period the condenser system was under close observation, and the losses therefrom were accurately determined. Moreover, the system was so constructed that very little, if any, mercury was absorbed by the condenser material. The ore was thoroughly roasted, and no gas leaked from the furnace. From the observation of furnace and condenser operations it was calculated that at least 95 per cent of the furnace input was delivered to the quicksilver room. The difference between this and the calculated input, amounting roughly to 12 per cent, constituted the error which was due to the method of sampling and calculating tonnage. The workmen were liable to record more carloads of ore than were actually delivered to the furnace.

A good method for controlling the furnace tonnage, observed at one of the Texas plants, is described in detail on page 60. At a number of plants where the ore is fairly accurately weighed and sampled the indicated extraction is close to 100 per cent. Not infrequently, even when tonnages are accurately determined, the calculated extraction is found to be over 100 per cent. At certain plants observed this is undoubtedly due to the tendency of the cinnabar to concentrate in the fines. At the plants mentioned the furnace-charge car was filled from a bin and then run some distance to the track scales before the sample was taken. The jarring movement of the car caused most of the fines to settle to the bottom, so that the sample which was taken from the top of the car did not contain the proper proportion of fines. Christy [31] shows that when weighing, sampling, and assaying are carefully done, consistent and reasonable results can be obtained.

SAMPLING OF TAILING.

Sampling furnace tailing requires the same precautions as sampling the ore. The character of the ore and the nature of the furnace operation influences quicksilver losses to occur whether in the fine or coarse material. When an ore containing cinnabar along fissures and veinlets in the country rock is treated in the Scott furnace the losses will usually be in the fines, owing to the tendency of the cinnabar to sift down through the furnace charge. Metallic mercury also may be occasionally in the fines. On the other hand, when the cinnabar is uniformly disseminated throughout lumps of ore, losses are more liable through incomplete roasting of the larger lumps, and examination of only the fines would give no evidence of loss.

When the weight of the ore is greatly reduced in roasting as the result of loss of moisture, carbon dioxide, and possibly sulphur, this shrinkage should be accurately known in order to interpret clearly the results of sampling and assaying the furnace tailing.

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[31] Christy, S. B., Quicksilver condensation at New Almaden: Trans. Am. Inst. Min. Eng., vol. 14, 1885–6, p. 229.

ASSAYING OF QUICKSILVER ORES AND PRODUCTS.

Assaying quicksilver ore has been treated at some length in a recent Bureau of Mines publication.[32] Two general methods which have had considerable application in quicksilver assaying are the Eschka[33] method and the glass-tube method.

Two forms of apparatus have been developed in the United States for carrying out the Eschka method, namely, the Whitton[34] and the James[35] apparatus. This method consists in general of heating the ore sample in a crucible, which is covered with a piece of gold or silver foil on which the mercury collects and is subsequently weighed. This method is reasonably accurate and is applicable to ores and other mercurial products which contain no more than a trace of organic matter, or a small percentage of sulphur.

The glass-tube method is based on distillation of the ore in a glass tube sealed at one end, and subsequent titration of the distilled mercury. This method was thoroughly studied by the Bureau of Mines and found to be rapid, accurate, and adaptable to a variety of mercurial material. For details, Technical Paper 227, cited,[32] should be consulted. In general, the procedure is so dependable that the total error need be but slightly in excess even with very low-grade material. The results for ore and other samples containing 0.5 to 5 per cent of mercury should be dependable to about 0.03 per cent.

COARSE-ORE FURNACES FOR ROASTING QUICKSILVER ORES.

FURNACES IN THE UNITED STATES.

The use of coarse-ore furnaces has steadily diminished in the United States during recent years; at present only one plant uses them and that occasionally. As their use in the past was common, a number are standing idle at many plants. These furnaces are particularly adapted to moderately high-grade lump ore. The simple shaft furnace using charcoal or coke mixed with the ore has better fuel efficiency that any other type.

The practical abandonment of coarse-ore furnaces in this country is due to several causes, among which is the lack of high-grade lump ore and the high price of charcoal or coke for use in the internally fired shaft furnace. The chief reason, however, is because

[32] Bouton, C. M., and Duschak, L. H., The determination of mercury: Tech. Paper 227, Bureau of Mines, 1920, 44 pp.

[33] Eschka, A., Beschreibung des Verfahrens zur Bestimmung des Quecksilbers in seinen Erzen: Oest. Ztschr. Berg.-Hüt., Jahrg. 26, 1872, p. 67.

[34] Whitton, W. W., The determination of mercury in ores: California Jour. Tech., vol. 4, Sept. 1904, p. 33.

[35] James, G. A., The James apparatus for quicksilver determination: Eng. and Min. Jour., vol. 90, 1910, p. 800.

crushing the oversized ore and sending it to one of the fine ore furnaces is more economical than running a separate coarse-ore furnace. Only two types of domestic coarse-ore furnace deserve discussion in any detail—namely, the New Idria furnace and the Neate furnace.

THE NEW IDRIA COARSE-ORE FURNACE

This type of furnace was constructed about 1900, and was designed by B. M. Newcomb, for many years general superintendent of the New Idria Quicksilver Mining Co., Idria, San Benito County, Calif. Two of these furnaces were working at this mine up to 1918, after which all ore was treated in rotary furnaces. These coarse-ore furnaces were about 24 feet long, 16 feet wide, and 40 feet high. The general design is shown in Figure 6. They were made largely of common red brick, fire brick being used for lining the fire boxes and the shaft. Cast-iron flues set in the shaft wall led the vapors from the ore shaft to the dust chamber.

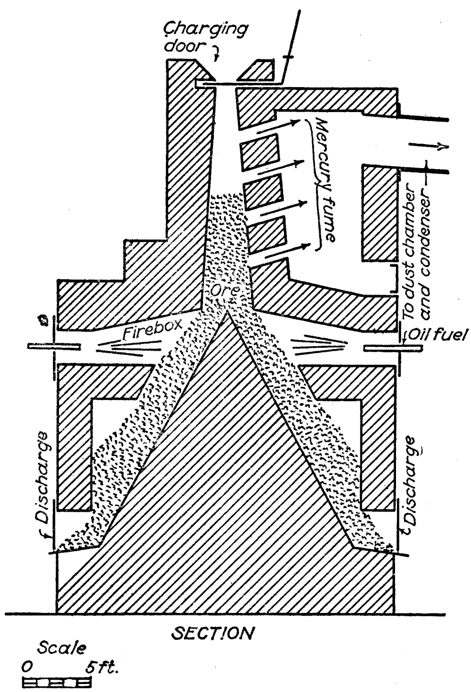

FIGURE 6.—Section of the New Idria coarse-ore furnace.

The two furnaces at New Idria were known as No. 2 and No. 3, respectively. No. 2 had a capacity of about 58 tons (52.7 metric tons) of 1.0 to 2.5 inch (22 to 64 mm.) ore for 24 hours; No. 3 handled about 62 tons (56.4 metric tons) of 2.5 to 10 inch (64 to 254 mm.) ore in 24 hours. The ore was mainly a soft sandstone, low in moisture and free from clay or other substances which would absorb heat during roasting in the furnaces. This ore weighed about 80 and 74½ pounds a cubic foot in No. 2 and No. 3 furnace, respectively.

In recent years these furnaces were fired with oil and used a little more than 6 gallons per ton of ore treated, corresponding to about 2.4 per cent of the weight of the ore. Forstner[36] gave the

[36] Forstner, William, Quicksilver resources of California: California State Min. Bur. Bull. 27, 1903, p. 214.

capacity of these furnaces as 90 to 100 tons of ore, with a fuel consumption of 2.5 cords of wood in 24 hours. At that time the character of the ore was somewhat different as it contained much pyrite and weighed about 120 pounds a cubic foot. This accounts both for the larger tonnage and the lower fuel consumption.

It is interesting to note that the same charge cars have always been used with these furnaces, and that the volume of ore charged (about 1,450 to 1,664 cubic feet in 24 hours with medium to coarse ore, respectively) has remained practically constant during the entire life of the furnaces. This brings out the point that the capacity of a furnace can be stated better in volume than in weight of ore.

A tabulation of the average of daily temperature observations at different points in the furnaces during their last period of work is given below.

Temperature reading in coarse-ore furnaces.

	Temperature.	
	No. 2.	No. 3.
	° C.	° C.
Fire boxes, near ore column	905	815
Shaft, just above fire boxes	606	561
Shaft, half way up	156	347
Shaft, top	86	96
Dust chamber, top	142	193
Exit pipes	164	216

It will be noted that the furnace handling the coarser ore showed a somewhat lower temperature in the fire boxes and a higher temperature in the shaft. The fact that the temperature at the top of the dust chamber is in each furnace lower than the average temperature of the escaping gases is of interest, as this shows the tendency of the hot gases to pass out through the slots into the dust chambers rather than to travel up through the ore in the shaft. Very little oxygen was found in several samples of gas taken from the exit pipes, thus showing that the furnaces ran with comparatively little excess air.

These furnaces worked satisfactorily. The furnace receiving the finer feed gave trouble occasionally by choking in the shaft, which caused a loss of mercury vapor from the fire boxes. A good draft must be maintained through the condenser system connected to this furnace; also, the ore column must be kept from 6 to 8 feet below the charging door. Carelessness of the firemen occasionally caused clinkers in the fire boxes. These furnaces gave some trouble from the escape of mercury-laden gases at the time of charging;

some leakage could be always detected. This could, no doubt, have been overcome by a suitable design of charging apparatus.

The crew for these two furnaces for an 8-hour shift was composed of one charge man for each furnace, one fireman for the two furnaces, and three men engaged in drawing and tramming the roasted ore, six men in all. The ore was charged and drawn at half-hour intervals.

THE NEATE FURNACE.

The Neate furnace is a simple shaft furnace of circular cross section, designed for solid fuel mixed with the ore. The shaft is about 21 feet deep and 2 feet in diameter at the top and bottom, widening to a little over 4 feet at about a third of the distance from the bottom. The shape is something like an iron blast-furnace. The ore is supported in the shaft by an iron fork, which is drawn out part way when roasted ore is to be removed from the furnace.

Two furnaces at the St. John's quicksilver mine, Solano County, Calif., had a capacity of 8 tons of ore each in 24 hours, with a consumption of 2.5 per cent by weight of coke. One man a shift could run the two furnaces.

THE SPIREK COARSE-ORE FURNACE.

The only coarse-ore furnace now used to any extent in Europe is commonly known as the Spirek shaft furnace. Detailed drawings of one of the earlier designs of this furnace are given in "Mineral Industry."[37]

The details of construction have been variously modified, but the shaft furnace at Monte Amiata, as described by Oschatz,[38] gives a good idea of the construction and operation of this type of furnace.

In 1914 the shaft-furnace equipment at Abbadia San Salvatore consisted of one furnace with two shafts and three furnaces with four shafts. A four-shaft furnace which was built in 1914 is shown in Plate VII, *B* (p. 52).

The furnace shafts were 1.2 meters (3.94 feet) square in cross section and about 7 meters (23 feet) high. The shaft walls were built of blocks of selected trachyte, 0.6 meter (1.97 feet) thick and supported by columns of the same material 0.6 meter (1.97 feet) square and 1.2 meters (3.94 feet) high. The cars to receive the roasted ore passed between these columns and under the shaft. The shafts were open at the bottom, the ore being supported by 30 by 30 mm. (1.2 by 1.2 inch) grate bars set at an angle of 20°,

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[37] Struthers, Joseph, Mineral Industry, vol. 10, 1901, pp. 560–561.

[38] Oschatz, K., Die Verhüttung der Zinnobererze am Monte Amiata: Glückauf, Jahrg. 54, 1918, p. 548.

sloping toward the condenser end. The ore was drawn by displacing these grate bars with a long slice bar working from the upper end of the grate. The Spirek charging device was used. This is also shown and described in detail in "Mineral Industry."[39]

IMPROVEMENTS.

The condenser connection is one of the important differences between this new battery of furnaces and the old. With the latter, a heavy cast-iron pipe was used to connect the top of the furnace with the condenser. In the new furnaces the gases were led down through brick flues built against the furance wall. These flues, which were provided with clean-out doors, were connected directly with the tile pipes of the condenser. Another variation in the construction of the new furnaces was the omission of the sheet-iron covering and the iron plates in the supporting columns. When the old furnaces were torn down after running 10 years no mercury was found in the walls.

The capacity of each shaft was 7.6 metric tons in 24 hours. Every 2 hours 630 kilograms of ore and a wheelbarrow load of charcoal was charged. A corresponding amount of roasted ore was withdrawn at the time of charging. The charge contained 23 kilograms of charcoal for a metric ton of ore, corresponding to a fuel consumption of 2.3 per cent per weight of the ore treated. It was concluded that an increase in the furnace capacity would result in a higher temperature in both the outgoing gases and the roasted ore, involving, therefore, an increased fuel consumption a ton.

The outgoing gases had a temperature of about 120° C., and the roasted ore was cooled to 80 ° to 100 ° C. before being drawn. This indicates a very good recovery of the heat in the roasted ore.

HEAT BALANCE.

Oschatz[40] discusses the heat balance for these furnaces at some length. His reasoning is difficult to follow, as he has to assume that nearly half of the total heat generated is used up in chemical reactions in the furnace, in spite of the fact that the limestone passes through undecomposed. His observations with the old furnaces, which were handling ore ranging from 40 to 200 mm. (1.6 to 8 inches) in size, led him to the conclusion that these furnaces ran with 400 per cent excess air.

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[39] Struthers, Joseph. Work cited, pp. 560–561.
[40] Oschatz, K. Work cited, p. 548.

He also estimated that a loss of fuel amounting to 10 per cent occurred through incomplete combustion, with the formation of carbon monoxide. The above conclusion in regard to the large quantity of excess air passing through the furnaces led to a change in the size of the feed, the minimum being reduced to 20 mm. (0.8 inch). It was estimated that the inclusion of this finer material in the furnace feed would reduce the excess air to 200 per cent, which was considered a suitable amount for good operation. The construction of the new battery of shaft furnaces at Monte Amiata in 1914 was the result of concluding that ore down to 20 mm. (0.8 inch) could be successfully handled in the shaft furnaces.

OPERATING COSTS.

Ten shafts were run with five men for each of the three shifts; two men charged the raw ore, two drew the roasted ore, and one trammed with a mule. Oschatz gives the cost of ore treatment for a metric ton with the coarse-ore furnaces as follows:

Cost of roasting in coarse-ore furnaces in Italy.

Mining	L9.00	$1.74
Drying and sorting, shaft furnace	2.00	.39
Fuel	2.60	.50
Labor	.70	.14
	14.30	2.77
Overhead	6.00	1.16
Total	20.30	3.93

(A lira, L,=$0.193.)

CONCLUSIONS.

The work of these shaft furnaces was regarded as highly satisfactory. No evidence of loss of mercury through imperfect roasting of the ore was ever observed. An unsuccessful attempt was made to use coke as fuel. However the failure is ascribed to the fact that the coke was made from local coal that yielded an inferior product. In view of the successful use of coke in the Neate furnaces at the St. John's mine in California, there is no apparent reason why a good grade of coke should not work in a satisfactory manner in the Spirek shaft furnace.

SUMMARY OF COARSE-ORE FURNACES.

For the purpose of comparison, the most important data in regard to the coarse-ore furnaces just described are given in Table 4.

TABLE 4.—*Comparison of coarse-ore furnaces.*

Name of furnace.	Name of mine.	Country or State.	Number of furnaces on which data are based.	Source of information.	Tonnage capacity per day, metric tons	Fuel	Fuel consumed, per cent by weight of ore.	Men per furnace per shift.	Metric tons per man per day.
New Idria	New Idria	California	2	([a])	54.5	Oil	2.4	3	6.05
Neate	St. John's	do	2	([b])	7.3	Coke	2.5	½	4.86
Spirek	Idria	Austria	10	Castek	14.6	Charcoal	2.1		
Do	do	do	11	Sterner-Rainer.	15	Wood or coal.	2.5	1	5.00
Do	Monte Amiata	Italy	10	Oschatz	7.6	Charcoal	2.3	½	5.06
Do	Almaden	Spain	2	Sterner-Rainer.	14	Coke	2.3	2½	2
Bustamente	do	do	11	do	19	Coal	9.5	[c] 2	[c] 3

[a] Data collected by C. N. Schuette.
[b] Private communication to the authors by C. G. Dennis, general superintendent St. John's Quicksilver Mines Co.
[c] Intermittent furnace, 2 men a shift needed for firing for two days, then 11 men discharge and refill the furnace, and clean up the recovered quicksilver.

PRACTICE AT IDRIA AND MONTE AMIATA.

The data in regard to the furnaces at Idria are taken from an article by Castek,[41] and also one by Sterner-Rainer.[42]

Practice at Idria does not differ essentially from that at Monte Amiata described above, except that the use of wood and coal in place of charcoal is mentioned by Sterner-Rainer. The same writer mentions that at Almaden, Spain, two twin-shaft Spirek furnaces are used for treating the low-grade coarse ore. This battery of four shafts has a capacity of 28 metric tons of ore in 24 hours, with a fuel consumption of 650 kilograms of coke.

MEASUREMENT OF FURNACE CAPACITY.

As already pointed out, a statement of furnace capacity in tons a day is not entirely satisfactory, as the character of the ore varies in different mines. Such factors as the bulk density of the ore and the presence of pyrite and of moisture, both free and combined in hydrous minerals, influence the quantity of heat needed for roasting and the rate at which this operation can be carried out. It is interesting to note that the fuel consumption for all the furnaces listed above is approximately the same. The fuel efficiency of the New

[41] Castek, Franz, Die Bestimmung und Verminderung der Verluste beim Quicksilberhüttenwesen: Oest. Ztschr. Berg-Hüt., Jahrg., 58, 1910, p. 231.

[42] Sterner-Rainer, Roland, Der derzeitige Stand der Quicksilberhüttenwesens in Europa: Oest. Ztschr. Berg-Hüt., Jahrg. 62, 1914, pp. 53–535. English translation by C. N. Schuette, Production of quicksilver in Europe: Chem. and Met. Eng., vol. 19, 1918, pp. 721–727; vol. 20, 1919, pp. 32–35 and pp. 82–84.

Idria, Calif., furnaces is distinctly lower than when wood fuel was used, owing to the higher calorific value of the oil fuel.

KROUPA FURNACE.

Sterner-Rainer mentioned a new type of shaft furnace known as the Kroupa furnace, which was to be constructed at Idria (Austria) in 1915. The design of this furnace evidently follows that of the vertical by-product coke oven heated by producer gas. No subsequent mention of this furnace has been found in the literature, and it is not known whether its operation was successful.

THE SCOTT FURNACE FOR ROASTING QUICKSILVER ORE.

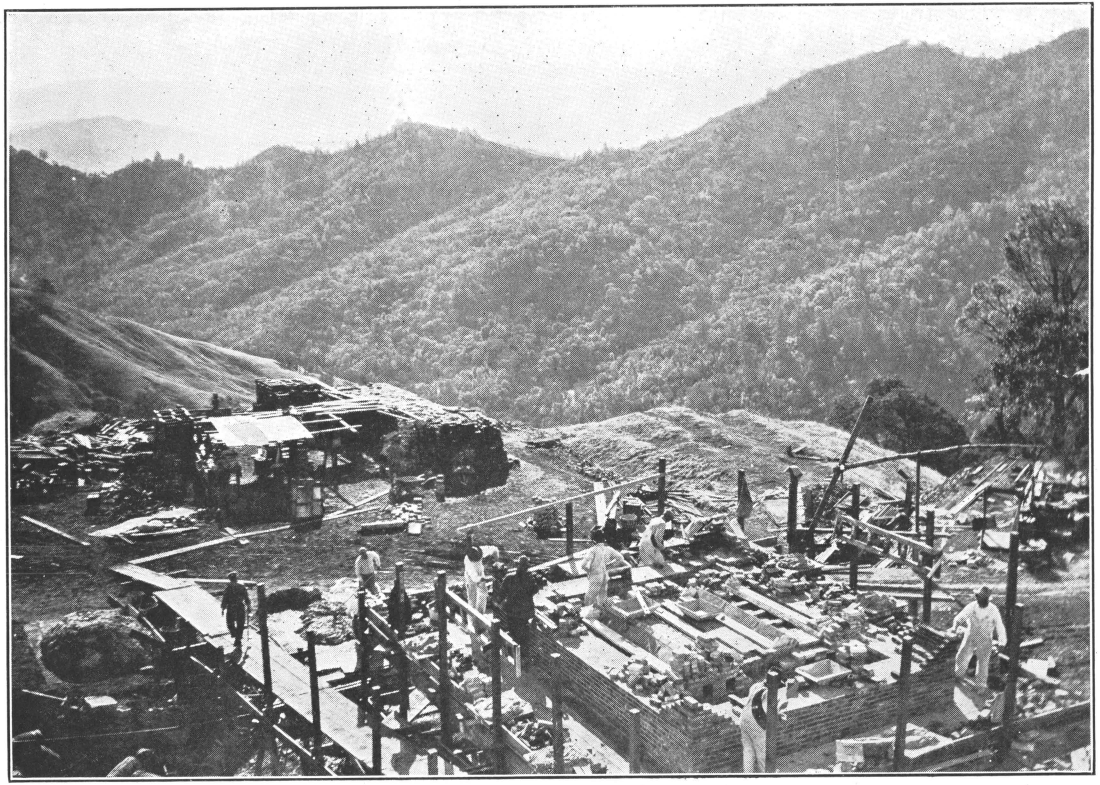
HISTORICAL SKETCH.

The development of the Scott furnace at New Almaden, Calif., marked a great forward step in quicksilver metallurgy, and to date this furnace has been responsible for a large part of the quicksilver production of the United States. Three men—namely, Robert Scott, furnace mason at New Almaden; H. J. Hüttner, mechanical engineer; and J. D. Randol, then manager for the Quicksilver Mining Co.—were responsible for the development of this furnace. The first furnace was built at New Almaden in 1875. Christy [43] gives a full description of the construction and operation of this first furnace. No essential changes have been made in the design up to the present time, the minor changes being simplifications of design. Plate I (frontispiece) shows a Scott furnace under construction.

The principal ideas which are combined in the design of the Scott furnace may be traced in part to the old intermittent furnaces then in use at New Almaden. As described by Christy, the latter consisted of an ore shaft communicating through perforated end walls with a fire box on one side and a dust chamber, to which the condenser was connected, on the other. The ore was introduced into the furnace from above, and as the charging proceeded, heating flues connecting the openings in the fire-box wall with those in the dust-chamber wall were constructed either of "adobes" or large lumps of ore. At the completion of a run the roasted ore was drawn off through discharging doors in the side walls at the bottom.

In the Scott furnace the open ore shaft is replaced by one or more pairs of narrow shafts containing shelves of fire-clay tile set at an angle of 45°, and so arranged as to form zigzag paths for the ore.

[43] Christy, S. B., Quicksilver reduction at New Almaden: Trans. Am. Inst. Min. Eng., vol. 13, 1884–85, p. 547.



FORTY-TON SCOTT FURNACE UNDER CONSTRUCTION AT A CALIFORNIA QUICKSILVER MINE. TILE BRICK OR SHELVES FOR ORE ARE SHOWN. (SEE PAGES 47–72)

Shortly before his death, at San Jose, Calif., on June 25, 1920, Scott gave the authors a brief account of the development of the furnace which now bears his name.

Before he came to California, Scott had been a brick mason at a cereal mill at London, Canada, where he had designed and constructed a roasting or drying apparatus consisting of inclined iron plates. While he was working as a brick mason at various California quicksilver plants he saw the possibility of applying a similar idea to the construction of permanent flues in a furnace for the continuous roasting of ore. He made a model embodying his ideas, and for about 10 years tried unsuccessfully to interest quicksilver men in its development. In the meantime at New Almaden and other mines in California there was a growing demand for a furnace in which the fine and relatively low-grade ore could be economically treated. In 1875 J. D. Randol became interested in Scott's idea and a large working model of the furnace was constructed. The success of this led to further development, in which Mr. Hüttner assisted. This furnace was patented (U. S. patent 183934) by Hüttner and Scott on October 31, 1876. From this date until his death Scott made a business of constructing the furnace which is now generally known as the Scott furnace.

Attention has been called to the similarity between the general design of the Scott furnace and the Hasenclever-Helbig shelf furnace,[44] which was developed in Europe in the early seventies for the roasting of fine sulphide ores. It seems doubtful whether the latter furnace was known to those who were responsible for the development of the Scott furnace at New Almaden, and the authors are of the opinion that the ideas embodied in the Scott furnace were derived from the sources given.

GENERAL DESIGN AND CONSTRUCTION.

The Scott furnace consists essentially of one or more pairs of narrow vertical shafts containing shelves of fire-clay tile set at an angle of 45° and placed alternately against the walls of the shafts. These form a series of inclined hearths down which the ore moves by gravity. The ends of the shafts are closed by perforated walls, these perforations communicating with the fire box and the dust chambers. The ore is heated by the hot gases from the fire box, which pass through the flues formed by the inclined tile. A single shaft usually contains about 26 tiers of tile, making the vertical dimension of the shaft itself about 30 feet. The length of the furnace is determined by the number of tile used in each tier, usually

[44] Lunge, G., Sulphuric acid and alkali. London, 1879, 1st ed., p. 196.

from two to five; the width of the furnace is determined mainly by the number of shafts. Small furnaces are built with one pair of shafts; the usual furnace has four shafts, but a few furnaces have been made with six shafts.

A FOUR-SHAFT FURNACE.

A typical four-shaft furnace with a nominal rating of 40 tons of ore in 24 hours is shown in Plate VI. This furnace has four ore shafts and three tiles to each tier, and is commonly referred to as a three-tile four-shaft furnace. The general relation of the fire box and dust chambers to the ore shafts is shown in the side section.

The walls separating the fire box and dust chambers from the ore shafts are commonly known as the "pigeon walls," and are perforated by ports known as "pigeonholes," which correspond to the flues under the various tiers of tile. The relation of these pigeonholes to the tile is clearly shown in the end section.

The fire box communicates directly with the flues in the lower third of the furnace. The hot gases, after traversing these flues, rise in and pass through the chamber at the rear of the furnace to the next set of flues, and enter the chamber above the fire box. Here the gas stream again takes an upward path, and after traversing the flues in the top third of the furnace, passes through the upper dust chamber to the exit pipes. The gas stream thus follows an S-shaped path and, in traversing the ore shaft three times, approximates countercurrent flow. Each pair of ore shafts is charged through a narrow throat which extends the length of these shafts.

The roasted ore leaves the furnace at the bottom through what is known as the "draw."

The Scott furnace is usually built upon a massive foundation of either masonry or concrete. Sometimes masonry arches supported upon piers have been used; this is necessary when the furnace is equipped with a mechanical discharging device that delivers the roasted ore into cars in a central tunnel under the furnace.

BRICK AND TILE USED.

The outer walls, and sometimes the upper part of the inner walls as well, are built of ordinary red brick. When transportation is difficult or expensive, this brick is usually made near the furnace site. Fire brick is used for the interior walls of the furnace, including the pigeon wall, and also the lining of the combustion chamber and the lower rear dust chamber. Only standard fire brick and a few regular shapes are used. The tile are made of a good grade of fire clay, and are 36 by 15 by 3 inches. In some of the early Scott furnaces, special shapes of tile and fire-clay supports were used; but this has long been abandoned in favor of the standard materials.

IRON SHEATHING.

Two of the original Scott furnaces at New Almaden were entirely encased in cast-iron plates, no doubt patterned on the construction of the Exeli continuous coarse-ore furnaces then in use. This iron sheathing was found to be unnecessary and has not since been used. Heavy iron plates were at one time built into the furnace foundation in order to collect any mercury which might work down through the furnace walls, but this precaution has also been found unnecessary; the only iron plates used in the bottom of the furnaces are those forming the bottom of the draw openings.

BRACINGS.

For the external bracing of many Scott furnaces wooden timbers have been used both for the buckstays and the horizontal members. This construction is possible, as the exterior of the furnace never becomes hot enough to char the wooden timbers. The use of wood is, however, open to the objection that in case of a fire the furnace is likely to suffer serious damage through destruction of its supports. The furnace shown in Plate VI has 6-inch channel irons for buckstays and 6-inch angle irons at the corners; no horizontal braces are used. In either construction the amount of metal used on the exterior of the furnace is small.

MATERIALS FOR CONSTRUCTION.

A list of the materials needed for the construction of the furnace shown in Plate VI, exclusive of foundation, is given. It is assumed that the outside, pigeon, and central walls extend to the level of the tracks, running below the draw openings where the foundation begins, and that all walls are red brick from the foundation to the draw level. Fire brick is assumed for the following parts: The three interior walls and the pigeon walls between the draw level and the offset; the lining of the main fire box, of the rear lower dust chamber, and of the side walls; and the entire pigeon wall of the lower rear dust chamber and all arches over the dust chambers. Red brick is assumed for the remainder of the interior construction and for all other parts of the furnace.

Materials required for construction of a 40-ton Scott furnace.

Brick and bonding material.

Common brick	140,000	Tile	360
Fire brick	25,000	Fire clay ... tons	5
Fire-brick arch	1,000	Lime ... barrels	150
Fire-brick end skew	800	Cement ... do	10
Fire-brick side skew	2,200	Sand ... tons	55

Iron work.

12 iron doors and frames, $1\frac{1}{2}$ by $1\frac{1}{2}$ feet.
6 sheet-iron counterbalanced draw doors, $2\frac{1}{2}$ by $2\frac{1}{2}$ feet.
6 cast-iron plates for the draw, 3 by 3 feet by 1 inch.
6 cast-iron plates for the draw, $1\frac{1}{2}$ by 3 feet by 1 inch.
6 sheet-iron draw bottoms, 10 by 3 feet.
12 cast-iron plates to support walls, 1 by 3 feet by 1 inch.
6 cast-iron plates to support walls, $2\frac{1}{2}$ by 3 feet by 1 inch.
8 I-beams or rails, 11 feet long, for throat.
6 slide doors, frames, and levers for charge.
4 6-inch angle irons 16 feet long.
4 6-inch angle irons 18 feet long.
6 6-inch channel irons 14 feet long.
6 6-inch channel irons 18 feet long.
16 $1\frac{1}{4}$-inch tie-rods 18 feet long.
16 $1\frac{1}{4}$-inch tie-rods 23 feet long.
48 $1\frac{1}{4}$-inch stud 2 feet long.
112 $1\frac{1}{4}$-inch hole washers 1 inch thick.
112 $1\frac{1}{4}$-inch nuts.
200 cast-iron peephole frames and plugs.

The approximate weight of the iron work is 8 tons.

INTERNAL CONSTRUCTION.

In the furnace shown, which is designed for burning wood, the bottom of the combustion chamber is formed by a perforated fire-brick arch, which takes the place of grate bars. This same construction is suitable for burning oil, if a burner is placed at each fire door. When coal is to be used, the brick arch is replaced by the regular iron grate bars, which are also used sometimes for wood fuel.

USE OF PREHEATED AIR.

Preheated air for combustion enters the ash pit from the flues below the grate. The air to be preheated finds its way into these flues through the chamber below the lower rear dust chamber. Hot air from the draw also finds its way to these flues through the column of roasted ore; and if the ore is coarse, most of the air may enter in this way. This arrangement for utilizing the heat in the roasted ore for preheating air also provides for the recovery of any mercury vapor which may still be escaping from the ore.

In Scott furnace practice in this country, this feature of the furnace has not always been fully utilized. In general, it seems likely that additional heat could be recovered from the roasted ore by providing a few additional flues for preheating the air, 10, for example, in a four-shaft furnace, instead of 6, as shown in the accompanying design. Moreover, the construction of this part of the furnace should be adapted to its working conditions.

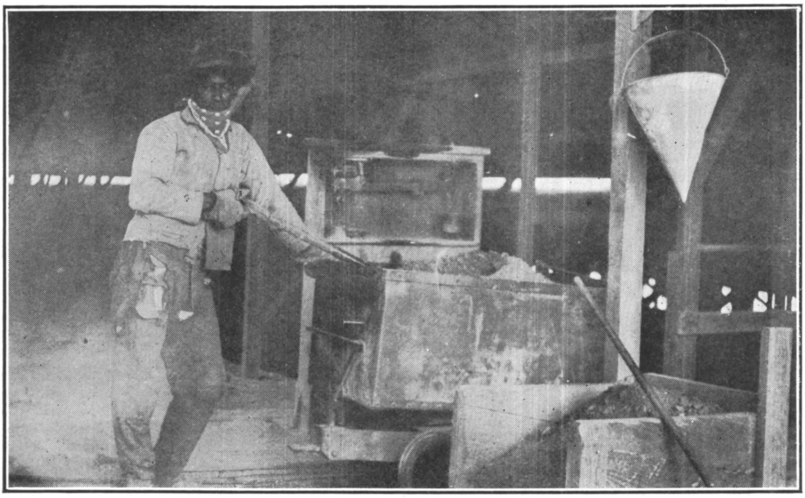
When wood or coal is used as fuel, part of the preheated air should be introduced above the fire bed. This can be done by placing the grate or fire arch so that part of the flues for preheating the air open above it, or by leading the hot air from certain flues across the ash pit in iron pipes, and thence through passages in the end wall of the furnace to a point above the fuel bed. With oil firing, preheated air is not necessary in order to obtain the requisite temperature and completeness of combustion in the fire box. In this case, the hot air may be used to better advantage for drying and preheating the ore, as described already (p. 22).

TOPS OF CHAMBERS.

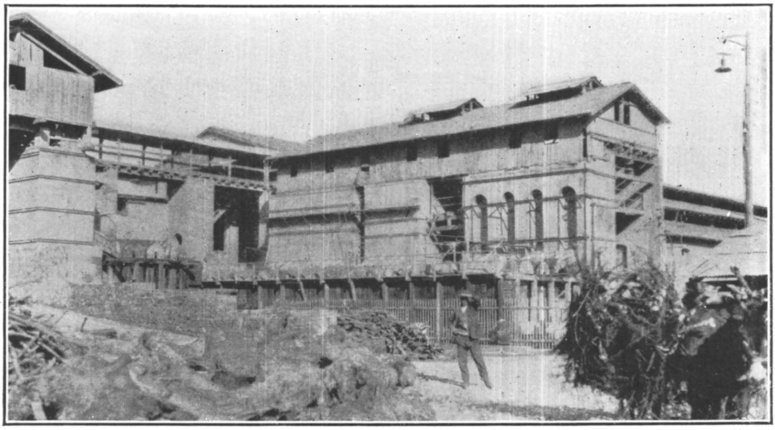
The top of the combustion chamber is formed by two arches sprung between the side walls and a central supporting arch, which in turn is sprung between the outside wall and the central wall that separate the two pairs of ore shafts. This construction is needed to avoid any thrust upon the pigeon wall, which is the most highly heated part of the furnace. The tops of the other dust or vapor chambers are made in like manner. In some furnaces the single large fire box is divided into two parts by a central wall below the central supporting arch. This arrangement makes it somewhat easier to insure uniform heating of the two sides of the furnace, particularly when the furnace is exposed to a strong wind from one direction. Clean-out doors are placed in the sides of the furnace at the bottom of each of the dust chambers. In anything larger than a two-shaft furnace these doors should be provided on both sides.

DRAW OPENINGS.

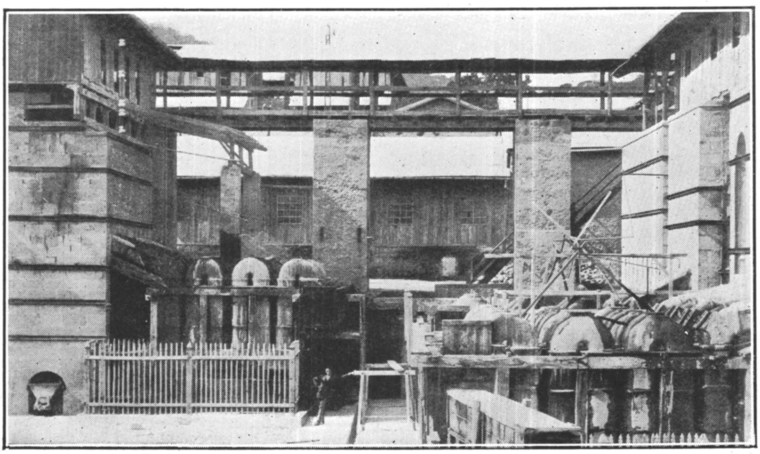
The number of draw openings on each side of the furnace for discharging the roasted ore is usually the same as the number of tile in a single tier—three for the furnace here shown. These openings are separated by low transverse walls that support the ironwork, including the beam or plate on which the dividing wall of each pair of ore shafts rests. The parts that are more exposed to the heat of the roasted ore are usually made of cast iron, but sheet iron may be used for the plates that form the bottom of the draw. Counterpoised



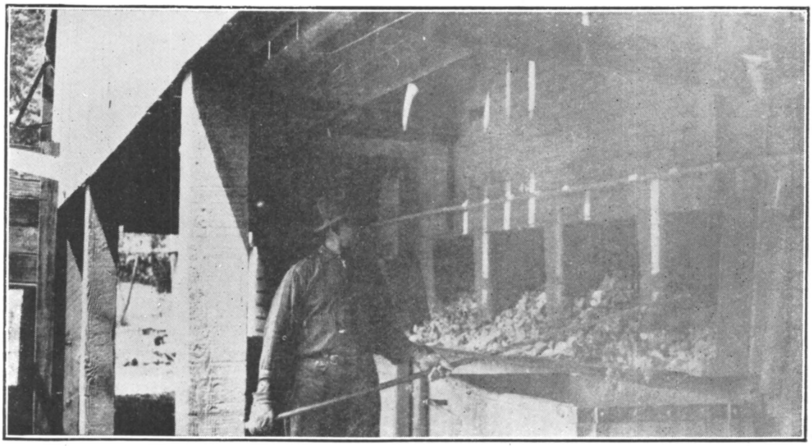
A. SAMPLING AND WEIGHING ORE AT CHISOS MINE, BREWSTER COUNTY, TEX.



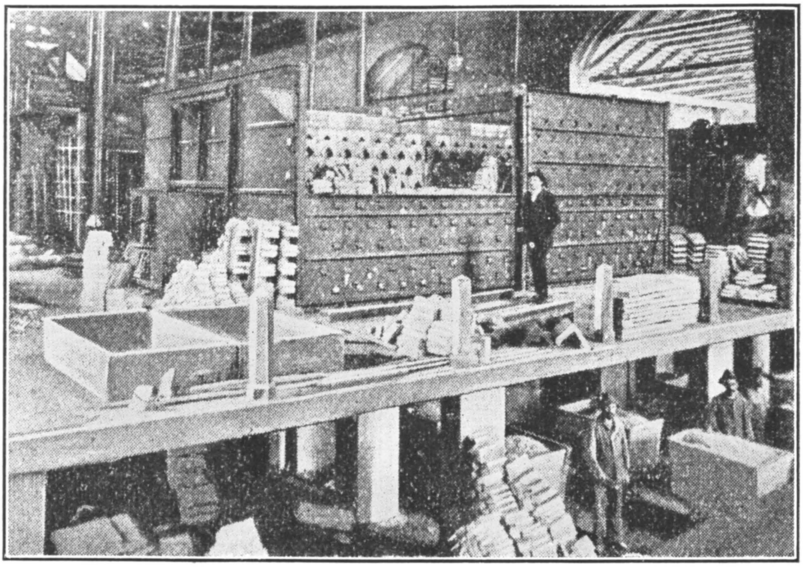
B. BATTERY OF SPIREK COARSE-ORE FURNACES, ABBADIA SAN SALVATORE, ITALY



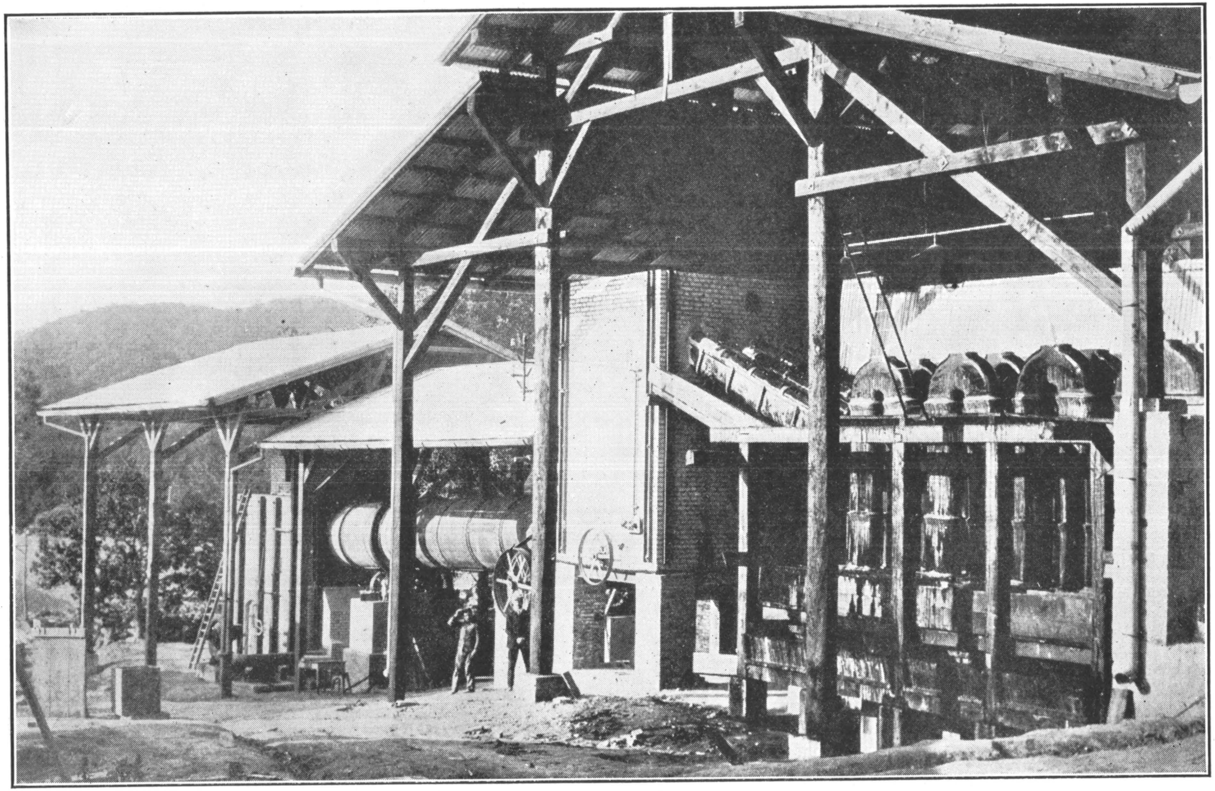
C. LATEST SPIREK FURNACE AND CERMAK CONDENSERS AT ABBADIA SAN SALVATORE, ITALY



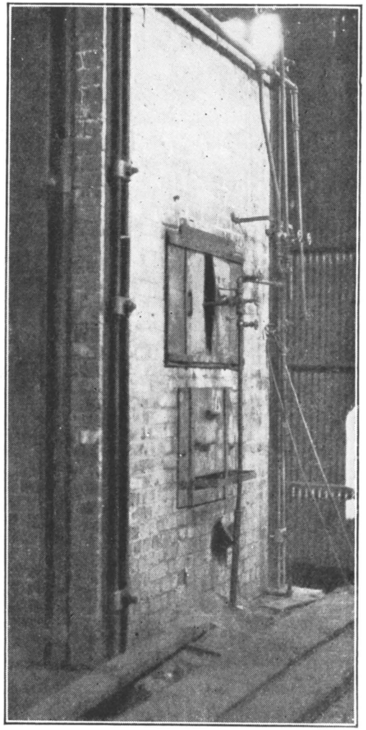
A. DRAWING ROASTED ORE FROM A SCOTT FURNACE



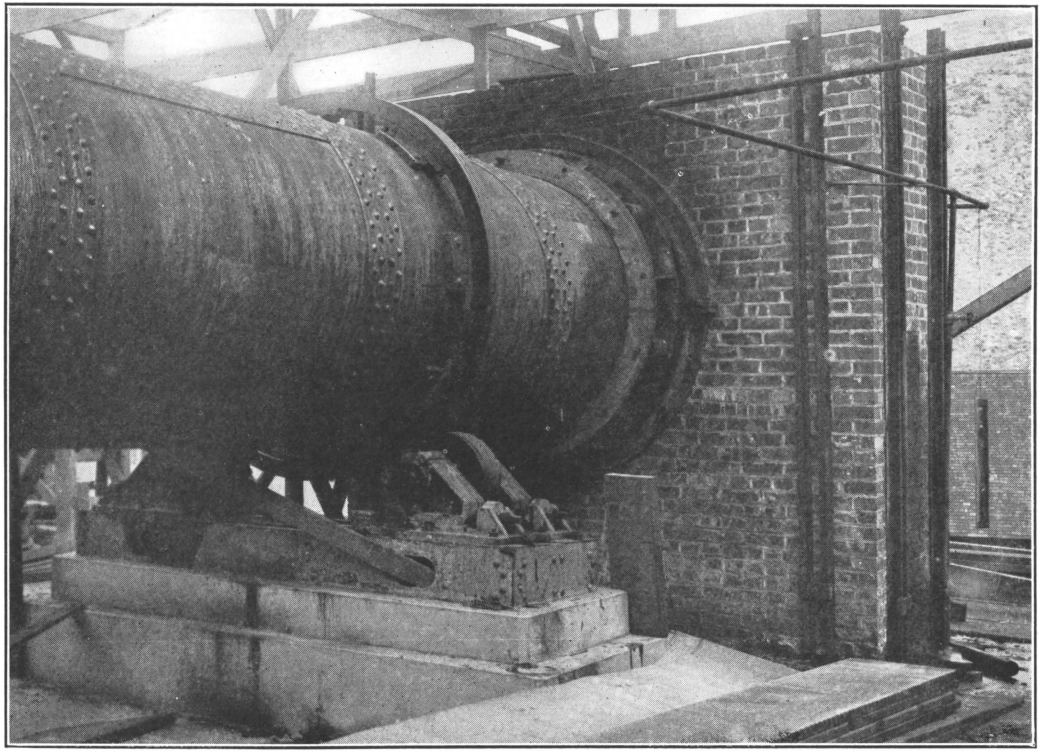
B. CERMAK-SPIREK FURNACE UNDER CONSTRUCTION AT IDRIA, AUSTRIA



ROTARY KILN, DUST CHAMBER, AND CERMAK CONDENSER SYSTEM AT ABBADIA SAN SALVATORE, ITALY



A. FIRE BOX AND OIL BURNER FOR ROTARY KILN AT NEW IDRIA, CALIF.



B. ROTARY-KILN CONNECTION TO FIRE BOX AT NEW IDRIA, CALIF., SHOWING FUME-TIGHT JOINT doors are usually provided for closing the draw openings and are essential whenever the furnace is exposed to a strong wind.

With the type of draw construction illustrated, the roasted ore is hoed by hand into cars at the sides of the furnace, as shown by Plate VIII, *A*. Christy [45] has described a form of mechanical discharging device which was used on one of the furnaces at New Almaden. A horizontal sliding gate worked by a lever at the side of the furnace is provided for each flight of tile. The roasted ore is discharged into a hopper from which it may be drawn at intervals into a car running in the tunnel below the furnace. Few of the many Scott furnaces built in this country have been equipped with a mechanical discharging device, and the advantage of such equipment is doubtful. With hand drawing, the quantity of ore discharged from each opening can be nicely adjusted to the condition of the furnace. The increased efficiency of operation and capacity, would probably more than offset any saving in labor through the use of a mechanical discharge.

SUPPORT FOR THE TILE.

One of the most important details in the construction of the Scott furnace is the method of supporting the tile. This is shown in the plan view of Plate VI (see p. 48). The tile rest on brackets 4 inches wide, which are built out from the sides of the ore shafts. The horizontal spacing of the brackets is 18 inches, so that each tile is supported at the middle as well as at the two ends.

The tile are supported at their lower corners by what are known as "brace brick," which rest in half-inch depressions in the tier of tile on the opposite wall just below. Ordinary fire brick cut to suitable length are used for this purpose, and in some places the upper edge is beveled to decrease resistance to the descending ore stream.

It will be noted that the bearing point of the brace brick on the lower tier of tile is just above the ends of the brackets supporting this tier, so that the thrust of the supported tile is really borne by these brackets. This detail of construction obviates any tendency of the tile to tip up from the brackets.

The two end tile of each tier project about 3 inches into the pigeon wall. In a furnace which is three tile long, two tile of the three in each tier are thus supported by the end walls of the ore shaft, making the construction particularly substantial. With furnaces which are four or five tile long, the danger of the tile becom-

[45] Christy, S. B., Work cited, p. 547.

ing displaced is greater, but unless the furnace is improperly handled, the displacement of brace brick and tile rarely occurs.

In a few furnaces, cast-iron plates have been used in place of tile for the topmost tiers, but there is little advantage in iron and, when the ore carries much sulphur, acid may form at this point and rapidly corrode the metal.

The space between the successive tiers of tile, sometimes known as the "shelf slit," is from 4 to 6 inches wide, the latter being the common width. Attention is called to the fact that as the ore passes from one tile to the next below, that part which was at the bottom of the bed next to the tile is brought to the surface; this turning over greatly facilitates the uniform roasting of the ore.

PIGEONHOLES.

The end walls of the ore shaft are perforated by pigeonholes which register with the open spaces or flues under the tiers of tile. These pigeonholes are usually 6 inches wide and 8 inches high. In some furnaces, the bottom of each pigeonhole is beveled both ways from the center line, forming what is known as a self-clearing pigeonhole. This is not the common practice, however, and the simple rectangular openings require only occasional cleaning. In one of the early furnaces constructed at New Almaden, a system of flues was arranged in the lower part of the pigeon wall adjoining the main fire box, through which air was introduced for the dual purpose of cooling this wall and supplying additional preheated air for the combustion of any unburned gas which might enter the pigeonholes from the main combustion chamber. As simplification has been the object of changes in construction, this device has not been used in any Scott furnaces that have been built in recent years.

PEEPHOLES.

Peepholes are placed in the outer end walls of the furnace, corresponding to the pigeonholes in the inner walls. These peepholes, which are either rectangular or circular in section, are formed by placing cast-iron frames, or short sections of pipe, in the brickwork. For closing them, tapering iron plugs luted with clay are used. In case of a hang-up, an iron rod may be introduced through the peepholes to loosen the ore.

FURNACE THROATS.

The ore enters the furnace through the narrow throats at the top. These are ordinarily closed by simple slide gates of the same length as the tile. In some furnaces these gates are omitted as the ore forms a self-supporting column from the draw to the charge hoppers.

The maximum size of the ore is ordinarily limited to $2\frac{1}{2}$ to 3 inch pieces. The ore should, of course, be so dry that it will not stick.

Ordinarily, the ore column in the throat forms a seal which prevents the escape of fumes from the charge hoppers. At some plants,

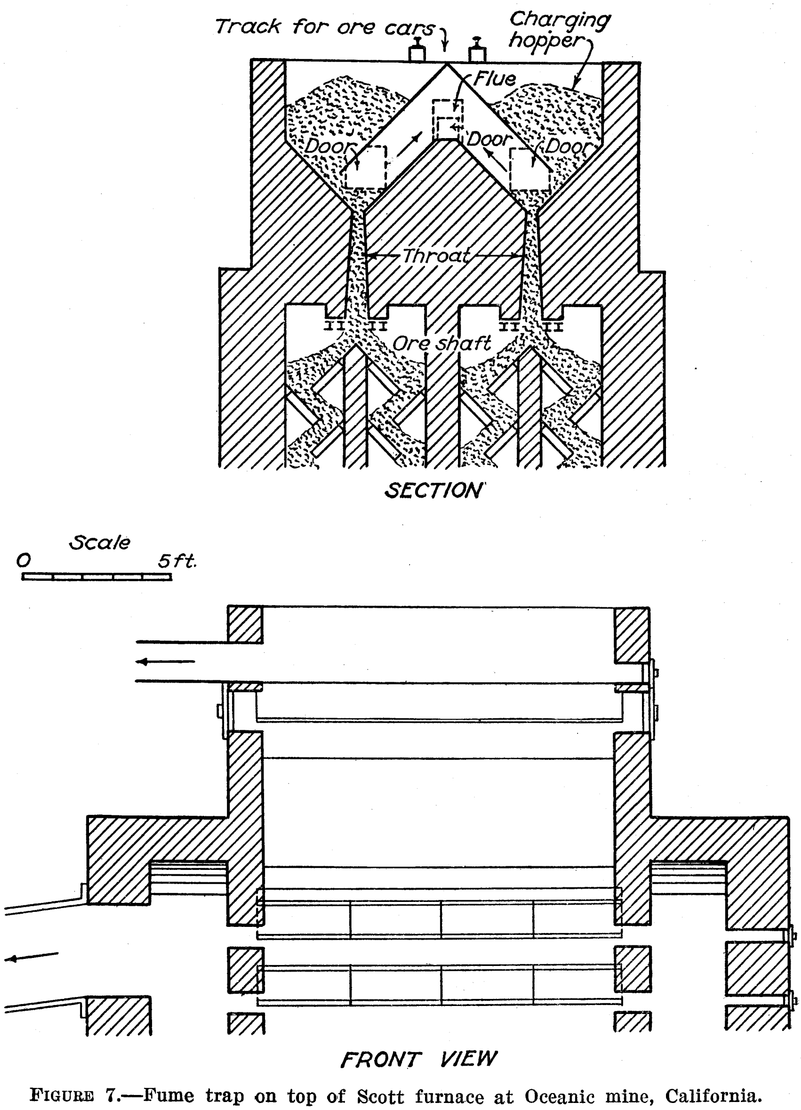


FIGURE 7.—Fume trap on top of Scott furnace at Oceanic mine, California.

tight-fitting wooden covers are provided for the charge hoppers. The two Scott furnaces at the Oceanic mine, San Luis Obispo County, Calif., are equipped with a device known as a fume trap, devised by Murray Innes, of San Francisco. This is shown in Figure 7. The large gable is made of wooden planking covered with sheet iron. A small wooden flue leads from the space under this gable to a wooden chamber which serves as a condenser. The chamber is provided with a short wooden stack passing up through the roof of the building. Doors are provided in one of the end walls so that the throats of the furnace are accessible. This point is important, as it should always be possible to get at the furnace throats in order to loosen ore that bridges, or to remove foreign objects.

The ore is ordinarily brought to the furnace in tramcars, but a Scott furnace built at the Senator shaft at New Almaden, Calif., is provided with an inclined belt-conveyer, which discharges into a small ore bin just above the furnace. At some point the feed for the furnace must be passed through a screen or grizzly to prevent oversize pieces from reaching the furnace throat. It is common practice to provide a screen for the top of the charge car so that oversize material may be rejected at the ore bin.

EXIT PIPES.

The mercury-laden gases escape from the furnace through flues which connect with the upper part of the upper rear dust chamber. A flue or exit pipe is usually provided for each pair of ore shafts. Various forms of construction have been used for these pipes, including circular sheet-iron pipes, square and circular cast-iron pipes, and square brick flues. Some form of damper for closing the exit pipes is usually provided. The exist pipes generally slope away from the furnace so that any mercury condensing therein will flow to the condenser.

FURNACE HOUSING.

Little need be said about the housing of the furnace, except that it is considered good practice to support it independent of the furnace. At some plants the tramway for charging the furnace is supported by the framework of the building so that the furnace itself supports only its own weight.

COST OF CONSTRUCTION AND OPERATION.

CONSTRUCTION COSTS.

Construction and operation costs vary so widely with local conditions that a general statement on the subject has little significance. As a basis for estimating the cost of a Scott furnace at any given point, data are presented below covering the cost of material and labor for a furnace of a nominal rating of 40 tons, such as is shown in Plate VI (see p. 48). These figures are based on information gathered in regard to the cost of several Scott furnaces recently built in California, and the unit prices given correspond in general to those prevailing before 1914.

Cost of construction of a 40-ton Scott furnace.

	Price.	Total.
MATERIAL.		
140,000 red brick	$18 per M	$2,520
29,000 fire brick	$40 per M	1,160
360 tile	$3 each	1,080
5 tons fire clay	$15 per ton	75
150 barrels lime	$1.50 per barrel	225
10 barrels cement	$2 per barrel	20
55 tons sand	$3 per ton	165
8 tons iron and steel	$60 per ton	480
LABOR AND INCIDENTALS.		
75 days, 5 brick masons, $8 per day		3,000
75 days, 2 hod carriers, $5 per day		750
20 days, 1 carpenter, $5 per day		100
75 days, 3 helpers, $5 per day		790
Blacksmith work		155
Superintendence		1,000
Incidentals		480
Total		$12,000

It will be noted that, apart from an item covering the superintending of actual construction work, no amount is included for engineering or other charges which would probably be involved if the work were done under contract. Items such as housing, ore bin, ore crusher, and auxiliary equipment are also not included in the list. The cost of transporting the material to the site of the furnace will, in general, have to be added to the above, and for inaccessible localities may double the above cost. As previously stated, the red brick may be manufactured on the spot, and in some places the lime is burned locally. Various cost figures which have been obtained for the burning of red brick at furnace sites indicate that the cost a thousand will be from $25 up.

The cost of a complete Scott-furnace plant has been estimated at $500 to $1,000 a ton-day capacity. On the basis of the prices given above, and assuming that the condenser is inexpensive, like the one described on page 129, the cost of a complete Scott-furnace plant of 40-ton capacity, including furnace housing and auxiliary building, ore bins, crushing equipment, condenser, and the various accessories, would be approximately $22,000. This corresponds to a cost of $550 per ton-day capacity.

OPERATING COSTS.

An estimate of the direct operating cost of a 40-ton Scott-furnace plant, exclusive of depreciation, repairs, or general overhead expense, is given below. With reasonably good arrangement the furnace crew can also take care of condenser operation, including the periodic clean-up of the condenser and bottling of the mercury.

Operating cost of a 40-ton Scott-furnace plant for 24 hours.

Rock-crusher attendant, 1 shift, at $3	$3.00
Power for crusher	.50
Furnacemen, 3, at $3.50 per shift	10.50
Foreman, 1, at $5 per day	5.00
Helper, 1, at $3 per day	3.00
Wood, 2 cords, at $5 per cord	10.00
Total	32.00
Cost per ton on the basis of 40 tons in 24 hours	.80

In the above tabulation wood is the assumed fuel. The corresponding oil requirement for the capacity of 40 tons a day would be about 7 barrels (294 gallons). Allowing for transportation and storage, the cost of oil at the plant may be estimated at $3 per barrel, which would bring the cost of furnacing the ore up to $1.08 a ton. However, it would probably be possible to increase the capacity of the furnace to 50 tons, making the cost a ton about 95 cents. Actual cost figures from 10 different plants in California and Texas range from 60 cents to $1.10 a ton. The over-all cost of ore treatment at a Scott-furnace plant, including all charges, may be roughly estimated at $1 to $2 a ton.

A 40-ton plant is probably the most economical to run, as a larger furnace would require more than one regular attendant a shift.

OPERATING PRACTICE.

In starting up a furnace from two to three weeks is usually taken to bring it up to working temperature. During the early stages the furnace is left empty, but as the temperature rises old roasted ore or barren rock is fed through slowly. When the desired working temperature has been reached, ore is charged, and the regular routine of drawing and charging is begun. During the heating-up process the nuts on the tie-rods or staybolts must be eased off from time to time to take care of the expansion of the furnace. If this whole cycle of "breaking in" a furnace is done carefully, little cracking of the brickwork will occur.

The common practice is to charge ore to the Scott furnace every half hour. The first step is to discharge a suitable amount of roasted ore, which, with the ordinary hand-worked furnace, is done with large iron hoes or rakes. A skilled "draw man" is able to judge by the appearance of the roasted ore when a proper amount has been taken from each draw opening. Normally, the last of the ore descending into the draw is at a dull-red heat. The drawing is usually somewhat heavier from the end of the furnace next to the main fire box. Instead of discharging the roasted ore directly into the waste or draw cars, it is usually left to cool on the forward part of the draw plates.

Some dust is always formed during drawing, the amount varying widely with the character of the furnace feed. For this reason, and because of the possibility that a little mercury vapor may sometimes escape from imperfectly roasted ore, the space about the furnace draws should be well ventilated.

TESTS FOR COMPLETE ROASTING.

Panning of samples of roasted ore is commonly practiced to determine whether the ore has been completely roasted. For this purpose a large pan is used, and a composite sample is made up by taking a small scoopful from each draw opening. If any cinnabar is detected, samples from the different draw openings are panned separately in order to determine what part of the furnace is not working properly. The appearance of cinnabar in the furnace discharge can usually be traced to a cold section of the furnace; then little or no ore should be drawn therefrom until conditions become normal. With a given furnace feed, a skilled operator becomes remarkably expert in interpreting the results of panning both the raw and roasted ore.

The practicability of such panning depends on the fact previously mentioned, that with most ores the cinnabar tends to concentrate in the fine material. In ore which has passed through the furnace the cinnabar, although sometimes darkened, can be readily detected in the residue left in the pan. At some plants, in addition to the panning test, an average sample of the roasted ore from each shift, or from each 24-hour period, is set aside for subsequent assaying.

When an ore contains pyrite or other sulphide of iron, the presence of pyrrhotite in the furnace discharge does not necessarily indicate imperfect roasting. With some ores, iron sulphide is completely oxidized, and furnishes part of the fuel needed for roasting. With other ores, pyrrhotite will be found in the roasted ore, even though the mercury has been completely expelled.

CHARGING ORE.

As the roasted ore is withdrawn from the bottom of the furnace, fresh ore enters from the charge hoppers at the top. The gates to the charge hoppers usually remain open, and are used only in case of emergency. Ordinarily enough ore will be in the charge hoppers to maintain the seal at the furnace throat, so that unless the drawing is unusually heavy, the charge hoppers are not refilled until drawing has been completed. If the ore does not overflow evenly from the charge hoppers, any obstruction in the throat should be removed with a slice bar and the ore already in the hopper redistributed.

When fresh ore is charged it may descend too rapidly in the furnace, and unduly cool one section. When fine material, such as dust, concentrate, or soot, is charged with the ore, it should be distributed uniformly over the charge.

At one plant, about 10 per cent of the total charge was concentrate, which was fed during two shifts. The ore for this furnace was the run-of-mine material passing a $2\frac{1}{2}$-inch ring. No trouble occurred when the concentrate was evenly distributed.

A very satisfactory method for controlling the quantity of ore sent to the furnace is used at the mines in the Terlingua district, Brewster County, Texas. A platform scale is placed on the track between the ore bin and the furnace, the weights of the scale being permanently set for the quantity of ore desired. A car full of ore is run on the scales, and enough ore is shoveled out of or into the car to balance the beam; a box for excess ore is placed near the scales. This is a good method of controlling the furnace tonnage. Before weighing, a shovelful from each car is thrown into a sample box.

LOOSENING HANG-UPS.

Occasionally, when the ore is wet and sticky, or when some foreign body has entered the furnace, a hang-up occurs. This can be loosened by a hooked rod introduced through the peepholes and pigeonholes. Loosening must be done carefully to avoid damage to the brace brick and tile. The job should be entrusted only to some one familiar with the interior construction of the furnace. For work in the upper part of the furnace, which is below a red heat and therefore not visible, a flexible iron rod of about $\frac{3}{8}$-inch diameter with a hooked end should be used, as such a rod when it comes in contact with a brace brick or other part of the furnace structure will bend and give warning before damage is done.

A slide of ore rarely takes place in the furnace except following a hang-up or after too much fine material has been charged. When a slide happens some of the pigeonholes must be cleaned with a small scraper, and if some ore has entered the main fire box it must be removed to prevent clinkering. The pigeonholes should be inspected regularly.

Several times a year the accumulation of dust is removed from the dust and vapor chambers through the doors provided for the purpose. These doors are usually lined with fire brick to prevent the mercury vapor from condensing on the metal. The dust which has been removed is returned to the furnace, as it may contain cinnabar and some condensed mercury.

FURNACE CAPACITY.

A furnace such as is described above requires about 750 cubic feet (21 cubic meters) of ore to fill it, corresponding to 32 tons (29 metric tons) of ore of ordinary bulk and density. In current practice, ore passes through the furnace in about 15 to 20 hours. In former days, with higher grade ore, the period in the furnace was sometimes 30 hours or more, with a corresponding decrease in capacity. The capacity of a furnace can be increased somewhat by drawing small quantities of ore at frequent intervals, particularly when oil fuel is used. Usually, however, the Scott furnace is not pushed to maximum capacity.

FURNACE FIRING AND TEMPERATURE CONTROL.

Wood and oil are the fuels commonly used in the Scott furnace, although coal and producer gas have been used. A high temperature in the fire box is not needed—in fact, should be avoided, owing to the danger of overheating the pigeon walls. The important considerations in furnace firing are uniformity of temperature, avoidance of an undue excess of air, and complete combustion. The last factor is particularly important, as, apart from the waste of fuel through the escape of combustible material, carbonaceous soot and tarry matter pass on to the condenser system and entrain particles of mercury, thus forming undesirable mercurial soot.

WOOD AS FUEL.

Well-seasoned oak cordwood is an excellent fuel for the Scott furnace, but at plants where this is not available soft wood and even brush have been successfully used. In burning wood, complete combustion of the volatile matter can be promoted by introducing part of the air for combustion, preferably preheated, above the fuel bed. Methods for providing preheated air for this purpose have already been discussed on page 51. Unless fuel is charged frequently and in small quantities, the fire bed has a tendency to develop openings which let an excessive amount of air into the furnace, and cause uneven furnace operation. The problem of efficient use of wood as fuel has been discussed by Holbrook.[46] The type of firebox which he recommends probably could be used to advantage in the Scott furnace.

OIL AS FUEL.

When oil is used as fuel two burners are usually operated in the main fire box—one at each end—and sometimes one or two burners in the lower rear dust and vapor chambers. There are several advantages in burning part of the oil in this rear chamber. The rear of the furnace is maintained at a higher temperature than otherwise, and also a draft is established at the bottom of the rear chamber which draws the hot gases from the main fire box through the lowest tiers of the flues, thus increasing the active hearth area. Owing to the high temperature of the oil flame, the danger of local overheating is less if the oil is introduced at several points. The distribution of temperature both in a wood-fired and an oil-fired furnace is shown in Figure 8. The temperatures recorded are the averages of many observations made with a long thermocouple and represent fairly typical conditions in the Scott furnace with the two types of fuel. The effect of the oil flame in the lower rear chamber, as noted above, can be readily seen from the diagram. A given Scott furnace will usually show an increase in capacity of 20 to 25 per cent when a change is made from wood to oil fuel.

REGULATING TEMPERATURE.

In temperature regulation of the Scott furnace the primary considerations are the proper roasting of the ore and the maintenance of a suitable exit-pipe temperature. The latter should be high enough to prevent by a safe margin the condensation of mercury in the upper part of the furnace or the exit pipe.

RECORDING EXIT-PIPE TEMPERATURES.

To make due allowance for variation in furnace temperature and in the mercury tenor of the furnace charge, the exit-pipe temperature in general should be kept at 40° to 50° C. above the mean saturation temperature of mercury vapor in the gas stream. This saturation temperature, or dew point, depends upon the concentration of the mercury vapor in the gas stream, which in turn depends upon the mercury content of the furnace charge and the volume of gas leaving the furnace.

The diagram given in Figure 9 shows the saturation temperature as related to the volume of furnace gas for furnace charges carry-

[46] Holbrook, E. A., Wood as a fuel for mine power plants: Eng. and Min. Jour., vol. 99, 1915, p. 645.

ing various percentages of mercury. With ordinary Scott-furnace practice, the volume of gas, measured at standard conditions, that

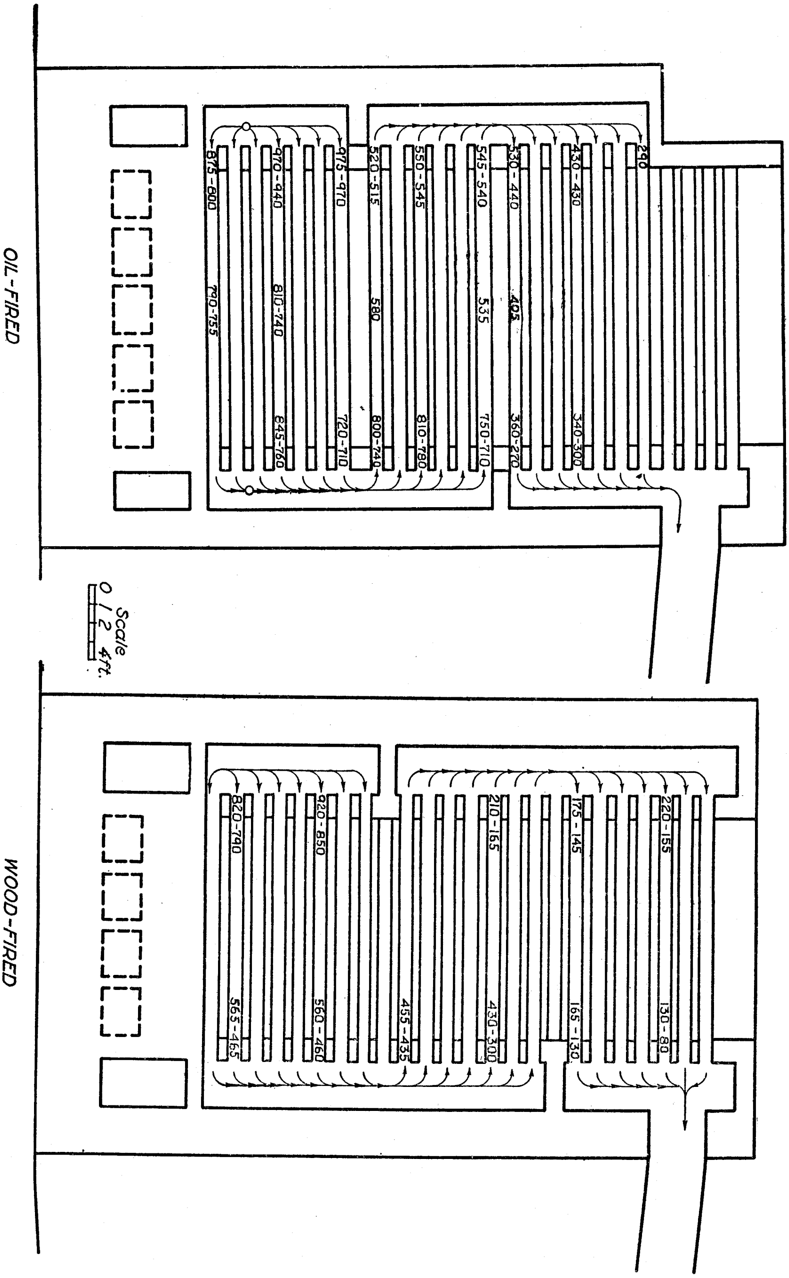


FIGURE 8.—Longitudinal sections of an oil-fired and a wood-fired Scott furnace, showing the temperature distribution.

leaves the furnace for each metric ton of charge, ranges from 400 to 700 cubic meters. If, for example, with a given furnace this gas volume averages 600 cubic meters, and the mean mercury content of the charge is 0.6 per cent, the temperature at which condensation

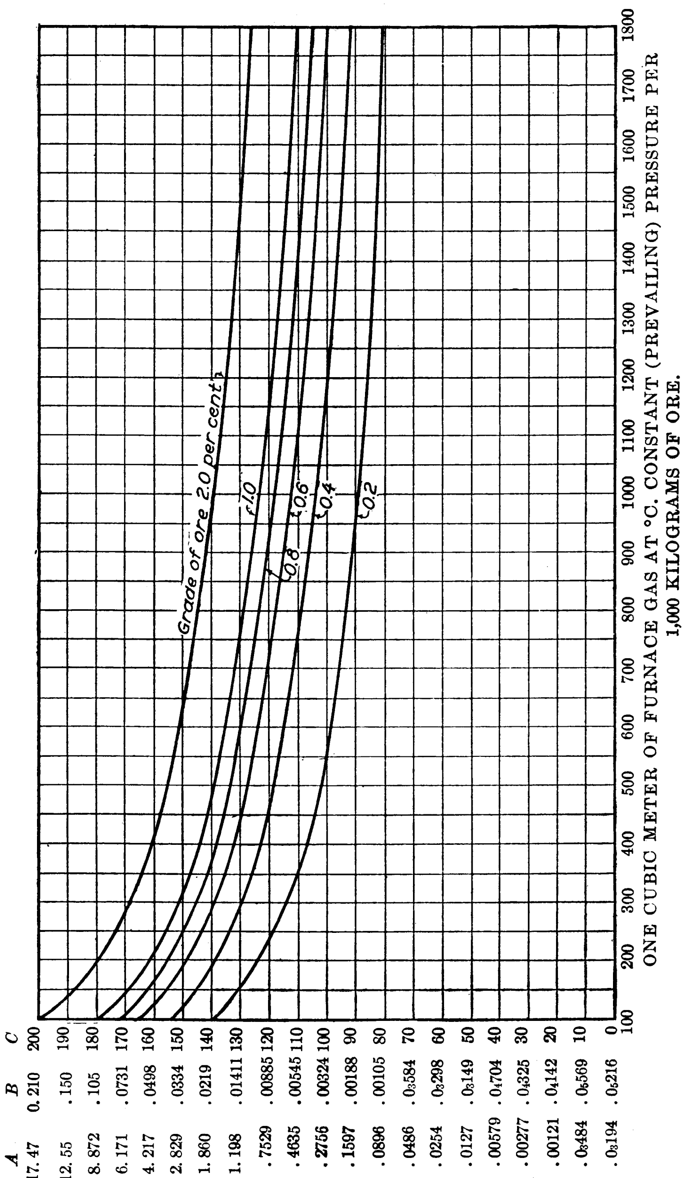


FIGURE 9.—Condensation temperatures for mercury vapor in furnace gases: *A*, Vapor pressure of mercury in millimeters of mercury; *B*, Kilograms of mercury required to saturate 1 cubic meter of air measured at 0° C. and heated under constant pressure to a given temperature; *C*, Temperature, °C. One cubic meter=35.14 cubic feet; 1,000 kilograms=2,204 pounds.

of mercury will begin is a little above 120° C.; consequently an exit-pipe temperature of about 160° C. should be maintained.

A mercurial thermometer is ordinarily used for observing the exit-pipe temperature, but a recording thermometer would be advisable sometimes, as a log of the exit-pipe temperature gives the operator a fairly good index of the way the furnace is being handled. Indicating or recording pyrometers are used at some plants. With oil fuel the temperature in the main fire box is so high that the life of a base-metal couple is short. In this event the thermocouple may be installed in the upper part of the lower rear chamber, as knowledge of this temperature, with the exit-pipe temperature, better enables the attendant to supervise the furnace. The Scott furnace at the St. John's mine, Vallejo County, Calif., is ingeniously equipped with one thermocouple of a recording system placed in the draw. Whenever roasted ore is withdrawn from the furnace a sharp break shows in the graph, thus giving the plant superintendent information as to the regularity with which the furnace was drawn, particularly during the "graveyard shift."

METHODS OF TEMPERATURE REGULATION.

The temperature of the Scott furnace may be regulated in several ways. With oil firing, to adjust a burner exactly by visual observation of the flame is difficult, and the effect on the furnace of a change in the amount of fuel supplied is not evident for several hours. For this reason it is the custom at some plants to work the oil burners without adjustment for days or even weeks at a time and regulate the temperature by changes in the furnace feed. When the furnace shows a tendency to heat up, the drawing of roasted ore is increased until normal conditions are restored, and vice versa. An increase in the amount of fine material in the furnace feed tends to lower the temperature, and, likewise, a cold furnace can be brought back to temperature by charging more coarse ore. With wood firing, the furnace temperature can be controlled by regulating the quantity of fuel.

EFFECT OF SULPHUR IN THE ORE.

When the ore contains more than enough sulphur, either elemental or as pyrite, to furnish the fuel for roasting the ore, the problem of regulating the temperature in a Scott furnace becomes particularly difficult. If air is admitted freely to such an ore, the temperature is likely to rise to a point where the ore will slag and attack the furnace structure. The only means available for controlling the temperature is to run the furnace slowly, and to reduce the amount of air admitted until only a part of the sulphur is burned. Reducing the air has the obvious disadvantage, which is discussed on page 68 of forming much mercurial soot. When heavy sulphide ores are roasted in mechanical furnaces of the McDougal type, excess heat can be prevented by passing through the furnace considerable excess air. This practice, however, is not applicable to the Scott furnace. One possible method of overcoming the difficulty would be to mix limestone with the ore; the decomposition of the limestone absorbs heat and prevents the excessive rise of temperature. This absorption of heat would, however, be offset to some extent by the heat liberated in forming calcium sulphate. This idea has been applied at the Big Bend mine in Brewster County, Tex., where a rhyolitic ore, containing considerable pyrite, was mixed with an approximately equal quantity of ore carrying a limestone gangue. In general, however, the use of barren limerock for this purpose would probably be too expensive.

FUEL REQUIREMENTS.

The fuel requirement of a furnace varies considerably with the character of the ore. Careful measurements made on a furnace treating 65 to 70 tons (59 to 64 metric tons) of dry ore containing very little combustible matter, or substances such as carbonates which would be decomposed in the furnace with the absorption of heat, showed a consumption of slightly less than 7 gallons of 18° B. fuel oil for a ton of charge, equal to 2.65 per cent by weight of the furnace feed. The quantity of wood required with an ore such as the above amounts to 7 to 8 per cent of the weight of the material treated. Records from a plant treating ore carrying about 3 per cent pyrite and a little less than 1 per cent bituminous matter show a wood consumption of only 3 per cent by weight of the material charged to the furnace.

REGULATING THE DRAFT.

The draft of the Scott furnace should be so regulated that it is practically balanced at the exit pipe; in other words, the furnace itself should all be below atmospheric pressure, so that danger of escape of mercury vapor is reduced to a minimum. Under these conditions a certain amount of air is sure to be drawn into the furnace through cracks around the peephole, and even through the brick walls of the furnace; therefore, in order to avert the leakage of gas from the furnace and to obtain good combustion of the fuel and clean roasting of the ore, the presence of a large excess of oxygen in the gas stream leaving the furnace seems unavoidable. This point will be discussed at greater length in connection with the problem of soot formation in the furnace.

SOFT COAL AND PRODUCER GAS AS FUEL.

Soft coal has occasionally been used with entire success in the Scott furnace; but, owing to the remoteness of the quicksilver mines of the United States from the present centers of coal production, coal is little, if at all, used at the present time. Producer gas generated from either wood or coal would seem to be an ideal fuel for the Scott furnace, but except for the experiments of Dennis[47] at the Black Butte mine in Oregon, no attempt has been made to use producer gas in this country.

SOOT FORMATION.

Although the problem of mercurial soot is ordinarily discussed in connection with condensation, some consideration of this subject is in order here. The condenser can collect only the material which is passed on to it from the furnace. As generally recognized, mercurial soot consists of finely divided mercury mixed with more or less mercury compounds and foreign material, which prevent the droplets of quicksilver from coalescing. Among the various substances that may contribute to the formation of soot are (1) ore dust; (2) mercury compounds, including mercuric sulphide, and mercurous and mercuric sulphates; and (3) carbonaceous matter. A part analysis of a few samples of soot collected by the writers is given in Table 5.

TABLE 5.—*Analyses of mercurial soot.*

	A	B	C	D
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Metallic Hg	36.7	25.4	45.7	44.1
$HgSO_4$	3.5	.1	.0	.1
HgS	.7	24.2	18.0	3.7
SO_3	7.3	10.6	3.5	4.4
Insoluble	23.5	18.1	18.9	25.9
Moisture at 100° C	14.1		10.6	11.9
Free carbon			2.9	11.0

The source of these soot samples is as follows:

A—Brick condenser adjoining Scott furnace in which a low-sulphur sandstone ore was roasted with oil fuel.

B—Wooden chamber forming intermediate part of condenser system for a coarse-ore furnace roasting low-sulphur ore with oil fuel.

C—Brick condenser adjoining Scott furnace roasting ore, which contained some pyrite, with pitch-pine fuel.

D—Wooden chamber forming intermediate part of condenser system from which sample C was taken.

The amount of free mercury contained in the soot is more or less accidental. All mercury combined with sulphur is reported as mercuric sulphate. Attention is called to the variation in the amount of sulphate present, and particularly to the variation in the quantity

[47] Dennis, W. B., Shortening the roasting period for mercury ores: Eng. and Min. Jour., vol. 88, 1909, p. 112.

of mercuric sulphide. Different analyses that are quoted in the literature show similar variation in the constituents of soot.

SOURCE OF MERCURIC SULPHIDE.

As indicated above, the formation of soot should be regarded primarily as a furnace problem, because the foreign materials responsible for its formation are derived largely from the furnace itself. The amount of ore dust picked up by the gas stream depends mainly on the physical characteristics of the ore.

Mercuric sulphide, which is one of the most objectionable constituents in mercurial soot, can be produced in several ways: When the oxygen is insufficient, cinnabar may be sublimed from the ore. Under similar conditions, sulphur may be distilled from pyrite and subsequently react with the mercury vapor, or even with mercury droplets as they form during the cooling of the gas stream. Hydrogen from incompletely burned fuel, and water vapor also, may react in the absence of oxygen with sulphides, such as pyrite, to form hydrogen sulphide, which reacts readily with mercury vapor or liquid mercury to form the sulphide. Because of the low ignition temperature of sulphur vapor and hydrogen sulphide, and the readiness with which cinnabar oxidizes, it would seem that these reactions should be largely avoided by maintaining a reasonable excess of oxygen in the furnace atmosphere. The large amount of mercuric sulphide shown in sample B is probably due to a deficiency of oxygen, as analyses of the gas leaving the furnace during this period never showed more than a small percentage of oxygen.

USE OF EXCESS AIR AS A PREVENTIVE.

According to the view generally held by practical quicksilver operators, ores containing carbonaceous matter and pyrite, particularly the latter, are liable to yield a certain amount of soot; the writers' observation at a number of Scott furnace plants confirms this. The question then arises as to the amount of excess air required to prevent the formation of soot.

Observation of the oil-fired Scott furnace at the New Idria mine, San Benito County, Calif., showed that with 90 per cent of excess air in the exit-pipe gas, no soot was formed in the furnace; but when, in an effort to reduce fuel consumption, this excess was reduced to 60 per cent, soot was formed. Leakage of air into the furnace was unavoidable; probably the excess of oxygen present in the furnace atmosphere in the upper part of the roasting zone was somewhat less than the quantities given. Moreover, when substances that can consume oxygen, other than cinnabar, are present in the ore, the oxygen below the surface of the bed of ore may be completely exhausted, thus permitting the volatilization of mercuric sulphide. Even though the temperature and oxygen concentration at the point at which this occurs are enough for the complete oxidation of this mercuric-sulphide vapor, the chilling of the gas stream by contact with cool ore may check the oxidation and permit a certain amount of mercuric sulphide to pass on to the condenser.

The idea that something of this sort may occur in the Scott furnace is supported by the fact that when the first rotary furnace was installed at New Idria, no soot was formed in the furnace when the escaping gas stream contained only 40 per cent of excess air. Apparently, the vigorous stirring which the ore receives in the rotary furnace makes the small excess of oxygen more effective.

It may be stated, in general, that the more sulphur and carbonaceous matter an ore contains the larger is the excess of oxygen required in the Scott furnace to prevent formation of soot.

REACTION BETWEEN SULPHUR DIOXIDE AND MERCURY

Experiments in the laboratories have shown that, in the presence of moisture, sulphur dioxide and mercury vapor may react to form mercuric sulphide; but it seems doubtful whether this reaction takes place to any appreciable extent in the furnace.

SOURCES OF SOOT.

Sulphates of mercury may be formed in the furnace by the reaction of SO_3 or H_2SO_4 vapor with mercury vapor. As mentioned on page 9, sulphates may also result from the direct oxidation of mercuric sulphide at comparatively low temperature. It is not possible to state definitely just what furnace conditions favor the formation of sulphates. In the upper part of certain furnaces a heavy incrustation of sulphates of mercury has been observed. A regulation of draft with the rapid removal of gas from the furnace apparently minimizes this action.

Carbonaceous matter may be derived from hydrocarbons in the ore or from the incomplete combustion of fuel. The soot samples C and D, in Table 5, were obtained from a Scott furnace in which pitch pine was used as fuel, and the carbon shown by the analyses was derived largely from this source, although the ore itself contained a small amount of carbonaceous matter. Average samples of gas from this furnace, collected over a period of several hours, showed from 9 to 10 per cent of oxygen in the gas leaving the furnace; but just after wood was put into the fire box there was an interval during which the furnace atmosphere contained carbonaceous soot and was presumably deficient in oxygen. During these periods mercuric sulphide was probably sublimed, and it is doubtful whether, with the ore in question, the excess of oxygen present in the furnace atmosphere (corresponding to about 90 per cent excess air) was enough to oxidize mercuric sulphide vapor completely in the intervals between firing. This ore contained about 3 per cent of pyrite, sulphates corresponding to 0.9 per cent SO_3, and carbonaceous matter corresponding to 0.6 per cent carbon.

W. B. Dennis,[48] of the Black Butte Quicksilver Co., Black Butte, Ore., who made some interesting experiments on the application of wood producer-gas to a furnace of a special design, reports[49] that, even when treating carbonaceous soot alone in his experimental furnace, he was able entirely to avoid the appearance of soot in the condenser by passing the gas stream from the furnace through a special chamber kept above 900° C. by burning producer gas. Additional air was supplied as required to secure the complete combustion of soot particles carried from the main furnace by the gas stream.

CONCLUSIONS.

In discussing the soot problem Dennis[50] calls attention to the fact that the amount of soot produced depends upon the character of the ore; also that for a given plant the quantity of soot produced under a given set of conditions is fairly constant, so that the proportion of free-running mercury obtained from the condenser system will, within limits, vary with the mercury tenor of the ore.

The conclusion to be drawn from a consideration of the soot problem is that complete combustion of the fuel and the maintenance at all times of a reasonable excess of oxygen in the furnace atmosphere will reduce the quantity of soot to a minimum. The necessity of maintaining an excess of oxygen, corresponding in some cases to the admission of 100 per cent or more of excess air, obviously reduces the thermal efficiency of the Scott furnace. Therefore the cost of additional fuel must be balanced against the disadvantages of collecting and treating soot.

SUMMARY.

As previously stated, the Scott furnace has been responsible for a large part of the quicksilver production of the United States and will no doubt be extensively used in the future. Mechanical furnaces, which are discussed on page 83, have been largely used in the

[48] Dennis, W. B. Work cited, p. 112.
[49] Private communication to the authors.
[50] Dennis, C. G., Modern quicksilver reduction: Min. and Sci. Press, vol. 99, 1909, p. 761.

last few years. The particular field of the Scott furnace in the quicksilver metallurgy of the future is an interesting question. Its advantages as compared with other types of furnace are low cost of maintenance and the fact that it requires no power, and having practically no moving parts it needs little mechanical attention. Furthermore, units with capacities as low as 10 to 20 tons in 24 hours can be run at low cost and with good fuel efficiency.

These considerations indicate that the Scott furnace is particularly well adapted to meet the conditions at the smaller mines, having an output, let us say, of less than 100 tons of ore a day, where power is expensive or difficult to obtain, and where machine-shop facilities and skilled mechanics required in connection with repair work are not easily available.

The Scott furnace has not shown itself well adapted to the treatment of ore containing a large amount of sulphur either in the form of pyrite or elemental sulphur; such ores produce an excess of mercurial soot. The treatment of these ores is one of the problems in quicksilver metallurgy requiring further study. Fortunately, most quicksilver ores in the United States do not belong to this class.

EFFICIENCY OF THE FURNACE.

Conclusions as to the recovery that can be obtained with the Scott furnace can only be arrived at indirectly because of difficulty, already noted, of sampling quicksilver ores accurately enough to make an accurate metallurgical balance. The examination of many samples of roasted ore from a Scott furnace shows that complete extraction of the quicksilver can be easily obtained. The loss of quicksilver by leakage of gas from the furnace can be readily avoided by taking various precautions, noted on page 55. The loss of mercury through absorption by the brickwork of the furnace has received some attention from quicksilver operators. Although this does take place to some extent, the writers' observation has led to the conclusion that with a properly constructed and operated furnace the amount so absorbed is small, and that no continuous loss of quicksilver can be traced to this source. There appears to be no inherent difficulty to prevent the delivery to the condenser of 95 to 100 per cent of the mercury contained in the charge fed to the Scott furnace, and it is believed that this has been and is being done at many of the Scott-furnace plants in this country.

SCOTT FURNACES USED.

As a matter of general interest, a fairly complete list of the various Scott furnaces now standing is given in Table 6.

TABLE 6.—*List of Scott quicksilver furnaces.*

Mine.	County.	State.	Number of furnaces.	Size, tile shaft
United States:				
Abbot	Lake	California	1	4-4
Aetna	Napa	do	1	5-4
Alpine	San Benito	do	1	4-2
Big Bend	Brewster	Texas	1	4-4
Black Butte	Lane	Oregon	1	4-4
Cambria	San Luis Obispo	California	1	4-4
Chisos	Brewster	Texas	1	4-2
Colquitt Tignor	do	do	1	2-2
Corona	Napa	California	1	4-4
Culver Baer	Sonoma	do	1	4-2
Dallas	Brewster	Texas	1	4-2
Great Eastern	Sonoma	California	2	4-2
Helen	Lake	do	1	4-4
Kings	King	do	1	4-2
Klau	San Luis Obispo	do	1	5-4
Mariposa	Brewster	Texas	2	2-2
Mariscal	do	do	1	4-4
Mercury	Nye	Nevada	1	4-2
Mirabel	Lake	California	1	4-4
Nevada Cinnabar	Nye	Nevada	1	4-4
New Almaden (Senator)	Santa Clara	California	1	5-4
New Idria	San Benito	do	1	5-4
New Mercy	Fresno	do	1	4-2
Oat Hill	Napa	do	1	3-4
Oceanic	San Luis Obispo	do	2	4-4
Phoenix	Stanislaus	do	1	4-4
Socrates	Sonoma	do	1	3-4
St John's	Solano	do	1	4-2
Terlingua	Brewster	Texas	1	4-4
War Eagle	Jackson	Oregon	1	4-2
Wide Awake	Colusa	California	1	3-4
Mexico:				
Huitzuco		Guerrero	1	4-4

This list contains 35 furnaces, with a total capacity of 1,310 tons a day. It is of interest to note that the Scott furnace at the Kings mine, King County, Calif., has been used for calcining magnesite; one is used for this purpose at present by the Western Magnesite Development Co., near Livermore, Alameda County, Calif.

THE CERMAK-SPIREK FURNACE.

INTRODUCTION.

The Cermak-Spirek furnace is used in Europe for treating fine ore, very much as the Scott furnace is used in the United States. A review of the history of this furnace by Harpf[51] and Spirek[52] shows that the idea underlying the design of the Cermak-Spirek

[51] Harpf, A., Der Idrianer Schüttofen und seine Verwendung zur Verhüttung von Quecksilbererzen: Ztschr. angew. Chem., Jahrg. 17, 1904, p. 1420; Der Idrianer Schüttofen: Ztschr. angew. Chem., Jahrg. 18, 1905, p. 1017.

[52] Spirek, Vinzenz, Der Schüttröstofen Cermak-Spirek, seine Entstehung und Verbreitung: Ztschr. angew. Chem., Jahrg. 18, 1905, p. 22; The quicksilver industry of Italy, Mineral Industry, vol. 6, 1897, p. 568.

furnace was suggested by the Scott furnace. The first Scott furnace was built at New Almaden, Calif., in 1875, and it was not until 11 years later that Cermak[53] introduced the first fine-ore furnace at Idria, Austria. Modifications in the design of the original furnace were made by Spirek, so that this type is now generally known as the Cermak-Spirek furnace.

DESIGN AND CONSTRUCTION.

A photograph of a Cermak-Spirek furnace under construction at Idria, reproduced from Sterner-Rainer's[54] article, is shown in Plate VIII, *B* (p. 53). A full description of one of the earlier forms of furnace is given by Spirek[55] in "Mineral Industry," and descriptions of more recently built furnaces are given by Harpf[56] and Oschatz.[57] Drawings and data in the above articles have been used in preparing Plate XI. This represents a furnace for the use of wood fuel and with a capacity of 20 to 30 metric tons, depending upon the quicksilver content and other characteristics of the ore. The Cermak-Spirek furnace has been built in various sizes, ranging in capacity from 6 to 140 tons in 24 hours, and for the use of various fuels, including wood, coal, and producer gas. Plate VII, *C* (p. 52) shows the latest Spirek furnace, with Cermak condensers, at Abbodia San Salvatore, Italy.

As will be seen from Plate XI, the furnace is rectangular in cross section, and the vertical dimension is proportionately much smaller than that of the Scott furnace. It is built upon a sheet-iron bedplate *a*, which rests upon a system of **I** beams, supported in turn by masonry columns, as shown in the transverse and longitudinal sections *A* and *B*. The sides and ends of the furnace are completely covered by cast-iron plates. The furnace is divided in the direction of its greater dimension into two main vertical sections by a central wall containing the combustion chamber and various flues, as will be seen by reference to the section *A*. In the vertical direction three distinct zones may be recognized, namely, from the top down, (1) the drying zone, which comprises the upper quarter of the furnace; (2) the roasting zone; and (3) the chamber where the roasted ore is cooled and air for combustion is preheated.

The horizontal section of Plate XI shows C_1, the charging screen on top of the furnace; C_3, a section through the roasting zone; C_4, a section through the cooling chamber; and C_2, the arrangement of the hoppers at the bottom of the furnace.

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[53] Schnabel, Karl, Handbook of metallurgy (translation by Henry Louis). New York, vol. 2, 1907, p. 388.

[54] Sterner-Rainer, R., Der derzeitige Stand des Quecksilberhüttenwesens in Europa: Oest. Ztschr. Berg.-Hüt., Jahrg. 62, 1914, pp. 529–563.

[55] Spirek, Vinzenz, Work cited, p. 568.

[56] Harpf, A., Work cited, p. 142.

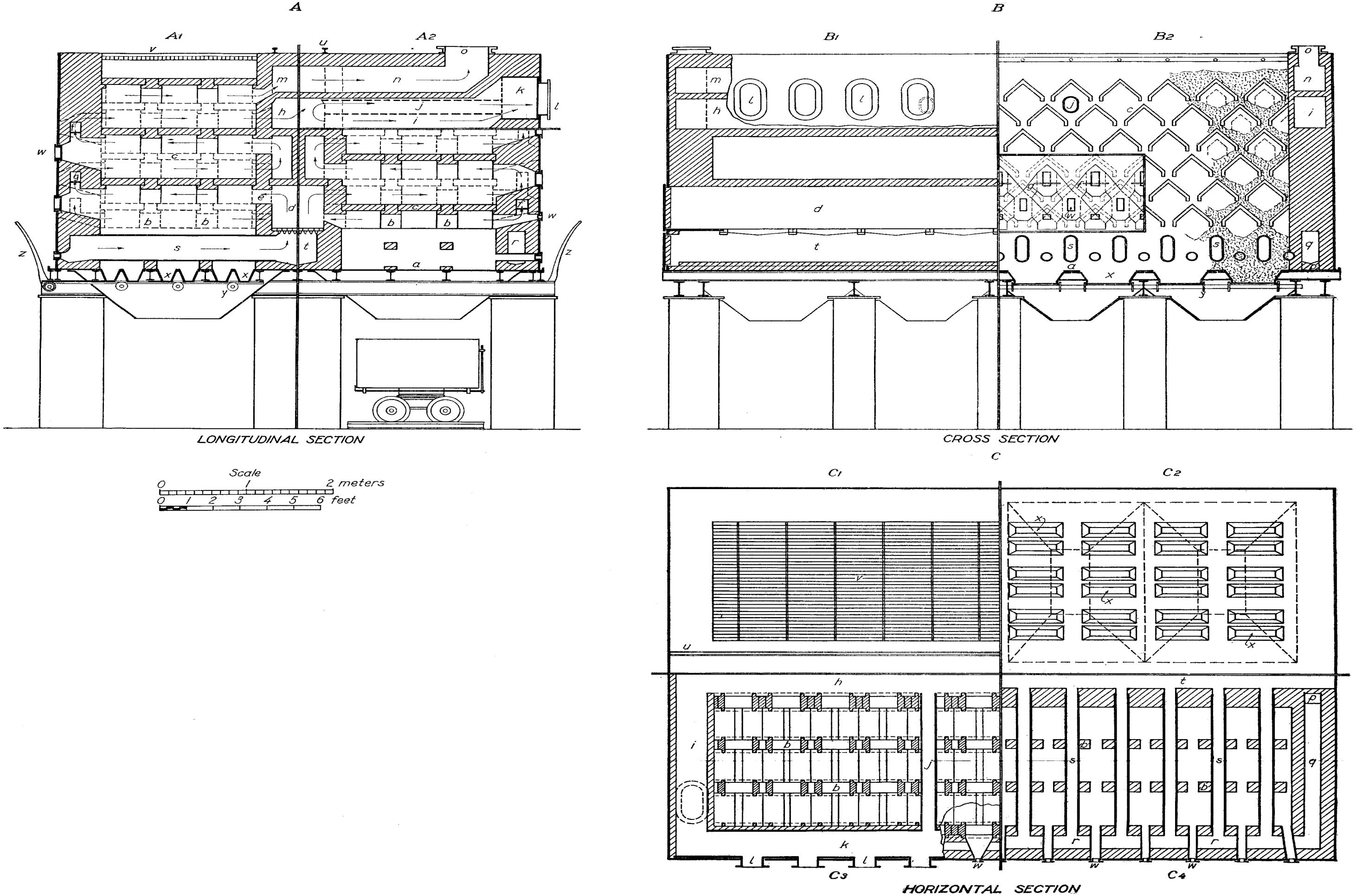
[57] Oschatz, K., Die Verhüttung der Zinnobererze am Monte Amiata: Glückauf, Jahrg. 54, 1918, p. 548.

FLUE AND GABLE TILE.

In the interior construction of the furnace, which is rather complicated, a number of special shapes of fire-clay brick and tile are used, the most important among these being the so-called "gable tile," which have the shape of an inverted V, with the lower edges extended vertically by way of reinforcement. The arrangement of these gable tile *c*, which serve to support the ore and form passages for the heating gases, is shown in the section B_2. These tile are supported by the main outer and inner walls of each of the two main compartments, and also by two secondary longitudinal walls *b* (section A_1) in each of these compartments. A row of three tile is required to span each compartment. The inner walls *b* are perforated by openings under the gables so that a continuous flue is formed under each row of tile. These flues are shown in section *A*, and section *B*.

The fire box *d*, which is for burning wood, extends longitudinally from end to end of the furnace. The hot gases from the combustion chamber pass through the opening *e* (section A_1) into the passages under the next to the bottom row of gables. After traversing these passages, the gases enter the outer wall and rise in Y-shaped flues *f*, to the passage under the next tier of gables. Each passage under the fourth tier of gables (counting from the top) thus receives half of the gas from the passages under the two adjoining gables of the fifth tier. This same plan is carried out throughout the entire roasting zone. The flow of gas in these Y-shaped flues is regulated by small tile plates *g* (see sections A_1 and B_2), which are adjusted to insure uniform distribution of the gas when the furnace is started. From the passages under the fourth tier of tile, the gases pass to flues in the central wall above the fire box, and thence make another circuit under the third and second tier of gables. The mercury-laden gases from this top tier of passages in the roasting zone are collected in the chamber *h*, which traverses the entire length of the central wall from end to end of the furnace. From this chamber the gases pass through two flues *i* in the end wall, and also through two centrally placed cast-iron pipes *j* to the cast-iron header *k*, to which 8 exit pipes, *l*, leading to the condenser, are connected.

The first tier of gables constitutes what is known as the drying zone. The passages under these gables connect with the chamber *m*, which also runs the full length of the central wall. The water vapor and any mercury vapor which is collected in this chamber are conducted by the flues *n* in the end walls to the exit pipes *o*, which connect with two separate short strings of condenser pipes. It will



LONGITUDINAL, CROSS, AND HORIZONTAL SECTIONS OF A CERMAK-SPIREK FINE-ORE FURNACE be noted that this arrangement is similar in principle to the so-called "fume trap," described on page 55, for the Scott furnace.

The sixth tier of gables forms the upper part of the so-called residue chamber, in which the roasted ore is cooled. The passages under this tier of gables connect both with the fire box and with the flue system in the outer walls, so that any mercury vapor escaping from the roasted ore at this level can pass by either of these connections to the upper part of the furnace.

The cool air for combustion enters through the vertical flue *p* (see sections B_2 and C_4 of Pl. XI), and passes through the flue *q* in the end wall and the flue *r* in the side walls to the elliptical cast-iron pipes *s*, in which it is preheated. The hot air is discharged from this pipe into the central chamber *t*, which extends the entire length of the furnace just under the fire box.

CHARGING AND DRAWING ORE.

The ore is trammed to the furnace on the track *u* and is dumped directly on the charging screen *v*, which covers the space over the gables and retains oversize ore. Owing to the fact that the ore is spread out over a large area on the top of the furnace, it is dried to a certain extent, no doubt, at this point.

The furnace operation at various levels can be observed through the peepholes *w*.

The spent ore leaves the furnace through a series of small hoppers *x*, which are shown particularly in sections A_1, B_2, and C_2 of Plate XI. These hoppers are closed by a system of gates attached to a framework which moves on the rollers *y*, and is operated by the lever *z*. The roasted ore may be discharged through the large hoppers into cars, or as at Monte Amiata, where water is available for the purpose, into large sluices.

Oschatz mentions that, at Abbadia San Salvatore, the draw pits under the furnaces are provided with special stacks 9 meters (30 feet) high for the removal of the fume and dust which arise when the water comes in contact with the piles of ore in the sluices.

CHANGES IN DESIGN.

The design of the Cermak-Spirek furnace has been modified in many different ways to meet the requirements of various kinds of ores and fuel, and to overcome various operating difficulties.

Spirek[58] has discussed the principal changes which he has made in the design of the original Cermak furnace, the most important of which are: (1) Elimination of the two auxiliary fire boxes at

[58] Spirek, Vinzenz, Der Schütttröstofen Cermak-Spirek, seine Entstehung und Verbreitung: Ztschr. angew. Chem., Jahrg. 18, 1905, p. 22.

the middle of the sides of the furnace and of the cross flues which connected them to the main combustion chamber, and substitution of a single combustion chamber fired from the two ends of the furnace as shown in Plate XI. (2) Introduction of the Y-shaped flue for leading the gases from one level of gables to another, instead of the simple large chambers previously used. (3) Replacement of the old exit pipes, which left the top of the furnace at the central point, by the system of flues and headers. (4) Introduction of a separate system of flues and condenser connections for removing the water vapor.

One of the more important variations in design is determined by the type of fuel to be used. The furnace previously described is for the use of wood, which is introduced alternately at the two ends of the furnace. For coal or lignite a semigas-producer type of fire-box was used for which small extensions were made at the two ends of the furnace. The main longitudinal combustion chamber remained unchanged, except that the grates were replaced by a perforated fire-brick bottom. Additional preheated air was introduced to the fire boxes above the grates. This practice was not successful, and was abandoned for wood firing on grates. When producer gas is used, the burners are distributed along the floor of the combustion chamber.

Sterner-Rainer [59] mentions a new furnace for producer-gas firing which has been constructed at Idria, with 10 tiers of superimposed gables instead of 6; the use of producer gas has considerably increased the capacity.

Among the minor modifications may be mentioned the use of cast-iron gables for the topmost tier.

Oschatz [60] states that, owing to their rapid corrosion, the use of cast-iron gables for the first tier is now considered to have been a mistake at Monte Amiata. The cast-iron flues leading from the central vapor chamber to the cast-iron header for the condenser connection are sometimes placed at the very top of the furnace in the charge hoppers, instead of under the upper tier of gables. Oschatz also mentions that at Monte Amiata the iron pipes for preheating the air have been replaced by two additional tiers of gables, the cold air entering the passages under the lowest tier at the center of the furnace under the ash pit. This same modification is mentioned by Sterner-Rainer, with reference to one of the new furnaces at Idria.

[59] Sterner-Rainer, Roland, Der derzeitige Stand des Quecksilberhüttenwesens in Europa: Oest. Ztschr. Berg.-Hüt., Jahrg. 62, 1914, pp. 529–563. Translation by C. N. Schuette. Production of quicksilver in Europe: Chem. and Met. Eng., vol. 19, 1918, pp. 721–727; vol. 20, 1919, pp. 32–35 and 82–84.

[60] Oschatz, K., Work cited, p. 548.

CONSTRUCTION COSTS.

According to Spirek,[61] a furnace similar to that just described requires for its construction about 45 tons of cast iron and steel. In addition to this, many special fire-clay shapes and tile are required. A rough estimate, based on Spirek's original data and on prices for labor and material prior to 1914 in the United States, indicates that the cost of construction of such a furnace in California would be between $20,000 and $25,000, exclusive of housing, condensers, and auxiliary equipment. This would make the cost of the furnace alone not far from $1,000 per ton-day capacity. Compare this cost with that for the Scott furnace, as given on page 57.

OPERATION.

The following account is taken largely from the description given by Oschatz of the practice at Abbadia San Salvatore, Italy. The large furnaces, which are approximately the same size as that shown in Plate XI, have a capacity of from 28 to 30 metric tons of ore for 24 hours, with a fuel consumption of 10 per cent by weight of the ore treated. Air-dried chestnut and beech cordwood are used. The wood is estimated to carry about 20 per cent of moisture, and to have a heating value of 3,500 calories a kilogram (6,300 B. t. u. a pound). The air entering the furnace is calculated to be 3.6 times that theoretically required for the combustion of the fuel. The two fire boxes at the end of the furnace are fired alternately, and mention is made of the poor draft and of the difficulty of insuring an even distribution of the fuel over the length of the combustion chamber.

The furnace feed is crushed to 40 mm. (1.5 inches), and the furnace is charged just after roasted ore has been drawn. In charging, the ore is dumped on the grizzly which covers the top of the furnace, and is evenly distributed by rakes. At Abbadia San Salvatore, the feed to the Cermak-Spirek furnaces consists largely of clay and marl carrying from 0.6 to 0.9 per cent of mercury. As described on page 26, the ore is dried to less than 7 per cent moisture before being charged.

INTERVAL OF DRAWING.

The roasted ore is drawn from the furnace every two hours. One man works the lever of the discharging apparatus, while another watches at a peephole to see that the hoppers at the bottom of the

[61] Spirek, Vinzenz, The quicksilver industry of Italy: Mineral Industry, vol. 6, 1897, p. 581.

so-called residue chamber are not entirely emptied. The ore in the roasting zone is then barred if necessary to induce proper movement to the bottom of the furnace, and, finally, fresh ore is charged at the top. When barring is done, the fire doors must be kept closed to prevent fumes escaping from the furnace.

The interval of drawing roasted ore and charging the furnace is much greater than in American practice with the Scott furnace, which is usually drawn and charged every half hour. The fundamental principle of moving the ore countercurrent to the gas stream is the same in each furnace, but in the Cermak-Spirek furnace the practice has been developed in the greater detail, the flow of the gas being reversed in each successive tier of flues. This has resulted in a furnace whose vertical dimension is proportionately much less than that of the Scott furnace; and it might be argued that the interval of charging and drawing should be less if anything with the Cermak-Spirek furnace, in order that the ore should remain a reasonable period of time in each zone of the furnace. The more frequent movement of the furnace charge also has a stirring action which should lead to more uniform heating of the ore stream. These considerations lead to the suggestion that the capacity and efficiency of the Cermak-Spirek furnace might be increased by more frequent charging.

When the furnace feed contains an undue proportion of fine material, slides take place occasionally at the time of drawing, which may result in the loss of unroasted ore and the choking of the Y flues with dust. Cleaning of these flues is said to be difficult, also dangerous to the health of the workmen.

The temperature in the roasting zone of the furnace is estimated to range from 300° C. at the top to 600° or 700° C. at the bottom. In the drying zone the temperature is a little above 100° C. The mercury-laden gases have a temperature of 250° C. in the central collecting chamber, and about 200° C. in the header leading to the exit pipes.

OPERATING COSTS.

The crew for a single furnace consists of three men a shift, one for charging ore and two for firing and drawing the roasted ore. A small furnace can be run by two men a shift.

The four large and two small furnaces at Abbadia San Salvatore have a combined capacity of about 144 metric tons a day, and need a crew of 48 men. This corresponds to about 3 tons of ore treated by a man a day. The cost is given as 5.7 lira ($1.10) a ton, of which

1.2 lira or 23 cents is chargeable to repairs. The economic limit for ore treated in these furnaces is placed at 0.4 per cent of mercury, and it is stated that no profit is made unless the ore contains over 0.5 per cent.

IMPERFECTIONS OF THE FURNACE.

The various accounts of the Cermak-Spirek furnaces indicate that breakage of the gable tile and other special refractory shapes causes much trouble. Oschatz states that at Abbadia San Salvatore, the interior of the larger furnaces is usually rebuilt completely every two years. A number of references are also made to the escape of mercurial fumes from the upper part of the furnace. The use of some kind of exhauster in connection with the condenser system appears to be the universal, and even then the draft through the furnace is said to be rather sluggish.

FURNACES IN USE.

A list, probably incomplete, of the various installations of the Cermak-Spirek quicksilver furnaces is given in Table 7:

TABLE 7.—*List of Cermak-Spirek fine-ore furnaces*

Mine.	Country.	Authority.	Number of furnaces.	Capacity of furnace.
				Tons.
Almaden	Spain, Province of Ciudad-Real	Sterner-Rainer	2	8
			1	4
Abbadia San Salvatore.	Italy, Monte Amiata district, Tuscany.	Oschatz	4	30
			2	18
Siele	do	Sterner-Rainer	1	18
			1	13
			1	3
Cornacchino	do	do	1	24
Morone	do	do	2	12
Petrineri	do	do	2	12
Monte Buono	do	do	1	24
Cortevecchia	do	Spirek	1	(?) 12
Idria	Austria, Province of Krain	Sterner-Rainer	3	45
			1	85
			1	140
Taghit	Algiers	Spirek	1	6
Koniah	Asia Minor, Province of Anatolia	Sterner-Rainer	1	8
Kara Bournu	Turkey, near Smyrna	do	1	8
			27	470

This furnace has not so far been introduced into the United States, although many operators have carefully considered the subject.

COMPARISON OF THE CERMAK-SPIREK AND SCOTT FURNACES.

Because they both play the same rôle in the metallurgy of quicksilver, a brief comparison of these two furnaces is of interest. The outstanding difference between the Cermak-Spirek and the Scott furnace is the complex design and construction of the former.

MATERIALS USED.

Many different special shapes of fire brick are required in the construction of the Cermak-Spirek furnace, and Oschatz[62] mentions the large amount of labor needed to trim these shapes so that true joints can be made and the gable tile properly supported. The structural weakness of the gable tile apparently causes much breakage. In building the Scott furnace only standard fire brick, tile, and red brick are used. The frequency with which repairs are needed depends upon the character of the ore and on the care with which the furnace is run. In general, it may be stated that the Scott furnace does not require repairing more frequently than at 5-year intervals, and a furnace has been known to run continuously for 10 years.

Little iron or steel is needed for the construction of the Scott furnace, as the sheathing of the furnace with metal has been found entirely unnecessary. On the other hand, the Cermak-Spirek furnace requires a large amount of cast iron and steel, including plates for covering all of the outer walls. The necessity for such covering probably arises from the fact that the walls of this furnace are thinner than those of the Scott furnace, particularly opposite the system of Y flues in the side walls. The apparent escape of mercury through the walls of the Cermak-Spirek furnace may be due in part to the condensation on the iron doors of the peepholes. Mercury so condensed would probably run down between the iron sheathing and the brickwork of the furnace, giving the appearance of having diffused through the furnace walls.

GENERAL DESIGN OF THE FURNACES.

In comparing the general design of the two furnaces the Cermak-Spirek may be conceived as being made up of two Scott furnaces placed end to end with a common fire box. The furnace has been developed horizontally rather than vertically, so that for a given capacity the Cermak-Spirek occupies considerably more area than the Scott. In the Scott furnace the inner walls, which support the

[62] Oschatz, K. Work cited, p. 548.

tile, run parallel to the flues; whereas in the Cermak-Spirek furnace the walls which support the gable tile are at right angles to these, resulting in a constriction of the gas passages under the gables at each supporting wall, with, no doubt, some interference with the draft.

The general design of the Cermak-Spirek furnace makes it necessary to collect the mercury-laden gas in a central canal from which it is led by a system of pipes and flues to the exit pipes. Judging by the various changes which have been made in the design of this part of the furnace, these connecting pipes interfere with the furnace operation, and the complexity of the whole arrangement is in marked contrast to the simplicity of the corresponding feature in the Scott furnace.

The difference in the internal structure of the two furnaces leads to a difference in the movement of the ore. The arrangement of the tile in the Scott furnace is such that the ore is turned over somewhat in flowing from one tile to the next below. This movement can be observed through the peepholes, and is of course advantageous in that it brings all parts of the ore stream into contact with the gas streams in the flues. In the Cermak-Spirek furnace there is a greater tendency for the ore to pass through the furnace in parallel streams without as much stirring action or exposure to the furnace atmosphere as in the Scott furnace.

The dissimilarity in general design of the two furnaces leads also to a marked difference in the area of the openings through which the ore is charged. A Scott furnace with four shafts is charged through two slots only 4 inches wide, running the length of the shaft. On the other hand, the Cermak-Spirek furnace is open over the greater part of the top, thus affording ample opportunity for the escape of mercury-laden gases, and also for the leakage of cold air into the furnace. The various accounts of the operation of the Cermak-Spirek furnace indicate that there is a good deal of difficulty with fuming, whereas with the Scott furnace little, if any, trouble arises from this source.

COMPARISON OF OPERATING DATA.

For purposes of comparison, certain operating data in regard to the Cermak-Spirek and Scott furnaces are presented in Table 8:

TABLE 8.—*Comparison of operation of fine-ore furnaces.*

Type of furnace.	Place.	Number of furnaces on which data is based.	Source of information.	Daily capacity.	Fuel used.	Fuel consumed compared with weight of ore.	Labor per furnace per shift.	Output per man-day.
				Metric tons.		*Per cent.*	*Men.*	*Metric tons.*
Cermak-Spirek	Abbadia San Salvatore.	4	Oschatz.	28-30	Wood	10-12	3	3
Do	do	2	do	18	do	10-12	2	3
Do	Almaden	3	Sternier-Rainer.	2-8 1-4	Coal	8.3	1	2
Do	Idria	3	Castek[a].	45.2	Wood Coal	8.87 6.6	3	5
Scott	Oat Hill	1	([b])	36	Wood	8	1	12
Do	Oceanic	2	([b])	40.5	do	3	1	13.5
Do	New Idria	[c]1	([b])	62.0	Oil	2.65	3	7
Do	Big Bend	1	([b])	40.5	Wood	7.7	$2\frac{2}{3}$	5
Do	Dallas	1	([b])	16.2	do	7.3	$1\frac{1}{3}$	4.1
Do	Chisos	1	([b])	18	do	6.5	1	6

[a] Castek, Franz, Die Bestimmung und Verminderung der Verluste beim Quicksilberhüttenwesen: Oest. Ztschr. Berg.-Hüt., Jahrg. 58, 1910, p. 231.
[b] Data collected by C. N. Schuette.
[c] At the time when the data were collected one shaft of the furnace, which was 20 years old, was obstructed by displaced tile, reducing the furnace capacity about 15 per cent.

FUEL CONSUMPTION.

A comparison of fuel consumption is difficult to make unless the character of the ore is known. For the two Scott furnaces at the Oceanic mine, for example, the ore carries some pyrite which acts as fuel, thus materially reducing the amount of wood required. At the Oat Hill and New Idria mines, however, the ore contains little pyrite or other material which would affect the thermal balance of the furnace, so that the fuel consumption at these two plants probably represents the maximum requirement of the Scott furnace with an inert ore.

The ore at Idria contains some pyrite, but at Abbadia San Salvatore there is practically none present. The latter ore, however, consists of clay and marl, and although it is dried before going to the furnace, some heat is liable to be absorbed due to the dehydration of the hydrous silicates. With these various factors in mind, the indications are, however, that the Scott furnace uses fuel somewhat more effectively than the Cermak-Spirek furnace.

EXCESS AIR.

A comparison of the available data indicates that a greater excess of air is used with the Cermak-Spirek furnace than with the Scott furnace. This is, no doubt, due in part to the strong draft which is necessary with a mechanical blower, to prevent gas leakage from the furnace, and to induce proper circulation of the gases under the gables. This larger excess of air would, of course, involve an increase in the fuel consumption. The arrangement for preheating the air for combustion has apparently been well worked out in the Cermak-Spirek furnace. It has been mentioned on page 52 that this feature of design in the Scott furnace deserves more attention than it has usually received.

ADVANTAGES OF THE SCOTT FURNACE.

This comparison leads to the conclusion that as far as quicksilver practice in the United States is concerned, the Cermak-Spirek furnace offers no advantages over the Scott furnace. The large amount of skilled labor required in making the special parts of the Cermak-Spirek furnace and in the construction of the furnace is, of course, less of an objection in Europe than it would be in this country.

In the matter of ore treated per man, the advantage is with the Scott furnace, probably due in part to the fact that it is built in larger units.

The question may be raised whether the Scott furnace would not offer some advantages in European practice.

MECHANICAL FURNACES.

Mechanical furnaces have not been used nearly as much in roasting quicksilver ores as in other branches of metallurgy. One of the reasons for this is the small tonnage of ore handled at quicksilver reduction works as compared with the tonnage treated in an average copper, lead, or zinc smelter, where much of the operating cost can be saved with mechanical furnaces. Certain requirements peculiar to the roasting of quicksilver ores developed difficult problems in adapting mechanical furnaces for this work. This circumstance, the isolation of many quicksilver deposits, and the lack of machine shop and other facilities for maintaining a mechanical furnace plant, have no doubt retarded development of mechanical furnaces for quicksilver.

As early as 1876, before the Scott furnace had been introduced, Purnell[63] suggested using a rotary furnace of the familiar cement-kiln type to treat quicksilver ore. The first recorded effort in this direction was made by H. Davie at the Socrates mine,[64] Sonoma County, Calif., in 1903. The mine yielded ore in which the quicksilver occurred largely as native mercury, and the furnace is reported to have been satisfactory. In 1911 a rotary furnace was erected at the Aurora mine[65] in San Benito County, Calif., but operation of this plant has been irregular, and offers no results of technical interest. In 1913 a rotary furnace was erected at Abbadia San Salvatore, in Tuscany, Italy. This installation, which has many features of interest, is discussed in detail below.

Dennis[66] discusses briefly the application of multiple-hearth mechanical furnaces of the familiar McDougall type to the treatment of quicksilver ores. In 1916, two of these furnaces were installed, one at the Senator mine at New Almaden, Calif., and the other at the Goldbanks mine near Winnemucca, Nev. In 1918, the first rotary kiln was erected at the New Idria mine, San Benito County, Calif.; it was followed by additional furnaces at this and at other mines.

ROTARY FURNACE AT ABBADIA SAN SALVATORE, ITALY.

Little information is available concerning the first two rotary kilns in California, which were operated for only short periods. The first installation deserving serious consideration is that at Abbadia San Salvatore, Monte Amiata district, Italy. This installation is mentioned by Sterner-Rainer,[67] and a full description of the plant and an account of its operation is given by Oschatz.[68] According to Oschatz, the main considerations which led to the installation of the first experimental kiln were a desire to reduce the labor cost in connection with ore treatment, and, particularly, the hope that the rotary-kiln plant would give less trouble from mercurial poisoning of the workman than had been experienced with the Cermak-Spirek furnaces.

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[63] Purnell, Samuel, Remarks on quicksilver: Min. and Sci. Press, vol. 3, Oct. 28, 1876, p. 289.

[64] Geary, John W., The rotary furnace for roasting quicksilver ores: Min. and Sci. Press, vol. 90, 1905, p. 22.

[65] Bradley, W. W., and others, Mines and mineral resources, Monterey, San Benito, etc., Counties: California State Min. Bur. Bull., 1917, pp. 57–58.

[66] Dennis, C. G., Quicksilver in 1914: Min. and Sci. Press, vol. 110, 1915, p. 91.

[67] Sterner-Rainer, Roland, Der derzeitige Stand des Quecksilberhüttenwesens in Europa: Oest. Ztschr. Berg-Hüt., vol. 62, 1914, pp. 529–563. Translation by C. N. Schuette. Production of quicksilver in Europe: Chem. and Met. Eng., vol. 19, 1918, pp. 721–727; vol. 20, 1919, pp. 32–35 and 82–84.

[68] Oschatz, K., Die Verhütting der Zinnobererze am Monte Amiata: Glückauf, vol. 54, 1918, p. 595.

DESCRIPTION OF FURNACE.

The construction of the first unit was begun in 1913; the installation is shown in Figure 10 and Plate IX (p. 52).

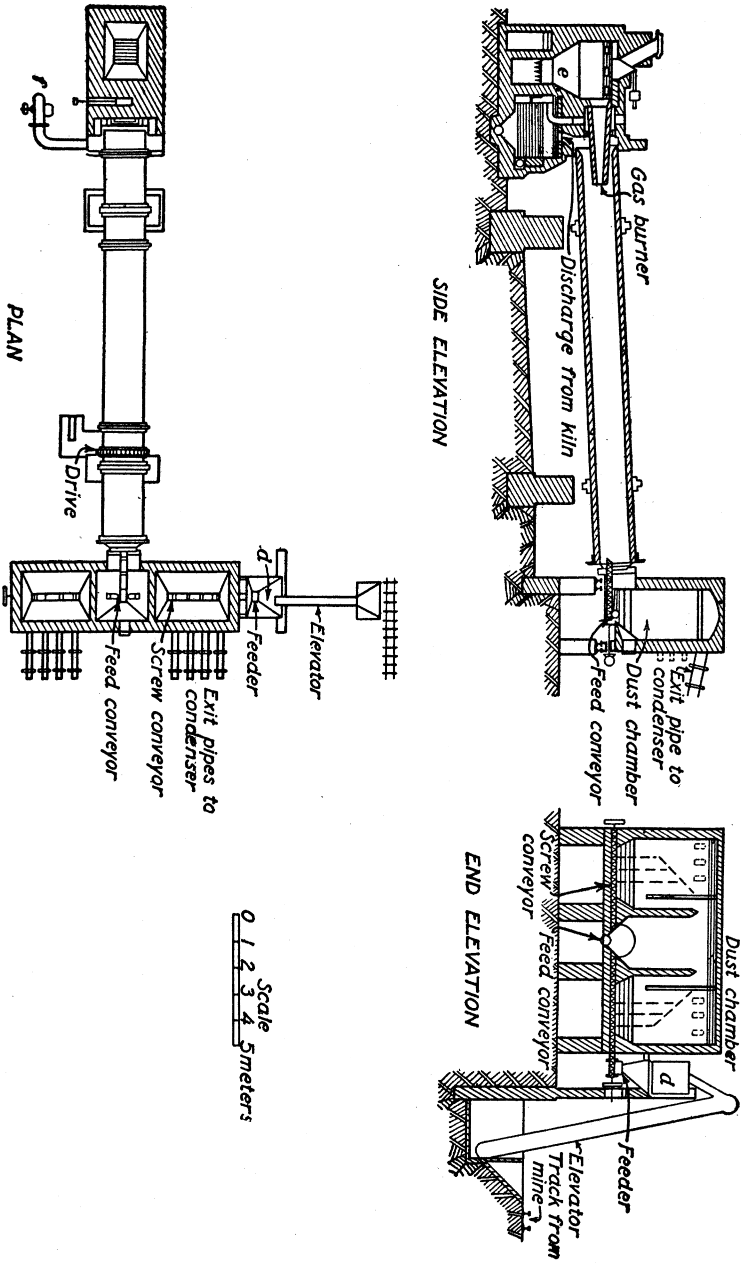


FIGURE 10.—Plan and elevations of rotary furnace at Abbadia San Salvatore, Italy. For explanation of lettering, see text.

The furnace shell is 16 meters (52.5 feet) long, and 1.65 meters (5.4 feet) outside diameter, and the original lining consisted of

200 mm. (7.9 inches) of fire brick. The kiln was set at an inclination of 1 in 20, and supported by two tires which ran on rollers, according to cement-kiln practice. The thrust was carried by two movable rollers bearing against the lower tire. The kiln was originally designed to rotate at a speed of one revolution a minute, and was driven by a train of gears actuating a "girth" gear. The furnace and auxiliary devices were run by two electric motors with a combined rating of 20 horsepower. Baffle joints were provided at two ends of the kiln, where connection was made with the fire box and dust chamber. The flanges of the sections forming the shell of the kiln were held with elongated bolt holes, so that adjustment could be made to insure proper alignment.

The dust chamber at the upper end of the kiln was divided into three compartments, as shown in Figure 10. The gas entering the middle compartment from the furnace was forced to pass over and under baffles into the side chambers, whence it passed to a set of Cermak condensers, shown at the right in Plate IX. Later, three screens, attached to iron pipes which projected through the walls, were installed in each dust chamber to provide additional baffling for the collection of the ore dust. These screens were cleaned by hammering the projecting pipes. The bottoms of the three chambers were hopper-shaped, and screw conveyers were provided to return the dust to the central chamber and thence to the kiln. The ore was also introduced by means of these conveyers.

The furnace feed was supplied to the kiln from a small bin, *d*, with a capacity of 4.8 metric tons. A vertical screw-and-plate feeder served to convey the ore into the horizontal screw conveyer in the side dust chamber, whence it passed to the central chamber and the furnace. Later, a magnet for the removal of iron and steel fragments was placed beside the plate feeder. The rotary kiln was intended primarily for the treatment of fine ore less than 5 mm. (0.2 inches) in diameter; also for the dust from the rotary driers and other fine material. The character of this material necessitated the handling of the furnace charge by screw conveyers in order to prevent the formation of dust by the free fall of the material. It was found necessary to make these conveyers strong enough to crush small particles of rock which tended to become wedged in the worm.

The roasted ore dropped from the kiln into a closed bin, whence it was removed by a screw conveyer. The air for combustion passed through this bin, being preheated thereby, and at the same time cooling the spent ore and sweeping back into the kiln any mercury vapor which might arise from it.

The method of heating the kiln is of particular interest. A gas producer, *e*, was built as an integral part of the strncture at the lower end of the kiln. This consisted of a rectangular shaft tapering at the bottom to a small square grate. Brush was used in the producer as fuel. The gas was drawn off at the top, passing through a brick-lined burner directly into the kiln. This burner, as shown in Figure 10, projected 1.4 meters (4.6 feet) into the kiln, so that part of the lower end of the kiln served for cooling the roasted ore and preheating the air. The flame was controlled by the amount of air admitted under the gas-producer grate. The air for combustion was supplied by a blower, shown at *f*. It passed through a set of iron pipes in the roasted-ore pit, and entered the space about the burner at a tangent so as to induce a spiral motion of the flame. In controlling the furnace operation, hourly records were made of the temperature at several points, also of the quantity of fuel used and ore treated.

A test made with the cold furnace showed that when running at a maximum capacity, 40 metric tons (44 short tons), the kiln contained nearly 1 metric ton of ore. The bed of ore was 0.65 meter (2.1 feet) wide, and 0.1 meter (4 inches) deep.

RESULTS OF TRIAL RUN.

The first trial run of the furnace revealed certain difficulties, as follows: The roasted ore was discharged at a temperature of 400° C., and the discharging mechanism—a screw conveyer—operated irregularly, so that the level of the roasted ore in the discharge bin could not be properly regulated. Incrustations formed in the top of the gas producer and in the burner, and so much dust was generated in the furnace that the condenser pipes and the exhauster were choked in a few weeks.

Extraction of the quicksilver from the ore was satisfactory in spite of the short roasting period of only 45 minutes, and the mechanical operation of the furnace itself gave no trouble. No evidence of mercurial poisoning of the workmen was observed. The difficulty arising from the incrustations of tar in the top of the gas producer and burner was overcome by placing a baffle wall across the top of the producer, so that the fuel gas passed directly from the combustion zone to the burner. This arrangement not only reduced the temperature in the upper part of the producer, but raised the temperature of the gas entering the burner from a range of 160° to 350° C. to one of 400° to 600° C. The difficulty experienced in discharging the roasted ore was overcome by removing the conveyer and using the bottom of the hopper as a sluice for removing the material with water.

DUST PROBLEMS.

The large quantity of dust generated in the furnace was the most serious problem. The first attempt to overcome this by installing the baffle screens in the dust chambers did not appreciably relieve the difficulty. A reduction of the speed of rotation of the furnace from 1 to 0.6 revolution a minute, with a corresponding reduction in the quantity of ore treated, gave no relief.

The next move was to replace the fire-brick lining of the kiln with a monolithic lining of crushed furnace tile and cement. This change was made because much dust had been carried to the top of the furnace between the joints of the fire box, and discharged into the upper part of the gas stream in the furnace. The change reduced the amount of dust, but not enough.

Next change was in the motion of the kiln from rotation to an oscillation of about 120°. This motion was obtained by inserting in the driving mechanism a reversing gear and two automatic friction clutches. With one oscillation a minute, the capacity of the furnace was about 30 metric tons in 24 hours. Much dust was still generated, and a further study of the problem finally revealed that most of the dust was generated by the dropping of the roasted ore into the discharge hopper, the high temperature of the roasted ore causing a strong draft.

Various devices were tried to overcome this difficulty, including the installation of a small counterbalanced door in the chute leading from the kiln to the chamber for the roasted ore, but none of these was satisfactory. Finally, it was decided to abandon the chamber for holding the roasted ore, in which were the pipes for preheating the air, and a screw conveyer was placed directly below the chute that received the roasted ore from the furnace. This arrangement prevented dusting. The loss of the two features contained in the previous arrangement, namely, the preheating of the air and the hopper for the escape of residual mercury vapor from the spent ore, was not considered serious. The screw conveyer discharged the spent ore into a sluice, whence it was removed by water.

After the successful change had been made, revolving the kiln was again tried, but difficulty with dust arose. The plan of oscillating the kiln was, therefore, permanently adopted, and the position of the kiln shifted from time to time so that wearing of the lining would be uniform.

The temperature of the central dust chamber into which the furnace gas was directly discharged varied between 180° and 360° C., but was normally between 250° and 320° C.

FUEL USED.

The brushwood used as fuel was charged into the producer in bundles, 30 to 35 inches long and 12 to 15 inches in diameter, each bundle weighing about 15 pounds. The charge hopper of the gas producer held from 2 to 4 of these bundles at one time, and from 15 to 20 were charged every hour. The fuel consumption amounted to 13.5 per cent of the ore treated, or 270 pounds a short ton. The moisture content of this brush was about 40 per cent. If the relative calorific value is taken into account, the heat required for a ton of ore in the rotary kiln was about the same as that for the Cermak-Spirek furnace where cordwood was used.

A mixture of lignite and brush was also tried. A blower had to be used to supply air to the producer owing to the greater resistance of the fuel bed to the passage of air.

OPERATING COSTS.

A foreman and two men were required each shift to run the furnace and gas producer. The furnace capacity finally averaged 30 metric tons (33 short tons) for 24 hours. The direct operating cost is given by Oschatz as follows:

Cost of operating rotary kiln at Abbadia San Salvatore.

Item.	Lira[1] per ton.	Dollars per ton.
Wages	1.10	0.21
Fuel	3.40	.66
Electric power	.90	.17
Lubricants	.10	.02
Maintenance (including condenser)	.50	.10
	6.00	1.16

[1] 1 Lira=19.3 cents U. S. normal exchange

The writers have been informed that since the above costs were calculated by Oschatz the cost of labor and materials has increased considerably.

EXPERIMENTAL KILN AT NEW IDRIA.

The first experimental rotary kiln at the New Idria mine, San Benito County, Calif., was put in operation in March, 1918. The furnace shell was 50 feet long and 4 feet in diameter, and was lined with 4-inch radial fire brick. The inclination of the kiln was 1 foot in 25 feet. The furnace was supported in the usual way by two steel tires and rollers, and was driven by a 5-horsepower electric motor through a variable speed pulley, reducing gear and girth gear.

At the feed end the furnace projected into a dust chamber. A tight joint at this point was obtained by two faced rings, one stationary on the dust chamber, and the other rotating with the kiln. This latter ring, which had an L-shaped cross section, was movable within a range of a few inches along the furnace shell and was held against the stationary ring by spiral springs. This arrangement is shown in Plate X, *B* (p. 53), and in the upper left of Plate XVII (p. 94). The joint between the fire box and the lower end of the kiln was closed in a similar way. Graphite was used as a lubricant, but by proper adjustment of the springs the pressure between the two rings could be made so light that the friction was unimportant.

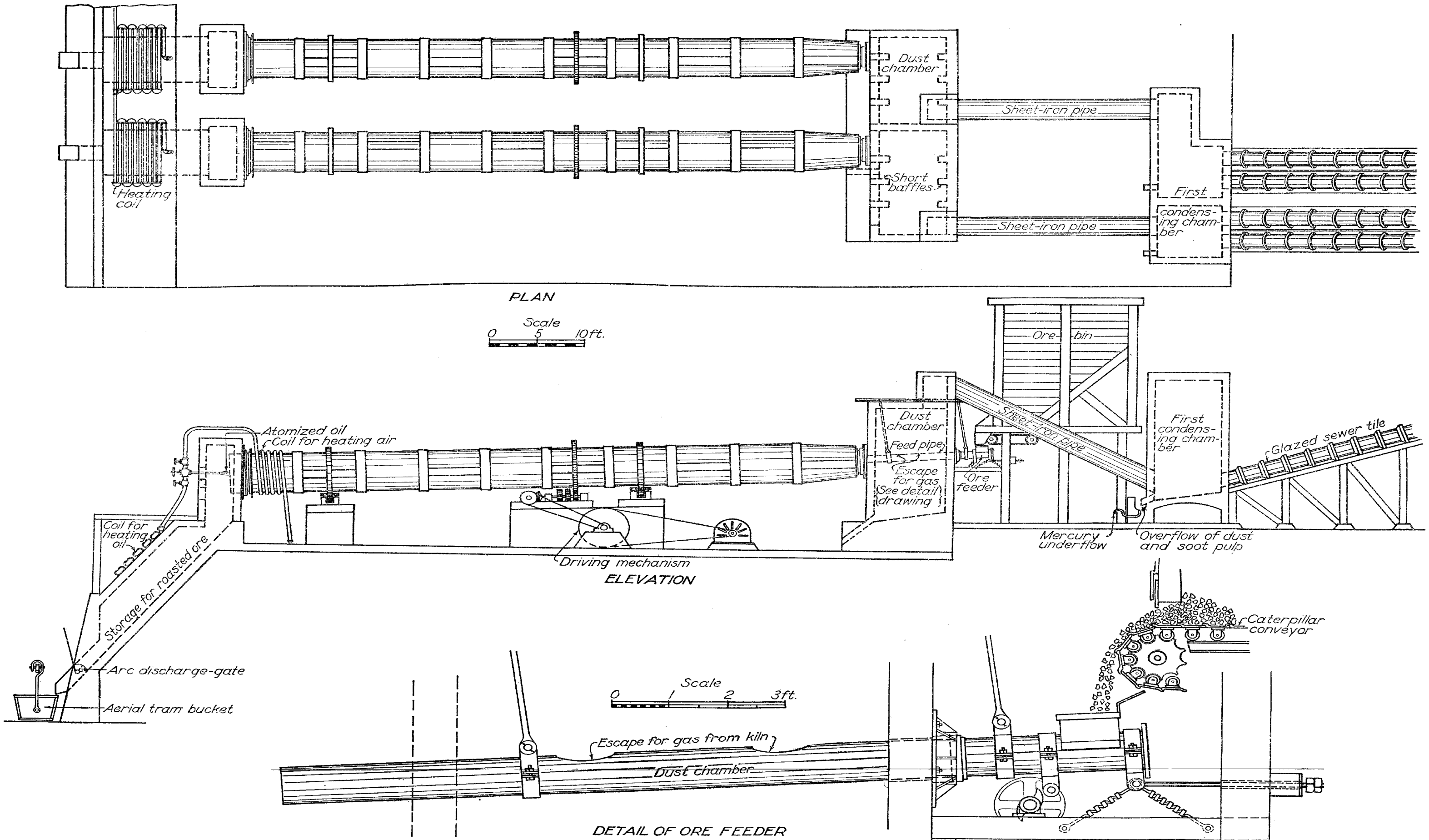
After the first dust chamber came a second small chamber, connection with which was made with a brick flue. The furnace gas escaped from this chamber at the side just below the top through a 2-foot tile pipe which leads to the tank-and-pipe condenser system, discussed on page 119. In order to remove the dust that was not settled in these two small chambers, a water spray was placed at the beginning of the tile pipe.

A fire box similar to that shown in Plate X, *A*, was used. The roasted ore, on leaving the furnace, dropped into a concrete chute under the fire box, from which it was drawn at intervals and trammed to the dump. A peephole was provided in the fire box through which the roasted ore on leaving could be observed and samples collected with a special iron scoop.

Ore was fed to the furnace through a screw conveyer which passed horizontally through the dust chamber. The hopper of this conveyer was fed in turn by a caterpillar feeder supplied from a small ore bin.

The furnace feed consisted of minus 1-inch material, and the feeding mechanism was run by a 3-horsepower motor. The kiln was normally run at a speed of one revolution per minute, and during its first four months of operation it treated an average of 57 tons of ore in 24 hours. During this period some time was lost, owing to necessary minor changes. The furnace was easily capable of treating 60 to 70 tons of ore in 24 hours.

Fuel oil of 18° B. atomized with steam was used, and the average consumption amounted to 6.85 gallons a ton of ore treated, or 2.56 per cent by weight. The gas leaving the furnace contained about 11 per cent oxygen, which, allowing for the small amount of carbon dioxide derived from the ore, corresponded to an excess of 40 per



ELEVATION AND PLAN OF ROTARY KILN AT NEW IDRIA MINE, CALIFORNIA cent of air entering the furnace. Further data in regard to the volume and composition of the furnace gas are given in the discussion of the condenser problem, pages 111 to 112.

During a 6-day run, 2 tons of dust, containing a total of 3.1 pounds of mercury, was collected in the dust chambers. The total weight of dry dust was, therefore, about 0.6 per cent of the weight of the furnace feed. As a result of frequent interruptions in operation during the first trial period, the grouting of the furnace lining became loosened, forming cracks between the bricks. These cracks filled with dust, which was carried up and discharged in the upper part of the furnace, just as happened at Monte Amiata. As the proportion of fine material in the furnace feed was relatively small at New Idria, the dust problem was not nearly so serious as at Monte Amiata.

The gas stream leaving the second dust chamber had a temperature of about 245° C., which was well above the condensing point for the mercury. The small amount of mercury which was found in the dust was, therefore, probably due entirely to particles of cinnabar coming over from the furnace.

ROTARY FURNACE PLANT AT NEW IDRIA.

DESCRIPTION OF FURNACE.

Following the successful operation of the first experimental kiln at the New Idria mine, four additional kilns of somewhat larger size were erected. This plant has been described by Bradley.[69] The plant was partly destroyed by fire in July, 1920, but was subsequently rebuilt with a number of modifications based on past experience. The four kilns were arranged in two units of two kilns each; Plate XII shows the plan and elevation of one of these units.

The furnace shells are 56 feet long and 5 feet external diameter, and are lined with 6 inches of fire brick. The last section of the kiln at the feed end is tapering, and the opening is reduced to 22 inches in diameter. This serves the double purpose of checking excessive draft through the kiln and preventing the ore from feeding backward into the dust chamber. The slope is 1 foot in 25 feet, as in the experimental kiln. Each pair of kilns is driven by a 15-horsepower motor, the driving mechanism of which is shown in Plate XIII, *A*. The normal speed is one revolution a minute. The fire boxes are stationary, the construction being similar to that of the experimental kiln. The bottom of each fire box opens into a large inclined chute formed in the concrete foundation. These chutes have a capacity of

[69] Bradley, W. W., Quicksilver resources of California: California State Min. Bur. Bull. 78, 1918, p. 248.

about 20 tons of ore, so that considerable time is allowed for the cooling of the ore, with the incidental escape of any residual mercury and the preheating of air for combustion.

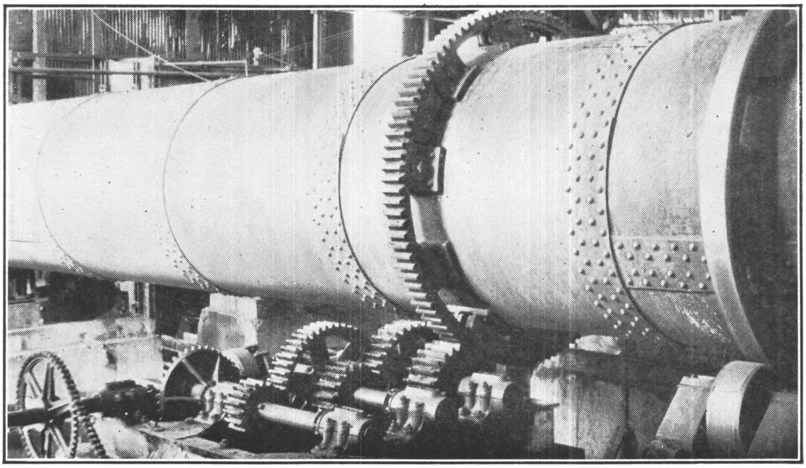
Contrary to the practice with the first kiln, the fuel oil is now atomized with compressed air at 80-pound pressure. This air was preheated to a temperature of about 70° C. in a coil of pipe which encircled the lower end of the kiln, and the oil is preheated to about 80° in a set of flat coils placed on the top of the inclined discharge chutes. The roasted ore is removed from the bottom of the chutes by a cable tramway.

CONCRETE DUST CHAMBERS.

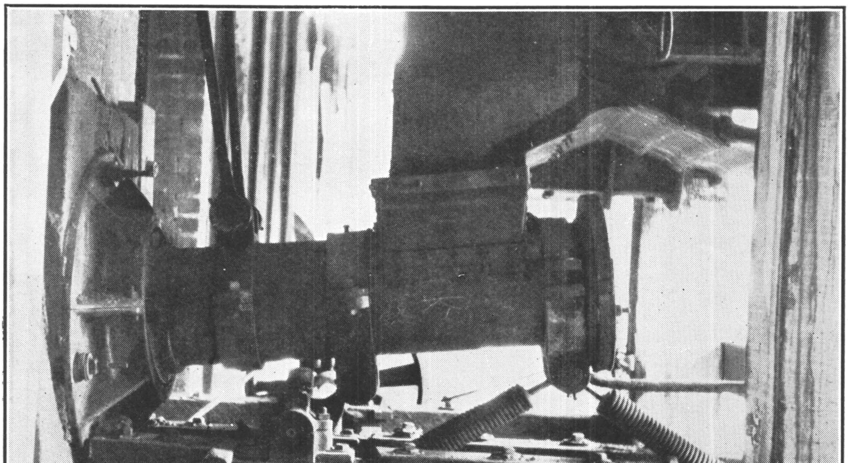
The general arrangement of the dust chambers is shown in Plate XII. Baffle walls had been built in the dust chambers, but were later taken out except for 1-foot projections from the inner walls, which were expected to set up eddy currents and thus assist the settling of the dust. A sheet-iron pipe leads from the top of the dust chamber to the bottom of what may be termed the first condensing chamber. Both the dust and condenser chambers of the newest unit were made of concrete. Concrete will probably be used for any additional kilns. Three water sprays, for the removal of dust not settled in the dust chamber, are placed in the inclined sheet-iron pipe. It will be noted that this wet dust, together with the mercury and soot from the first strings of tile pipe, all collect in the first concrete condensing chamber, passing thence to a concrete pot. A gooseneck at the bottom of this provides for the escape of the mercury, and the dust and soot carried by the water overflows to a tank. The treatment of this material is described on page 140. The joints at the two ends of the kiln are of the same construction as that used with the first experimental kiln.

ORE FEEDER.

One of the most important modifications in the new plant is in the ore-feeding device. The shaking type of feeder was developed at the Cloverdale plant, described on page 93. The design as used at the New Idria plant is shown in Plate XII. Plate XIII, *B*, shows the mechanism. The cam-shaft is operated at a speed of about 60 revolutions a minute, and the motion of ore in the pipe is induced by the sudden impulse given to the pipe by the cam and the sudden checking of the pipe's return movement by the rod and bumper. The feed is regulated by a sliding gate over the caterpillar feeder, and ore up to 3 inches in diameter can be easily handled with this equipment. The range of adjustment is from 50 to 150 tons in 24 hours.

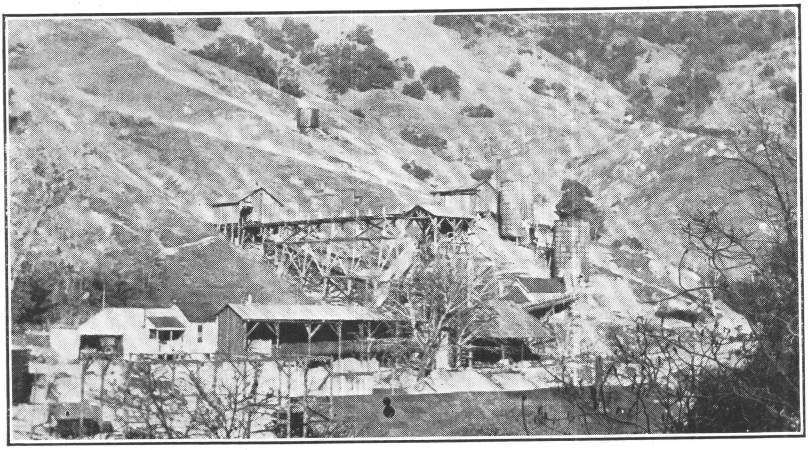


A. DRIVING MECHANISM FOR ROTARY KILN AT NEW IDRIA, CALIF.

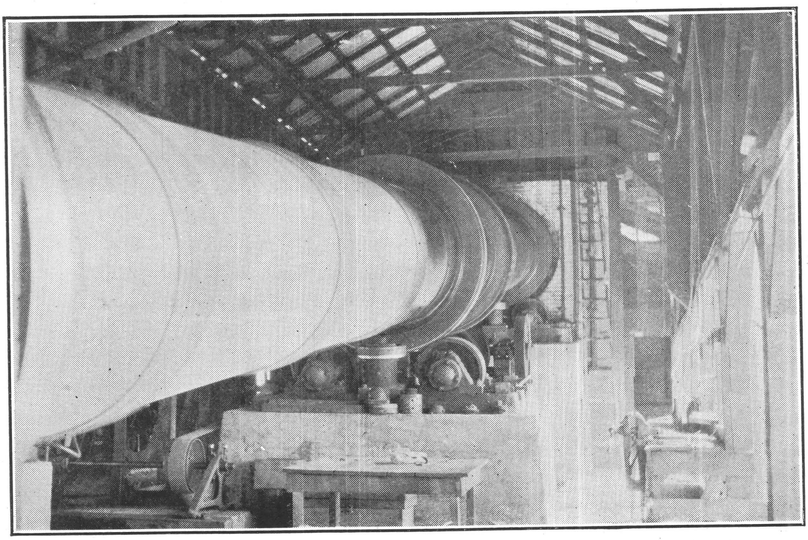


B. SHAKING FEEDER FOR ROTARY KILN AT NEW IDRIA, CALIF.

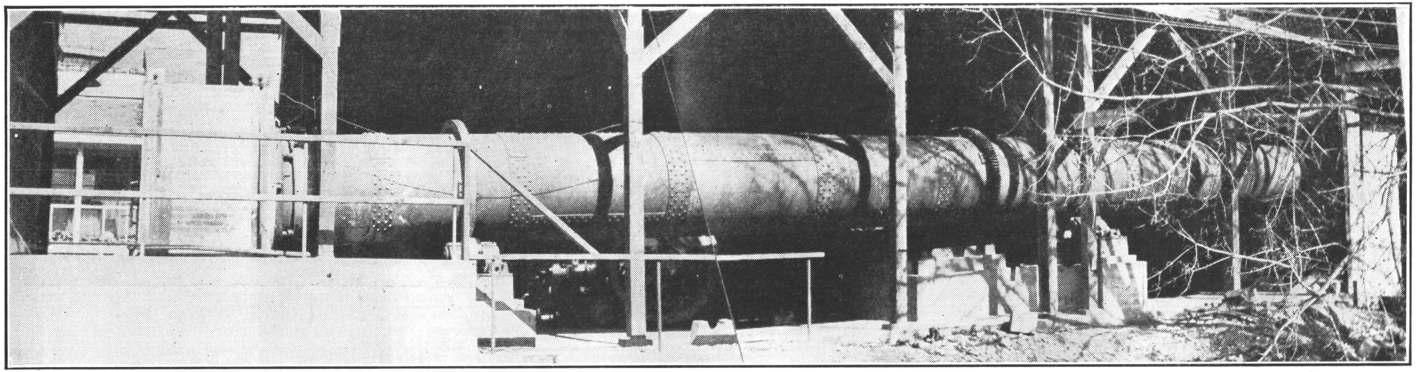
ROTARY FURNACE UNDER CONSTRUCTION AT THE NEW IDRIA MINE, CALIFORNIA, SHOWING TANK-AND-TILE CONDENSERS



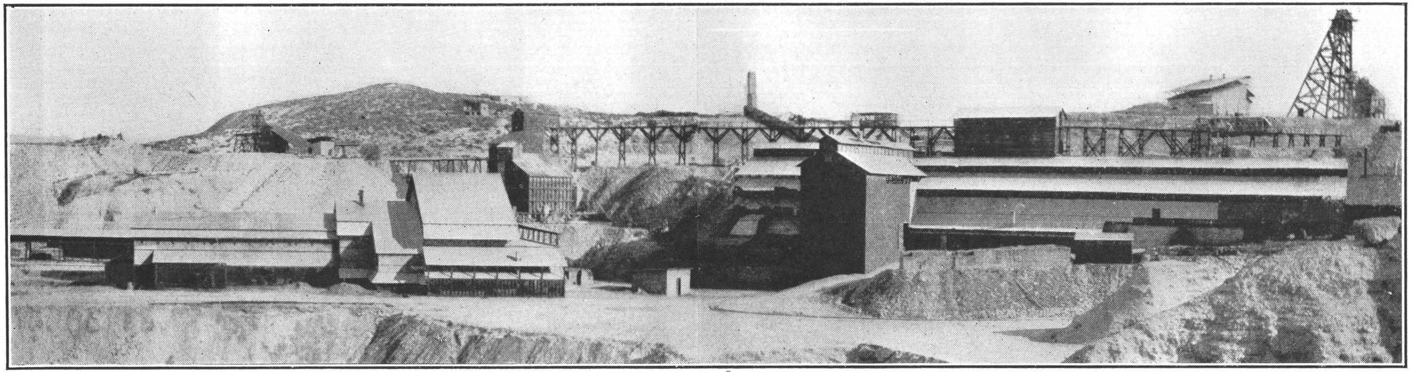
A. GENERAL VIEW OF THE ROTARY-FURNACE PLANT AT THE CLOVERDALE MINE, CALIFORNIA



B. ROTARY KILN AT THE CHISOS MINE, TEXAS, SHOWING THE HORIZONTAL THRUST ROLLERS



A. ROTARY KILN AT CLOVERDALE MINE, SOHOMA COUNTY, CALIF. (SIDE VIEW)



B. GENERAL VIEW OF THE CHISOS MINE AND REDUCTION WORKS, BREWSTER COUNTY, TEX. THE ROTARY FURNACE IS IN THE BUILDING AT THE LEFT AND THE SCOTT FURNACE IN THAT AT THE RIGHT

GENERAL DATA.

At first, some difficulty was experienced through the leakage of furnace gas back through the feeder pipe, but this was overcome by cutting two 6 by 12 inch holes in the top of the pipe, whereby any gas that entered the pipe from the furnace was drawn into the dust chamber by the draft which was there maintained.

The ore is crushed at the tramway terminal and taken to the furnace bins by belt conveyers.

Each kiln should be able to treat 100 tons of ore a day. The fuel consumption is roughly the same as that given for the first experimental kiln.

The dry dust collected in the dust chamber amounts to about 0.3 per cent by weight of the ore treated, and as it contains a negligible amount of mercury it can be rejected without further treatment.

The tank-and-pipe condenser system, such as used with the first experimental kiln, was adopted with some slight modification. Two strings of 18-inch tile, instead of a single string of 24-inch tile, were used to connect the concrete condensing chamber with the first wooden tank. A general view of the plant, including the condensers, is shown in Plate XIV. The top of the stack is 132 feet above the furnace level, and the plant operates readily under natural draft. The furnace plant, apart from the ore-crushing and distributing system, is run by two furnace men a shift; two or three extra men are employed during the day shift, making, with the foreman, a total of 9 or 10 men for the complete crew.

ROTARY KILN AT CLOVERDALE MINE, CALIFORNIA.

The rotary-kiln plant at the Cloverdale mine, Sonoma County, Calif., was built in the latter part of 1918, and put into operation in the spring of 1919. The general arrangement of the plant and certain features of construction are shown in Plate XV, *A*. A side view of the kiln is shown in Plate XVI, *A*. The kiln is 56 feet long and 4 feet in external diameter, and is lined with 5 inches of fire brick, the inclination being 1 in 25, as at the New Idria kilns. The furnace driving mechanism is, in general, similar to that used at New Idria, and the speed of rotation of the kiln is about one revolution a minute. The slip joints at the ends of the kiln are similar to those used at New Idria and are shown in detail in Plate XVII. The stationary brick fire box stands over a large concrete pit for receiving the roasted ore. An opening for the oil burner, and a small door giving access to the kiln, are in the end of the fire box; on the side ends is a peephole through which samples of roasted ore can be taken. The concrete pit is large enough to hold the roasted ore that accumulates during two shifts, so that tramming this ore to the waste dump is done only during one shift. The fuel oil and the compressed air for atomizing it are preheated in a way similar to that used at New Idria.

The concrete dust chamber is divided by baffle walls into three compartments. An inclined pipe was at first used for introducing the ore into the furnace, but this was found to be unsatisfactory, as it did not project far enough into the kiln to prevent some of the ore from rolling back into the joint and the dust chamber. The shaking feeder, shown at the left top in Plate XVII, was then designed, and worked effectively. It places the ore 6 feet within the kiln, and can be adjusted to feed at any desired rate.

A tank-and-pipe condenser system, similar to that developed at New Idria, is used, the connection between the dust chamber and first wooden tank being made by two strings of 15-inch glazed sewer tile. The furnace normally handles about 70 tons of ore in 24 hours, but on a trial run reached a capacity of 100 tons. The ore is a hard chert carrying from 10 to 15 pounds (0.5 to 0.75 per cent) of mercury a ton, and from 2 to 7 per cent moisture. The furnace feed is reduced to about 2 inches in diameter. Practically complete extraction of the quicksilver is obtained, and owing to the hardness of the ore very little dust is formed. The hardness of the ore, however, wears the furnace lining, which had to be replaced after about 20,000 tons of ore had passed through. Fuel consumption amounts to about 7 gallons of oil a ton of ore, and 7.35 cubic feet of free air at a pressure of 55 pounds is consumed a minute in atomizing the oil.

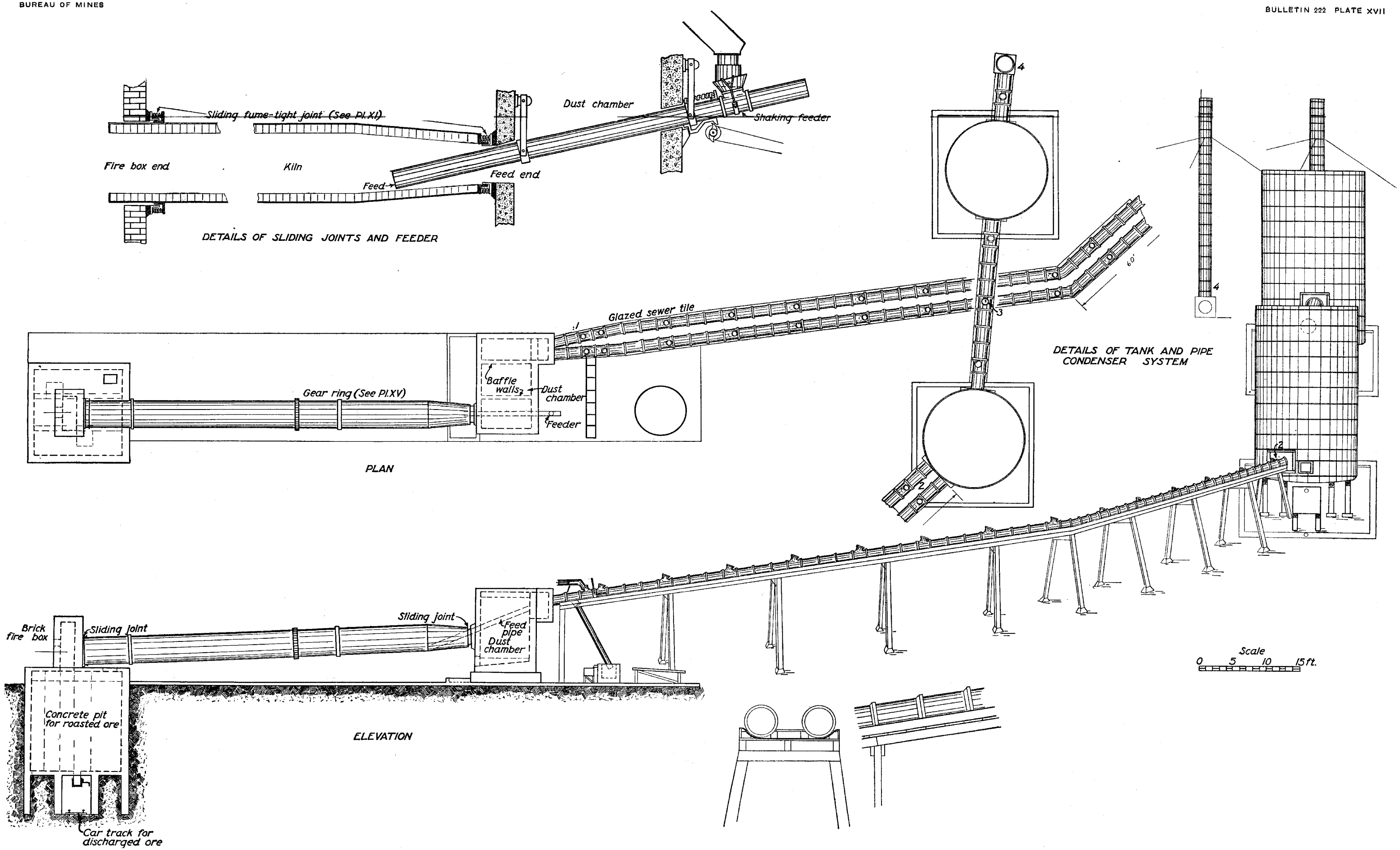
In March, 1922,[70] this furnace was adapted to the burning of wood fuel. The wood is burned in a grate fire box at the head of the kiln, and air from the compressor is blown into the kiln above the fire. This arrangement is similar to the gas producer at Monte Amiata, Italy, and has reduced the fuel cost by about one-half.

The furnace crew consists of one man a shift; an extra man during the day removes the roasted ore to the dump.

ROTARY KILN AT THE CHISOS MINE.

The furnace equipment at the Chisos mine, Terlingua district, Brewster County, Tex., includes a rotary kiln 65 feet long and 4½ feet in external diameter. A view of the kiln is shown in Plate XV, *B*. The fire-brick lining was originally 12 inches thick, thus reducing the internal diameter to 27 inches, but this was subsequently replaced by a 1-inch layer of Sil-O-Cel (infusorial earth) and 6 inches of fire brick. This increased the internal diameter to 41

[70] Private communication to the authors.



PLAN AND ELEVATION OF THE ROTARY KILN AT THE CLOVERDALE MINE, CALIFORNIA inches. The kiln is set at an inclination of 1 to 8, which is much greater than that at New Idria and Cloverdale. The kiln is supported on three steel tires and rollers, the driving gear being between the upper two.

The stationary brick fire box stands over a small pit into which the roasted ore is discharged. The arrangement of the oil burner differs from that previously described. A 4-inch pipe (see Pl. XX, *A*, p. 120) passes through the end of the fire box, and extends into the kiln for a distance of 16 feet. The oil and compressed-air pipes lie within this pipe. This arrangement of the burner converts the lower part of the kiln into a rotary cooler in which the air for combustion is preheated.

The sliding joints at the two ends of the kiln are made tight by an asbestos packing ring, and are held in place by adjustable sheet-iron strips. The joints are lubricated with oil and graphite.

At first, only a small dust chamber was used at the upper end of the kiln, but a larger one with internal baffles was found necessary to prevent an excessive amount of dust from entering the condenser system. The condenser system consists of 12 strings of Cermak condensers, of chemical stoneware, which were made in this country. The outlets from the lower return U's of the condensers extend into water-filled concrete soot boxes, the water forming a seal.

The furnace feed consists of minus 2-inch ore, from which the minus ⅛-inch material has been removed. The ore enters the kiln by gravity through a cast-iron pipe, which passes vertically downward through the dust chamber, and then turns at an angle of 45° so that the charge is delivered as far down the kiln as possible. The pipe is 4½ inches in diameter at the top, and expands to 7 inches at the kiln end. The roasted ore is withdrawn from the pit every half hour.

The furnace is rotated at one-third to two-thirds revolution a minute, and roasts about 50 tons of ore in 24 hours. From 4 to 5 gallons of 22° to 24° B. oil is consumed per ton of ore. This low consumption is due partly to the relatively good thermal insulation of the kiln, and also to the fact that the ore carries an appreciable amount of pyrite and marcasite. The roasted ore frequently contains some iron sulphide—presumably pyrrhotite, from the decomposition of the pyrite and marcasite—but elimination of the cinnabar is complete. This observation is in agreement with the results obtained in small-scale roasting experiments on pyritiferous ore mentioned on page 14, in which it was found that roasting for a brief period would expel all of the mercury and only partly decompose the iron sulphides. The furnace works normally with natural draft, and the temperature of the gas stream leaving the first dust chamber is about 200° C. Plate XVI, *B*, gives a general view of the plant.

Two men a shift attend the furnace; an additional man takes care of the machinery, including an oil engine which is used to drive the kiln.

GENERAL DISCUSSION OF ROTARY-KILN PRACTICE

USE OF ROTARY FURNACE.

The time is hardly ripe to make any positive statement as to the advance which has been made in quicksilver metallurgy through the application of the rotary kiln. Generally speaking, this type of furnace seems particularly well suited for ore treatment at the larger mines where the ore mined amounts to 100 tons or more a day, and where power, mechanical facilities, and expert attendance are readily available.

The roasting process taking place in the rotary kiln is quite different from that occurring in the Scott or Cermak-Spirek furnaces. In the latter, the ore is heated very gradually, and frequently takes 24 hours or more to pass through the furnace; whereas with the rotary kiln the total time is an hour at most, and the period within the zone of maximum temperature is measured in minutes.

The success of this radical change in roasting rests on the fact brought out in the writers' experiments with the roasting of quicksilver ores, namely, that the ordinary types of quicksilver ore part with their mercury very quickly in the temperature range of 500° to 700° C. The exception to this is found in some rather unusual types of dense, highly silicified ore, from which the mercury appears to escape with great difficulty. The rotary kiln has not as yet been successfully applied to the treatment of this type of ore, and the practicability of so doing is still an open question. Crushing to at least 6 or 8 mesh would undoubtedly be necessary, and this in itself would offer no serious difficulty, as an ore of this character would presumably yield very little dust.

In view of the short roasting period, the grade of ore fed to the rotary furnace should be fairly uniform. Any considerable change in the mercury content or other characteristics of the ore is likely to require some adjustment of the furnace operation, and uncontrolled variations in the character of the ore might lead to incomplete roasting or a waste of fuel.

WEAR OF LINING.

Mention has already been made of the abrasion of the furnace lining when a hard ore is treated. This, in a measure, is offset by the absence of trouble from dust. Owing to the moderate temperature that prevails in the upper part of the kiln, it seems possible that hard paving brick, which would be much more resistant to abrasion than fire brick, might be used for lining this part of the kiln. With a friable ore, which forms much dust, special consideration must be given to this problem.

DUST PROBLEMS.

Experience in both the United States and Italy shows that one of the sources of dust in the furnace atmosphere is the dropping of the roasted ore from the furnace to the waste bin or chute. At New Idria this is avoided by keeping the chutes for roasted ore practically filled all the time, so that there is very little drop. An extreme case of dust trouble was at Monte Amiata, where the material fed to the furnace consisted largely of dust. At this plant careful regulation of drop, the use of monolithic lining, and oscillation instead of rotation of the kiln were found necessary to reduce the difficulty to practical working limits.

In the United States the ore is fed to the rotary kilns without previous drying, and where the moisture content is not excessive this seems to offer some advantage, as it minimizes the production of dust in the upper end of the kiln. Feeding a dry and dusty ore to a rotary kiln means that a good deal of cinnabar dust will be carried back into the dust chamber. This should be avoided so that the dust caught in this chamber can be discarded without further treatment. The rotary kiln is capable of treating relatively wet ore that would give trouble in the Scott furnace.

In spite of the various precautions which may be taken to prevent the generation of dust in the rotary kiln, it seems probable that this type of furnace will always have a dust problem. The coarser dust can be settled in a baffled dust chamber without serious difficulty, but there will always be a certain proportion of fine dust having the characteristics of a metallurgical fume, which can not be readily settled. This is apparently the condition at Monte Amiata, and the installation of large dust chambers with baffles, as built there, has its own difficulty. The use of large settling chambers involves cooling the gas to a point where mercury begins to condense, the result being that the dust must be re-treated. The practice of returning the dust to the furnace, as is done at Monte Amiata, throws upon the system the continually circulating load of fine material, which must decrease capacity.

It has been mentioned that the dust collected in the first chamber of the New Idria plant contains a negligible quantity of mercury, and it is evident that if the dust can be removed from the gas stream without cooling the latter to the point where mercury begins to condense the dust cap can be at once rejected. This arrangement is obviously the most desirable.

The practice followed at New Idria, and also at the Cloverdale plant, of removing the fine dust by water sprays, is open to the objection that a certain amount of mercury in a very finely divided condition is thrown down by these sprays, and the resulting watery mud consequently requires treatment for the recovery of the mercury. This can be done by the use of settling devices, but there is always the danger of the loss of finely divided mercury in the overflowing water, and the mercurial product obtained from the settling system is in the form of a mud which is difficult to handle.

ELECTRICAL PRECIPITATION.

Probably the best way to remove dust from the gas stream is by electrostatic precipitation, with the Cottrell process. This is being used in connection with a McDougall type of furnace at New Almaden, which will be discussed on page 104. One advantage of electrostatic precipitation is that the gas can be treated at a relatively high temperature, and the dust removed at a temperature well above the condensing point of the mercury. The cost of installation and operation is not high. As is well known, the Cottrell process is being used for removal of fume and dust in a variety of metallurgical operations, and it would seem that this process would offer particular advantage where several kilns are in operation, as at New Idria.

ORE FEEDER.

The device used for feeding the ore should have a wide range of adjustment. For a fairly coarse furnace feed, the shaking feeder used at the Cloverdale and New Idria plants appears to meet all requirements. This type of feeder makes it possible to place the ore well within the kiln and also to build up, when desired, a thicker ore bed than is possible with an inclined pipe feeder. An increase in the quantity of ore fed to the furnace quickly reduces the temperature should the furnace become overheated.

The large pit for receiving the roasted ore, such as used at the Cloverdale plant, saves labor, as the waste material can all be handled during one shift. This arrangement readily adapts itself to the preheating of the air for combustion.

FUEL EFFICIENCY.

The fuel efficiency of the rotary kiln turned out to be somewhat better then was expected, and compared favorably with that of an oil-burning Scott furnace, mentioned on page 62. If, however, a Scott furnace were run with a view of obtaining good fuel economy, it is probable that the oil consumption on a tonnage basis would not be more than 66 or 75 per cent that of a rotary kiln. This difference is due partly to the necessity of discharging the gases from the rotary kiln at a temperature high enough to permit the separation of the dust without condensation of mercury and partly to the relatively large heat loss from the surface of the rotary. Direct wood firing has been tried at two small rotary plants in California, but has not worked out very well. Geary [71] refers to the successful use of wood fuel in a 5 by 50 foot rotary at the Socrates mine several years ago. The fuel consumption is given as 1.6 cords of wood in 24 hours, but the tonnage, though not stated, was evidently small. The rotary kiln is characterized by a short roasting zone and rapid heating of the ore, and thus requires relatively intense combustion. Oil in the United States, and wood producer-gas in Italy, have been successfully applied to rotary kilns, and there is no apparent reason why wood or coal producer-gas could not be successfully used in America. The use of some thermal insulation between the fire-box lining and the furnace shell would somewhat improve the thermal efficiency.

SULPHUR IN ORE.

The rotary kiln has not as yet been successfully applied to the treatment of ore high in sulphur, present either as elemental sulphur or in the form of pyrite or marcasite. The treatment of ore high in free sulphur was attempted in a small kiln at one of the California mines, wood being used as fuel. The condenser product consisted entirely of soot, but the difficulty was probably due fully as much to the type of fuel employed as to the characteristics of the rotary kiln.

The roasting experiments already mentioned have shown that the quicksilver can be completely expelled from an ore containing pyrite or marcasite in a short period of roasting, with the elimination of little more than half of the sulphur. The treatment of this latter type of ore, or an ore containing elemental sulphur, in the rotary kiln, would undoubtedly release much sulphur vapor; and the question remains as to the completeness with which this sulphur can be oxidized in the brief period during which the furnace gas stream remains at a temperature above the ignition point of sulphur vapor. This latter approximates 300° C., and unless the sulphur vapor is completely oxidized before the gas stream is chilled mercuric sulphide would be formed in the condenser.

[71] Geary, John W., The rotary furnace for roasting quicksilver ores: Min. and Sci. Press, vol. 90, 1905, p. 22.

It is possible that more favorable conditions for oxidizing the sulphur in the rotary kiln would follow introducing the fuel at the upper end of the kiln, so that the gas stream and the ore moved in the same direction. This scheme would tend to reduce the maximum temperature of the roasting zone, and at the same time extend it over a considerably greater length of kiln, owing to the heat carried forward by the ore. The thought underlying this suggestion is that the hot ore would maintain the furnace gas stream at a temperature above the ignition point of sulphur, clear to the discharge end of the kiln, and also serve as a contact surface for assisting the oxidation of the sulphur vapor. The success of this plan would depend upon so adjusting the furnace operation that all readily volatilizable sulphur would escape from the ore at some distance from the discharge end of the kiln. The thermal efficiency of a kiln so operated would not be as good as when countercurrent flow is practiced, but this should be offset by the heat derived from the oxidation of the sulphur.

POISONING HAZARD.

With respect to the possible danger of mercurial poisoning among workmen, the rotary kiln is somewhat safer than the Scott furnace, and distinctly more so than the Cermak-Spirek furnace. In the design of the first plants in the United States, a good deal of importance was attached to securing fume-tight joints (see Pls. X, *B*, p 53, and XVII, p. 94), particularly at the upper end of the kiln; but with the types of condenser systems that have been used with these kilns, a good indraft always exists at the top of the kiln, so that the principal function of the tight joints now appears to be to avoid excess inward leakage of air. No difficulty appears to have been experienced with the joints now in use.

ADVANTAGES OF ROTARY FURNACES.

Operation of the rotary kiln is very flexible. It should be adaptable to a variety of ores. For a given capacity, the length of time that the ore stays in the furnace can be regulated by the depth of the ore bed, which is in turn controlled by the feeder and the pitch and speed of rotation of the kiln. In general, the rotary kiln appears to be particularly suited to the treatment of large tonnages of low-grade ore, and its application may make profitable treatment of ore of a lower grade than has hitherto been regarded as feasible.

COMPARISON OF SCOTT AND ROTARY FURNACES.

In the light of the present state of our knowledge of quicksilver furnaces, the choice of the type of a furnace for a new plant appears to lie between the Scott and the rotary kiln. The discussion of these two types of furnace has indicated in a general way the local conditions which would be favorable to one type or the other. There will, however, be plants in which either type of furnace might be used, consequently a brief comparison of the two types of furnace with respect to economy is desirable.

CONSTRUCTION COSTS.

For this purpose, data will be used concerning two plants which were built in 1918 under practically identical conditions, so far as cost of labor and materials is concerned. It is true that these costs are high at both plants, but the figures are nevertheless of comparative value. The 40-ton Scott-furnace plant, built partly under contract and partly by the owner, cost approximately $19,300, and was completed in 90 days. The rotary-furnace plant, which was built under contract, except for minor accessories, cost $32,000, and was completed in 80 days.

CAPACITY AND OPERATING COSTS.

A number of mechanical imperfections were encountered when operation of this latter plant was started, and certain alterations were necessary before it could be successfully run. The Scott-furnace plant, using wood fuel, showed a capacity of 40 tons a day, the cost per ton-day capacity being, therefore, $482. With oil fuel, the tonnage could probably be increased to about 50 tons, corresponding to a cost of $386 per ton-day. The rotary-furnace plant treated 70 tons of ore in 24 hours with a ton-day cost of $457. The construction was in some respects less rugged than seems advisable; a plant of like dimensions, constructed of somewhat heavier, and in some cases higher grade material, would cost a little more. On the other hand, the capacity of this kiln probably could be somewhat increased.

The Scott-furnace plant included a drier, but the condensers and furnace buildings at the two plants are comparable, so that it may be concluded that the construction costs of the two types of furnace plants, including condensers and buildings, is essentially the same for unit capacity.

Certain data of operating cost are given in the table below. The ore was crushed to the same size; the same scale of wages was paid at both plants; and at both the management was efficient.

Cost of operation per ton of ore.

	Scott furnace plant.	Rotary furnace plant.
Furnace labor	$0.34	$0.23
Fuel	.25	.56
Power	.00	.05
Maintenance	.02	.06
Amortization	.13	.12
	.74	1.02

The higher labor cost at the Scott furnace plant is due to the smaller tonnage treated. The number of men employed at the two plants was the same, the skilled laborers at the rotary plant receiving the higher wage. With plants of larger capacity, the discrepancy in labor cost would become greater unless mechanical devices were used in connection with the Scott furnace.

In the matter of fuel cost the advantage is distinctly with the Scott furnace, which burned wood costing $5 a cord at the rate of 0.05 cord (one-tenth ton) for a ton of ore. The fuel cost for the rotary furnace, 8 cents a gallon in storage at the plant, is based on the use of 7 gallons of oil for a ton of ore. The use of producer gas from wood for the rotary furnace would probably lower the fuel cost, as a good supply of wood is available in the vicinity of the plant at about $5 a cord. Obviously the fuel cost depends largely upon local conditions.

The maintenance cost is based on operating experience covering about two years, and, in spite of the larger tonnage, is considerably higher for the rotary, owing to the necessity of renewing the furnace lining from time to time and replacing worn and broken parts of the mechanical equipment.

AMORTIZATION.

Amortization is based on a period of 10 years' operation for both plants, and owing to the fact that the first cost of each plant for unit capacity is practically the same, the amortization charge for a ton of ore is virtually the same. It is probable that at the end of a 10-year period the Scott furnace plant would be worth relatively more than the rotary plant (provided the mine was not worked out), but rotary kilns have not been used long enough on quicksilver ore to permit any accurate judgment on this point.

MISCELLANEOUS COSTS.

Interest, taxes, and general overhead charges have not been included. The first two would be the same for a ton for both plants, and although general overhead might be lower for the rotary plant, owing to its larger tonnage, this would be roughly offset by the cost of miscellaneous supplies which have not been included in the calculation.

SUMMARY.

For the furnaces cited the advantage is distinctly with the Scott furnace plant. When only a short period of operation is contemplated, owing to a limited supply of ore, or for other reason, the rotary plant has the advantage of a considerably higher abandonment value. On the other hand, it is only in rare instances that it pays to dismantle and move a Scott furnace—in fact, the ironwork and furnace tile, and possibly the fire brick, are the only things which will stand much of a transportation charge; but the shell of a rotary furnace, when well loaded on a motor truck and trailer, can easily be transported over reasonably good roads.

The various advantages and disadvantages of a rotary plant have been contrasted with those of the Scott furnace plant, apart from the cost of ore treatment. In general, the rotary plant possesses the greater flexibility in the matter of quantity and character of ore treated and in the ease with which the plant can be shut down and restarted with little loss of time. When a choice between the two furnaces is to be made for any given situation the various considerations mentioned must be carefully balanced, and the relative merits as indicated by the particular condition cited should not be taken as general conclusions.

Comparative data concerning the several rotary-furnace plants discussed above are given in Table 9. The information in regard to the furnace at Abbadia San Salvatore, Italy, is taken from the article by Oschatz,[72] and that for the others has been collected by the writers. The quantities of ore are expressed in metric tons.

TABLE 9.—*Comparison of rotary kilns.*

Name of mine.	Number of furnaces.	Net tonnage a day.	Fuel used.	Percentage of fuel by weight.	Men per furnace per shift.	Tons per man per day.
New Idria	1	52	Oil, 19° B	2.37- 3.16	3	6
Do	4	[a] 64	do	2.37- 2.76	0.8	25.6
Cloverdale	1	64	do	2.76	1.33	16
Chisos	1	45.5	Oil, 24° B	1.5 - 1.9	2.33	6.5
Abbadia San Salvatore	1	30	Wood producer-gas.	13.5	3	3

[a] Sixty-four tons each.

[72] Oschatz, K. Work cited, p. 595.

The following is a list of all of the rotary-furnace plants which, according to the latest information available to the writers, are in existence at the present time. Of these, only the four already discussed in detail—namely, those at the New Idria, Cloverdale, Chisos, and Abbadia San Salvatore mines—have been working for any lengthy period. The 13 American furnaces have a total capacity of 840 tons; the one European furnace handles 30 tons.

TABLE 10.—*List of rotary furnaces in the United States and Europe.*

Mine.	Situation.	Number of furnaces.	Tonnage.
New Idria	San Benito County, Calif.	5	[a] 70
Cloverdale	Sonoma County, Calif.	1	70
Rutherford	Napa County, Calif.	1	70
Aurora	San Benito County, Calif.	1	50
Stayton	do.	1	50
Sulphur Bank	Lake County, Calif.	1	30
January	Yolo County, Calif.	1	70
Rinconada	San Luis Obispo County, Calif.	1	70
Chisos	Brewster County, Tex.	1	50
Abbadia San Salvatore	Tuscany, Italy	1	30

[a] Seventy tons each.

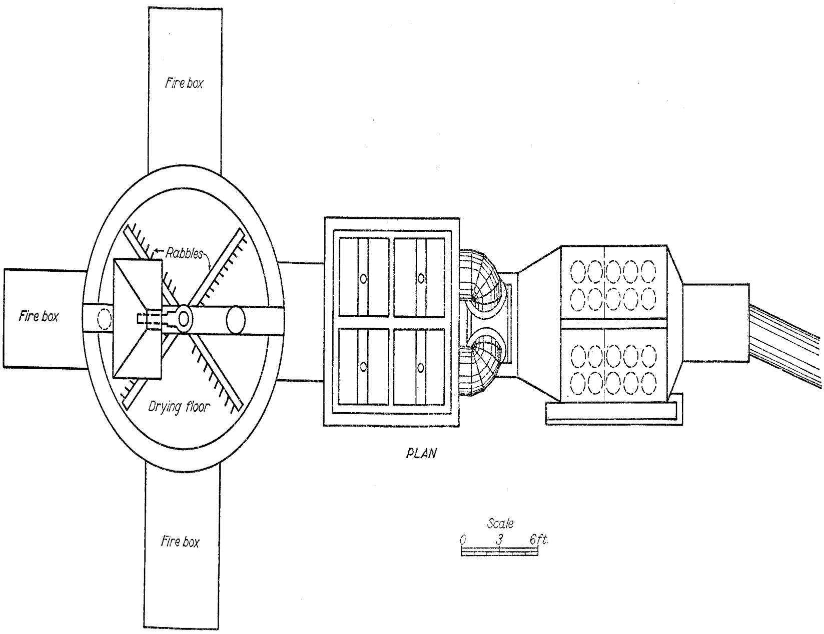
APPLICATION OF THE MULTIPLE-HEARTH ROASTING FURNACE TO QUICKSILVER ORES.

Although the multiple-hearth roasting furnace of the familiar McDougall type has been used for many years in several branches of metallurgy, not until 1916 was an attempt made to apply this furnace to the roasting of quicksilver ores. A six-hearth Herreshoff furnace, 16 feet in diameter, was installed at the Senator mine, New Almaden, Calif., and a similar one at the Goldbanks mine near Winnemucca, Nev. The original installation at New Almaden has been described by Landers.[73]

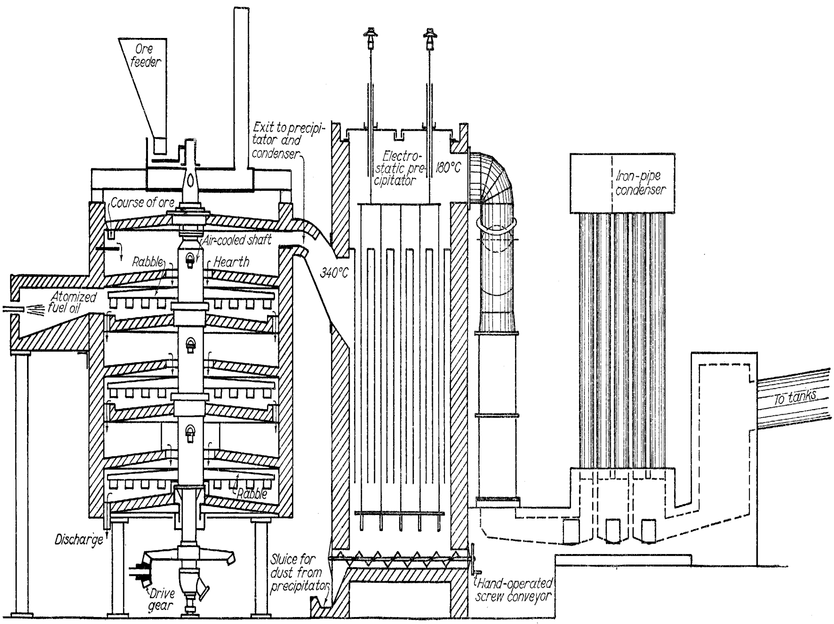
The furnace at New Almaden was used to treat ore crushed to ¾-inch diameter, also table and flotation concentrates. The excessive amount of dust carried from the furnace by the gas stream and the unsatisfactory performance of the condenser lead to certain modifications in the plant. Plate XVIII shows the general design of the plant at present; Plate XX, *B*, shows the actual installation (see p. 120).

The furnace is driven by a 10-horsepower motor, the air-cooled shaft making one revolution in about 1¼ minutes. Each hearth has two rabble arms which are also air cooled; the drying floor on the top has four rabble arms.

[73] Landers, W. H., The smelting of mercury ores: Eng. and Min. Jour., vol. 102, 1916, p. 634.



PLAN



ELEVATION

ELEVATION AND PLAN OF ELECTROSTATIC PRECIPITATOR AND IRON-PIPE CONDENSER OF THE HERRESHOFF FURNACE AT THE SENATOR MINE, CALIFORNIA

Oil, atomized with compressed air, is used as fuel; and at first some difficulty was had in obtaining good combustion and satisfactory heating of the furnace. At present two large external fire boxes are used on opposite sides of the hearth next to the bottom, and a third smaller fire box communicates with the fifth hearth from the bottom. This latter was found necessary to prevent the precipitation of mercury in the upper part of the furnace.

USE OF ELECTRIC PRECIPITATOR.

The difficulties with dust, mentioned above, were overcome by installing the Cottrell precipitator in a concrete chamber next to the furnace. This precipitator consists of twenty 12-inch pieces of standard well casing of 13 inches outer diameter, placed vertically with a wire discharge electrode suspended axially in each. The electrical equipment consists of a 15-kilowatt motor-generator set, 220–440 to 50,000-volt transformer of like capacity, and a 30-inch rectifier. (See Pl. XXI, *A*, p. 121.)

The electrical precipitation equipment was installed by the Western Precipitation Co. of Los Angeles, California. During the early period of its operation, the furnace handled daily about 30 tons of ¾-inch ore, containing a small amount of pyrite. Changes in the arrangement of the fire boxes, and experience with the operation of the furnace, increased the capacity to about 42 tons of ore in 24 hours. Fuel oil of 17° B. is atomized with 80-pound air pressure, the fuel consumption being about 12 gallons for a ton of ore, or 4.7 per cent by weight. The roasted ore drops directly from the furnace into a tramcar, which is inclosed in a sheet-iron chamber to confine the dust.

The dust, which is collected by the electrostatic precipitating apparatus, amounts to from 1,200 to 1,600 pounds in 24 hours. It is removed from the hopper below the precipitation pipes by a small hand screw conveyer, and, as it contains practically no mercury, it is at once discharged into a sluice. The power consumption of the precipitating equipment is about 5 kilowatts.

The volume of gas leaving the furnace is about 5,000 cubic feet a minute, the temperature at the inlet to the hot precipitator being about 340° C., and 180° at the outlet. From the outlet the gas passes through an iron-pipe condenser, shown in Plate XXII, *B* (p. 120), and thence to some small wooden tanks. A second electrostatic precipitator, constructed of 12-inch terra-cotta pipes, was placed at the end of the condenser system to remove any mercury still in suspension, but the exit gases from the condenser were found to be so lean that this treater is not used.

DISADVANTAGES.

The mechanical operation of the furnace has been satisfactory; only two rabble arms have been replaced in two years' operation, and the rabble teeth last for about six months. The electrostatic precipitating apparatus also gave entire satisfaction.

In comparison with the rotary kiln and other types of quicksilver furnace, the fuel consumption is high. This is contrary to what would be expected ordinarily, as the multiple-hearth furnace exposes a much larger hearth area in proportion to its surficial area than the rotary kiln, and is ordinarily regarded as giving high thermal efficiency. The large fuel consumption observed at New Almaden is probably due in part to the fact that the hot air derived from the cooling of the rabble arms is not used in any way, and also to the large area exposed by the external fire boxes and the lack of suitable thermal insulation for them. The use of the auxiliary fire box near the top of the furnace to maintain the furnace gas at a temperature high enough to allow it to pass through the hot treater without cooling to the condensation point of the mercury is also a factor in this connection.

The second feature wherein the multiple-hearth furnace does not compare favorably with the rotary kiln is in the quantity of dust produced to a ton of ore treated. This dust is, no doubt, thrown into circulation by the dropping of the ore from hearth to hearth, and the quantity might be considerably reduced by providing separate paths for the descending ore and the ascending gas stream. Aside from the imperfect utilization of heat in this particular plant, the Herreshoff furnace, in common with the rotary furnace, must use more fuel than is needed merely to roast the ore, because the dust must be separated from the gases at temperatures above the condensing temperature of mercury.

HERRESHOFF FURNACE AT GOLDBANKS MINE.

No data are available in regard to the Herreshoff furnace plant at the Goldbanks mine, but it is understood that the plant was satisfactory. The ore at this mine is dense and highly silicified, with a good deal of the cinnabar disseminated through the silica. By reference to the experimental work on the roasting of quicksilver ores, page 11, it will be seen that this ore required an unusually long roasting period in order thoroughly to eliminate the mercury. In practice, the ore was crushed to about one-fourth inch, but owing to its hardness very little dust was generated in the furnace.

The amount of information which is available in regard to the application of the Herreshoff furnace to the treatment of quicksilver ores is not sufficient to permit any definite expression of opinion as to its practicability. In the light of the experience already gained, a number of improvements could be made in adapting this type of furnace to quicksilver roasting. One of the ways in which excessive dust formation might be avoided has been mentioned, and it is possible that a larger furnace with a greater distance between the hearths on which fuel is applied would have better thermal efficiency.

PYRITIC ORES.

Reference has been made to the fact that no thoroughly satisfactory method for the treatment of quicksilver ores high in pyrite has yet been devised, but it is possible that the McDougall type of furnace offers a satisfactory solution of this problem, particularly at mines where the ore carries enough sulphur to supply the fuel for sustained roasting. In other branches of metallurgy it has been amply demonstrated that pyritic ore can be roasted in this type of furnace with the formation of gas entirely free from sulphur vapor, and there is no reason to think that a similar result can not be obtained with quicksilver ores. Even though the ore contained a high percentage of iron sulphide, it should be possible so to conduct the roasting that little, if any, mercuric sulphide would be formed in the condenser.

THE RECOVERY OF MERCURY FROM FURNACE GASES.

INTRODUCTION.

The problem of recovering quicksilver from the furnace gas stream may be considered as a separate subject; the general requirements of a condenser system are the same, regardless of the type of furnace used to roast the ore. The gas stream passing to the condenser system should be reasonably free from dust from the furnace, as the presence of dust promotes the formation of mercurial soot and complicates the problem of quicksilver recovery. The Scott furnace gives rise to little dust, but, as already noted, mechanical furnaces handling certain types of ore generate so much dust that special provision must be made for removing it from the hot furnace gas before the latter passes to the condenser system. Methods of dust removal have already been discussed, and the freeing of the gas stream from the suspended ore particles is not properly a part of the function of the condenser system. When carbonaceous soot forms in the furnace, most of it, because it is light and fluffy, is unavoidably carried into the condenser; but in general the condenser system is required only to cool the gas stream and to collect the condensed mercury.

The condenser problem has always received considerable attention from quicksilver operators. In the early days, when high-grade ores were being treated, the mercury content of the gas stream passing through the condenser was comparatively high, and the condenser problem was relatively simple, because losses that are absolutely inadmissible with the present low-grade ores could be tolerated; in fact they escaped detection.

Under the stimulus given to the quicksilver industry in the United States by the World War, the treatment of ores of somewhat lower grade than those hitherto regarded as economically valuable was undertaken. The necessity of making a high recovery from these low-grade ores directed attention anew to the condenser problem and a number of new types of condenser systems were tested. Owing to the difficulties already mentioned in obtaining an exact metallurgical balance by which the over-all recovery of the furnace and condenser system could be determined, the question of the efficiency of condensers had to be attacked in another way. In 1917 the present authors began a study of this problem and their report[74] on it was published some years ago. They found no serious losses at any of the plants examined. Since the report appeared, additional work has been done on the condenser problem, and further data on condenser losses will be given in later pages of this bulletin.

GENERAL DISCUSSION OF THE CONDENSER PROBLEM.

The condenser system receives the gas stream from the furnace at a temperature that may at times rise to 300° C., cools the gas stream to a temperature that should not in general exceed 30° or 40° C., and at the same time collects the condensed mercury. As the concentration of mercury vapor in the gas stream is often as low as 0.1 per cent by volume, and less than 1 per cent by weight, the recovery of a large percentage of the mercury seems at first glance to be a rather formidable problem. Besides the gaseous constituents of the gas stream, a certain amount of dust and soot also enters the condenser system and must be collected with the mercury. In a quicksilver condenser two fundamental operations take place—the cooling of the gas stream and the collection of the condensed mercury. The former is ordinarily accomplished by bringing the gas in contact with the cooled surface of the condenser, from which

[74] Duschak, L. H., and Schuette, C. N., Fume and other losses in condensing quicksilver from furnace gases: Tech. Paper 96, Bureau of Mines, 1919, 29 pp.; Min. and Sci. Press., vol. 117, 1918, p. 315.

the heat is removed in turn by the atmospheric air, or in some plants by water. The rate of the heat transfer depends on the thermal conductivity of the condenser material, the difference in temperature between the gas and the cooling medium, and the velocity of the gas next to the condenser surface; in other words, it is a function of both the form of the condenser system and the material of which it is constructed.

The recovery of the condensed mercury and the precipitation of mercurial soot is essentially a mechanical problem of separating suspended particles from a truly gaseous medium. Apart from the use of electrostatic precipitation, this separation is accomplished by reducing the velocity of the gas until the particles settle by gravity, or by the use of baffles to throw the particles against the walls of the condenser. The efficiency of a condenser system in collecting these suspended particles is, therefore, essentially a matter of design, and involves only in a secondary way the material used in building the condenser. Although the two processes of cooling and collecting are considered separately, in the actual operation of a condenser they are largely simultaneous.

Various designs and a number of different structural materials have been used in condenser systems. In Europe, more particularly in Italy and Austria, the accepted type of condenser system is made of stoneware pipe, the design apparently being based on the form of condenser commonly used in connection with the manufacture of nitric and other acids. The development of condenser practice in Europe has been discussed by Schnabel, Mitter, Spirek, and Castek.[75]

In the United States, the early condensers usually consisted of a series of brick chambers, and with the development of the Scott furnace this type of construction became standardized. Plate XXI, *B* (p. 121), shows a typical installation of this type. Considerable work on condenser design was done at the New Almaden mine, California, during its period of greatest production, as related by Christy.[76] During the last few years several different forms of condenser systems have been constructed, which differ from the older type chiefly in the use of wooden chambers and glazed tile instead of massive brickwork.

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[75] Schnabel, Carl, Handbook of metallurgy. Translated by Henry Louis, New York and London, vol. 2, 1907, pp. 360–392, 398–406, 409–413.

Mitter, C. [Stone condensers at Idria]: Oest. Ztschr. Berg-Hüt., Jahrg. 38, 1890, p. 333.

Spirek, Vinzenz, The quicksilver industry of Italy: Mineral Industry, vol. 6, 1898, pp 570, 576–581; vol. 10, 1902, p. 561 (drawing): vol 11, 1903, p. 550 (drawing).

Castek, Franz, Die Bestimmung und Verminderung der Verluste beim Quecksilberhüttenwesen: Oest. Ztschr. Berg-Hüt., Jahrg. 58, 1910, p. 235.

[76] Christy, S. B., Quicksilver condensation at New Almaden, Calif.: Trans. Am. Inst. Min. Eng., vol. 14, 1885, pp. 206–265.

DETAILED CONSIDERATION OF THE CONDENSER PROBLEM.

It will be well to consider here, first of all, the character and the quantity of material that the condenser system has to handle. Three examples have been selected, namely, the 70-ton oil-burning Scott furnace at the New Idria mine (now dismantled), the first rotary kiln installed at the New Idria mine, and a 40-ton, wood-burning Scott furnace at the Oat Hill mine, both in California. The essential data in regard to these three furnaces are given below:

NEW IDRIA SCOTT FURNACE.

The quantity of ore treated per 24 hours was 62 metric tons.

The furnace charge, made up a siliceous ore and some concentrate, had volatile constituents approximately as follows:

Volatile constituents of furnace charge.

	Per cent.
Mercury	1.25
Mechanical moisture	8
Combined water	4
Carbon dioxide	.5

Fuel oil, gravity 18° B., atomized with steam, was used; the consumption was 2.65 per cent by weight of furnace charge.

NEW IDRIA ROTARY FURNACE.

The quantity of ore treated per 24 hours was 49 metric tons.

The furnace charge was siliceous ore that had volatile constituents approximately as follows:

Volatile constituents of furnace charge.

	Per cent.
Mercury	0.535
Mechanical moisture	8.0
Combined water	4
Carbon dioxide	.5

Oil fuel of 18° B. gravity was used. The consumption was 2.56 per cent by weight of the ore treated.

OAT HILL SCOTT FURNACE.

This furnace treated approximately 36 metric tons of ore in 24 hours.

The ore was siliceous, essentially free from sulphides of iron or organic matter, and after passing through the drier assayed as follows:

Mercury and moisture content of ore.

	Per cent.
Mercury	0.25
Moisture	1.00

Wood (fir, manzanita, and oak) was used as fuel, The consumption corresponded to about 8 per cent by weight of the ore treated.

Table 11 shows the quantity and composition of the gas stream entering the condenser per metric ton of ore treated. The excess air was calculated from the average of a number of analyses of the gas entering the condenser. In calculating these data it was necessary to make allowance for the steam used in atomizing the oil and the moisture contained in the wood, fuel, and the air.

TABLE 11.—*Components of furnace-gas stream.*

Per metric ton of ore.

Substance.	New Idria Scott furnace.	New Idria rotary furnace.	Oat Hill Scott furnace.
	Kg.	*Kg.*	*Kg.*
Mercury vapor	12.5	5.4	2.5
Sulphur dioxide	4.0	1.7	.8
Carbon dioxide:			
From ore	5.0	5.0	0
From fuel	83.3	80.5	104.3
Water vapor:			
From ore	120.0	120.0	10.0
From air (⅓ saturated at 20° C.)	4.0	2.8	4.5
From steam for atomizing oil	13.3	8.5	0
From combustion of fuel	34.2	32.9	51.6
Nitrogen:			
From air for combustion	304.0	297.0	253.7
From excess air	274.0	119.0	413.5
Oxygen: From excess air	82.0	35.3	160.4
Total	936.3	708.1	1,001.3

VOLUMETRIC COMPOSITION OF FURNACE GASES.

For the purpose of the discussion that follows, the volumetric composition of each gas stream has been calculated, and is given in Table 12. As a matter of convenience the volume of the various constituents of the gas stream has been expressed in cubic meters at standard conditions—0° C. and 760 mm. pressure. The volumes given for the mercury and water vapor are of course hypothetical, as these substances could not exist in the vapor form at 0° C. in anything like the proportion indicated.

TABLE 12.—*Volumetric composition of furnace gases.*

Substance.	Volume per kg. (standard conditions).	New Idria Scott Furnace.				New Idria rotary furnace.				Oat Hill Scott furnace.			
		Weight.	Volume (standard conditions).		Partial pressure.	Weight.	Volume (standard conditions).		Partial pressure.	Weight.	Volume (standard conditions).		Partial pressure.
	Cu. m.	*Kg.*	*Cu. m.*	*P. ct.*	*Mm. Hg.*	*Kg.*	*Cu. m.*	*P. ct*	*Mm. Hg.*	*Kg.*	*Cu. m.*	*P. ct.*	*Mm. Hg.*
Hg	0.112	12.5	1.4	0.2	1.233	5.4	0.6	0.1	0.7	2.5	0.28	0.04	0.254
SO_2	.350	4.0	1.4	.2	1.2	1.7	.6	.1	.7	.8	.3	.04	.3
CO_2	.509	88.3	44.9	5.7	39.3	85.5	43.5	7.2	49.6	104.3	53.2	6.8	48.3
H_2O	1.245	171.5	214.0	27.4	189.1	164.2	204.3	33.6	231.8	66.1	82.4	10.5	74.6
N_2	.800	578.0	462.5	59.2	408.8	416.0	332.8	54.9	378.9	667.2	534.0	68.2	484.2
O_2	.700	82.0	57.4	7.3	50.4	35.3	24.7	4.1	28.3	160.4	112.3	14.4	102.3
Total		936.3	781.6	100.0	690.0	708.1	606.5	100.0	690.0	1001.3	782.5	100.0	710.0
Total permanent gases			565.9	72.4	499.7		401.6	66.3	457.5		699.8	89.5	635.2

It is now possible to draw certain conclusions as to the behavior of the gas stream while cooling in the condenser. Considering first the New Idria Scott furnace, the reader will note that the partial pressure of the mercury in the gas entering the condenser is 1.233 mm. By reference to column A of the chart given in Figure 9 the reader will see that the vapor pressure of mercury becomes equal to this partial pressure at about 131° C.; that is, when the gas stream has cooled to this temperature it is saturated with mercury vapor, or is at the so-called dew point, and any further temperature drop will result in the condensation of mercury. The corresponding temperatures for the gas streams from the New Idria rotary and Oat Hill Scott furnace are 118° and 98° C., respectively.

SATURATION TEMPERATURES OF CONDENSER ATMOSPHERE.

Similarly, the temperatures at which the condenser atmosphere becomes saturated with water vapor and begins to deposit liquid water, with a further lowering of temperature, are as follows:

Saturation temperatures for water vapor, °C.

	°C.
New Idria Scott furnace	61.5
New Idria rotary furnace	70
Oat Hill Scott furnace	46

The lower saturation temperature in the Oat Hill furnace is largely due to the ore being dried before it is charged to the furnace. A comparison of the temperatures at which mercury vapor and water vapor begin to condense is of interest in showing the possibility of obtaining dry soot and mercury from a part of the condenser system.

provided it is sectionalized in such way that the liquid condensing at different points can be drawn off separately.

The table above also furnishes a basis for calculating the proportion of the total quicksilver that may be condensed by cooling the gas stream to any given temperature. Take the work of the New Idria Scott furnace as an example and assume that the temperature of the gas leaving the condenser system is 40° C. It is then necessary to compute the volume of gas leaving the condenser per metric ton of ore charged to the furnace. If leakage of air into the condenser is disregarded, this volume of permanent gases, saturated with water vapor at the temperature in question, may be calculated as follows:

Vapor pressure of water at 40° C	mm	55
Partial pressure of dry gas at prevailing barometric pressure of 690 mm.=690—55	mm	635
Volume of permanent gases per metric ton ore at standard conditions	cu. m	566
Volume of permanent gases at 0° C. and partial pressure of 635 mm.= $566\times\frac{760}{635}$	cu. m	678

The actual volume of gas saturated with water vapor leaving the condenser system at 40° C. may be calculated from this last value by making the usual correction for a change of temperature from zero to 40° C.; but this calculation is not necessary for the present purpose. By reference to column B of the chart in Figure 9, the reader will see that the quantity of mercury required to saturate 1 cu. m. of gas measured at 0° C. and heated under constant pressure to the temperature in question—40° C.—is 0.00007 kg. The mercury carried away in vapor form by the above volume of gas is therefore

$$0.00007\times678=0.047 \text{ kg.}$$

The total quantity of mercury originally associated with the above quantity of gas was 12.5 kg., the theoretical vapor loss amounting to about 0.4 per cent. This means that if the gas stream is cooled to 40° C. (neglecting condenser leakage), and if the condenser system is so designed as to collect effectively the condensed mercury, the possible theoretical recovery is 99.6 per cent. A similar calculation for the New Idria rotary furnace indicates a possible theoretical recovery of 99.3 per cent at a condenser discharge temperature of 40° C.

THEORETICAL MERCURY-VAPOR LOSS.

For the Oat Hill Scott furnace the corresponding theoretical vapor loss at 40° C. is 2.3 per cent, or considerably more than for the other two furnaces, owing to a slightly larger gas volume per metric ton of ore and particularly to the smaller content of mercury in the furnace charge. The vapor loss calculated for a condenser discharge temperature of about 15° C. amounts to only 0.3 per cent and indicates a possible recovery as good as that for the other two furnaces. These calculations show the need for more effective cooling when very low-grade ore is treated. In actual practice, as will be pointed out later, the mercury loss from the condenser stack is likely to be somewhat higher, because of the leakage of air into the condenser system and the failure of the condenser system to precipitate completely all of the mercury that condenses in the form of mist.

In the design of condenser systems little attempt seems to have been made so far to base design on the actual work of cooling that the condenser is expected to perform. As already pointed out, the condenser has two main functions—to cool the gas stream and to collect the condensed mercury. The latter is essentially a problem of separating from the gas stream the droplets of mercury it mechanically carries, and falls under the head of the general problem of separating suspended particles from a gas stream. Its solution in general involves mechanical or electrical details, and will be discussed in detail later.

DISSIPATION OF HEAT.

The dissipation of heat from the gas stream involves the transfer of heat through the walls of the condenser to the cooling medium, which may be either air or water, and the efficacy of a condenser in cooling the gas depends both on the thermal conductivity and structural arrangement of the condenser material.

The heat to be dissipated from the gas stream comes from two sources—the sensible heat of the permanent gases, including the water vapor that leaves the condenser as such, and the latent heat of liquefaction of that part of the water vapor which is liquefied in the condenser. The latent heat of condensation of the mercury vapor is in general such a small proportion of the total that for practical purposes it can be neglected.

The gas stream from the Scott furnace at New Idria, mentioned above, may be taken as an example. It is necessary first to determine what proportion of the total amount of water vapor present in the gas stream will condense. It has been shown above that, on the assumption that the gas stream leaves the condenser at 40° C., the partial pressure of the permanent gases will be 635 mm., and at the prevailing pressure and 0° C. the volume of permanent gases arising from one metric ton of ore will occupy a volume of 678 cu. m. The water vapor required to saturate this volume at 40° C. is as follows:

$$678\times\frac{55}{760}\times\frac{18}{22.4}=39.5 \text{ kg.}$$

The quantity of water condensed to liquid is therefore

$$171.5-39.5=132.0 \text{ kg.}$$

The calculation of the total heat to be dissipated, according to the above assumptions, is given in the Table 13, when thermal quantities are in kilogram-calories.

TABLE 13.—*Heat to be dissipated in cooling the gas stream from 200° to 40° C. per metric ton of ore.*

New Idria Scott furnace.

Substance.	Quantity.	Specific heat.	Heat evolved. Cooling. 200° to 100° C	Heat evolved. Cooling. 100° to 40° C.	Heat evolved. Total	Heat evolved. Per cent.
	Kg.		*Kg.-cal.*	*Kg.-cal.*	*Kg.-cal.*	
CO_2	88.3	0.22	1,940	1,165	3,105	2.7
H_2O:						
Remaining as vapor	39.5	.43	1,700	1,002	2,702	2.3
Condensing	132.0		5,680		5,680	4.9
Latent heat of condensation of 132 kg. H_2O (537 kg.-cal. per kg.)				71,400	71,400	61.6
Cooling of condensed water				7,930	7,930	6.8
N_2	578.0	.24	13,870	8,330	22,200	19.2
O_2	82.0	.22	1,805	1,080	2,885	2.5
Total	919.8		24,995	90,907	115,902	100.0

The calculation above neglects the minor constituents of the gas stream and makes no allowance for air leaking into the condenser system. Attention is called to the fact that a large proportion, 68.4 per cent, of the total quantity of heat to be dissipated is due to the water vapor liquefied in the condenser. Even though, as here, a certain amount of the resulting water-mist escapes with the outgoing gas stream, the heat evolved in its condensation must nevertheless be dissipated by the condenser. It is evident from this that the burden of heat to be dissipated by the condenser can be materially reduced by drying the ore before charging it to the furnace. The presence of a large amount of water vapor is disadvantageous, not only because of the heat it evolves in condensing, but also because of condensation taking place along a considerable length of condenser, thereby retarding the cooling of the gas stream and causing the mercury to be distributed over a greater length of condenser than it would be otherwise.

The above figure for the amount of heat to be dissipated by the condenser per metric ton of ore treated is fairly representative of quicksilver practice. As indicated, a decrease in the quantity of water vapor entering the condenser system will materially decrease this amount of heat, and in some plants where very damp ore is being treated, the amount of heat may be 20 to 25 per cent greater. The Scott furnace, for which the above calculations were made, treated about 62 metric tons of ore per 24 hours, thus making the total amount of heat to be dissipated by the condenser system, roughly, 300,000 kg.-cal. per hour, or 5,000 kg.-cal. per minute.

CHOICE OF MATERIAL FOR QUICKSILVER CONDENSER.

The designer of a condenser system may approach his problem in a theoretical way by calculating the area of condenser surface required to transmit to the cooling medium the quantity of heat involved. The choice of material for a quicksilver condenser is limited in several ways. The material must be practically unaffected by mercury, water, and dilute sulphuric acid, and possess the necessary structural qualifications. In addition, its conductivity should of course be relatively good and its cost should be moderate. The materials that have been used in building condensers include wrought iron, cast iron, glass, red brick, glazed or chemical stoneware, and wood. Of these, wrought iron and cast iron have the highest thermal conductivity. Wrought iron has the advantage that it can be used in comparatively thin section, thus bringing the weight of material needed within reasonable limits, but it has the disadvantage, not possessed by cast iron, of being readily attacked by dilute sulphuric acid. Both kinds of iron have been employed to some extent for condenser construction, and as far as serviceability is concerned the use of thin wrought-iron pipe for the part of the condenser system near the furnace, in which there is no condensation of dilute sulphuric acid, is entirely feasible. Owing to the greater thickness of cast-iron pipe, its use for the part of the condenser system exposed to the attack of acid involves a relatively large expense.

A condenser constructed partly of glass was used at New Almaden, but because of the cost of construction and maintenance, glass as a condenser material does not deserve serious consideration. Wood, particularly if impregnated with pitch or similar material, and well-burned red brick have the necessary resistance to mercury, water, and dilute sulphuric acid, but have the disadvantage of low thermal conductivity. Red brick has only 0.01 and wood about 0.001, the thermal conductivity of iron. Chemical or glazed stoneware has a chemical conductivity 1.3 to 1.5 times that of red brick, and has the advantage over the latter that it can be used in relatively thin section; moreover, it meets all of the other requirements mentioned above. The suitability of chemical stoneware for condenser construction is indicated by the fact that its use is standard at European plants. A consideration of the question of available condenser material in the United States led the writers of this report to conclude that ordinary glazed sewer tile could be used, and the progress made in this direction during the last few years will be discussed in detail later.

RATE OF HEAT TRANSFER.

The rate at which heat may be transmitted from one medium to another through a solid wall depends on a number of factors. In general, the rate of heat flow is proportioned to the difference in temperature and to the thermal conductivity of the wall. It also depends on other factors, including the thermal capacity of the substances between which the heat interchange takes place. The transfer of heat may be regarded as consisting of two processes: (1) The flow of heat through the wall, and (2) the transmission of heat to and from the surfaces of the wall from and to the substances that are being cooled and heated. The second process involves the factor of so-called "contact resistance," which is none too well understood. When a substance to be cooled and the cooling medium are both gases, and the separating wall is made of metal or other material of relatively good thermal conductivity, the contact resistance is the limiting factor. That is to say, the temperature drops between the two surfaces of the separating wall and the gas on the two sides will be relatively large, and the fall of temperature in the wall itself will be quite small. When, however, heat is transferred from gas to liquid, or vice versa, through a separating wall, this wall, if made of material of relatively good thermal conductivity, will take on a temperature very close to that of the liquid and the big temperature drop will be between the wall surface and the gas. Under these conditions, the thermal conductivity of the wall exerts only a minor influence on the rate of heat flow, but when the wall is very thick or is made of material having a low thermal conductivity, the thermal resistance of the wall itself may be the chief factor in determining the rate of flow.

Data in regard to heat transfer are extremely variable. A few values for thin separating walls, taken from papers by Hering[77] and others believed to be reasonably accurate, will be of interest. The rate of heat transference is expressed in kilogram-calories (kg.-cal.) per square meter of surface per 1° C. difference in temperature per hour, as follows:

Ratio of heat transfer.

	Kg.-cal. per sq. m. per 1° C. difference per hour.
Air to metal to air	5–10
Air to metal to water	30–70
Air to stoneware to air	4–8
Air to stoneware to water	23–55
Air to wood to air	0.3–0.8

[77] Hering, Carl, Flow of heat through bodies: Met. and Chem. Eng., vol. 9, 1911, p. 652; Flow of heat through contact surfaces: Met. and Chem. Eng., vol. 10, 1912, p. 40.

These figures indicate comparatively little difference between the heat-transmitting power of metal and stoneware but a much lower coefficient for wood, presumably because of its relatively low thermal conductivity. It is also to be noted that the rate of transfer from gas to water, or vice versa, is, roughly, seven times greater than from gas to gas. This difference, which was to be expected, is due to the relatively small contact resistance between the water and the surface of the separating wall. When condensation occurs on the separating surface the contact resistance is also much less than when heat passes from a gas to the solid surface. When transfer of heat is from gas to solid the contact resistance can be somewhat reduced by increasing the velocity of the gas in contact with the surface, either by increasing the rate of flow of the gas stream as a whole or by setting up eddy currents by means of baffles or frequent changes in direction. The thermal conductivity and contact resistance of ordinary red brick, such as have been used in building quicksilver condensers, are probably not very different from that of stoneware or glazed tile; but the necessarily greater thickness of the brick wall materially reduces the efficiency of heat transmittal because of the large drop in temperature in the wall itself.

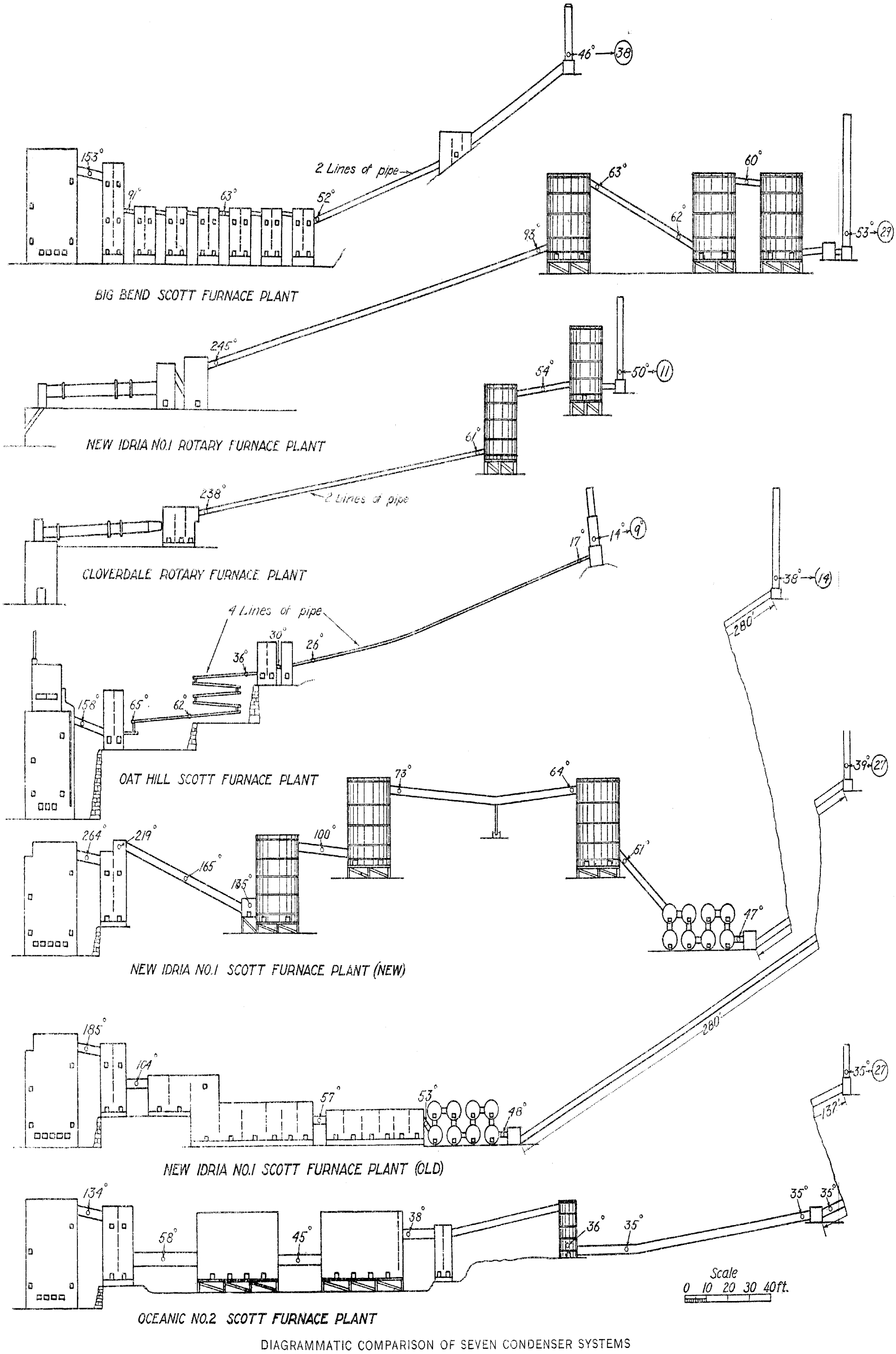
The conclusions to be drawn from this discussion are that metal and stone ware are the most suitable materials for condenser construction, as far as efficiency of heat transfer is concerned; and that with water cooling the necessary condenser area is much less than with air cooling.

ANALYTICAL DISCUSSION OF VARIOUS CONDENSER SYSTEMS.

Plate XIX shows seven different types of condensers that are now used or until recently have been used in this country. The diagrams are all drawn to scale, and the actual elevations are shown. Several of the condensers are built on hillsides and have a more compact arrangement than is indicated by the diagrams. The numbers at various points along the condenser systems show the mean temperature of the gas stream in degrees Centigrade. The numbers in circles next to the stacks are the corresponding mean atmospheric temperatures.

1. SCOTT FURNACE AT BIG BEND MINE.

The first diagram (Pl. XIX) illustrates the 50-ton (nominal rating) Scott furnace and condenser at the Big Bend mine, Brewster County, Tex. (See also Pl. XXI, *B*, p. 121.) The chambers are built of brick, the connection between the chambers and a part of the flue leading to the stack are of glazed tile, and the remainder of the flue is of concrete.



DIAGRAMMATIC COMPARISON OF SEVEN CONDENSER SYSTEMS

2. ROTARY FURNACE AT NEW IDRIA MINE.

The second diagram (Pl. XIX) illustrates the first rotary furnace installed at the New Idria mine, San Benito County, Calif. Glazed sewer-tile and redwood tanks form the condenser system. The tile are 2 feet in diameter, and the wooden tanks 30 feet high and 20 feet in diameter. The wooden stack is 50 feet high and 4 feet in diameter. In the tile pipe just beyond the second brick dust-chamber is a water spray for the removal of dust. A few feet beyond a stream of cold air, supplied by a No. 2 Buffalo Forge Co. blower, is led into the tile pipe to effect a quick cooling of the gas stream. The volume of the air is roughly equal to the volume of gas from the furnace. Water supplied to the wooden tanks at the top overflows down the outside; the primary purpose is to keep the tanks from drying and cracking, but the evaporation of this water no doubt has some cooling effect.

3. ROTARY FURNACE AT CLOVERDALE MINE.

The third diagram (Pl. XIX) illustrates a similar plant at the Cloverdale mine, Sonoma County, Calif. (See Pl. XXII, *C*.) Two lines of 15-inch tile are used between the dust chamber and the first wooden tank; the remaining connections are of 21-inch tile. The wooden tanks are 26 feet high and 15 feet in diameter. At both the New Idria and Cloverdale plants the tanks have sloping bottoms; a drain at the lowest point leads to concrete pots below. The water used for the spray at the Cloverdale plant amounted to about 4.5 gallons per minute. Use of the stream of cold air for cooling was abandoned after a short trial because it caused a slight back pressure in the dust chamber.

4. SCOTT FURNACE AT OAT HILL PLANT.

The condenser at the Oat Hill plant, Napa County, Calif., shown in the fourth diagram (Pl. XIX), was constructed according to a general design suggested by the authors, and consists essentially of glazed tile 12 inches in diameter. The Scott furnace is of the three-tile, four-shaft type with a nominal rating of 40 tons a day, and is equipped with the inclined shelf drier, previously described.

5. NEW IDRIA NO. 1 SCOTT FURNACE (NEW).

The fifth diagram (Pl. XIX) illustrates the first tank-and-pipe condenser constructed at the New Idria works. The five-tile, four-shaft Scott furnace, which was oil-fired, had a capacity of about 70 tons. In the reconstruction of the old condenser system, which is shown in the diagram just below, the first so-called tall brick condenser of the old system was retained; it was connected with the first redwood tank by a line of 30-inch glazed tile. A stream of cold air equal to about one-third of the gas volume leaving the furnace was introduced at the beginning of this line of 30-inch tile, and a water spray was also used there. Circular wooden flues, 45 inches in diameter, connected the remaining tanks, and the long 36-inch flue leading to the stack is also of wood. The reader will note that the wooden barrels that formed part of the old system were retained in the new plant.

6. NEW IDRIA NO. 1 SCOTT FURNACE (OLD).

The sixth diagram (Pl. XIX) illustrates the old system at New Idria, which consisted of the conventional stone-and-brick chambers, followed by eight small wooden tanks laid horizontally, and a long wooden flue leading to the stack.

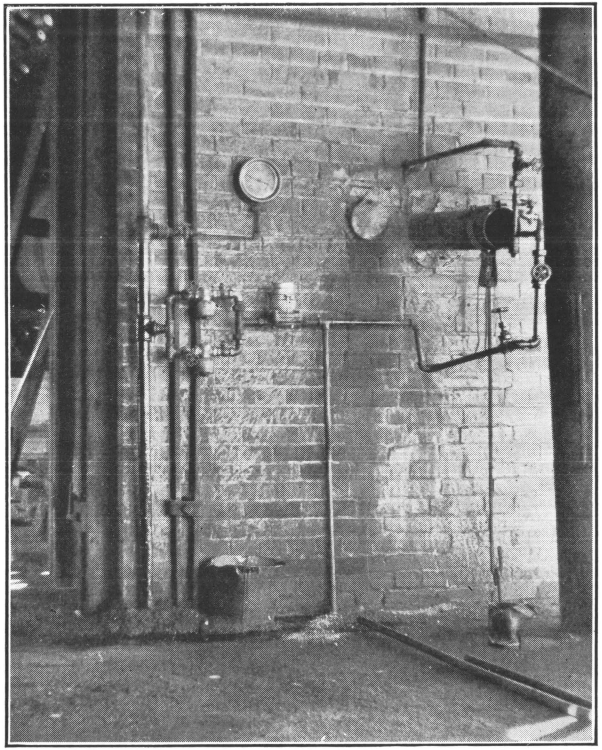
7. OCEANIC NO. 2 SCOTT FURNACE.

The bottom diagram (Pl. XIX) illustrates the condenser system at the Oceanic mine, San Luis Obispo County, Calif. The Scott furnace has a capacity of about 50 tons a day, and the distinguishing feature of the condenser system is the use of two large rectangular wooden chambers.

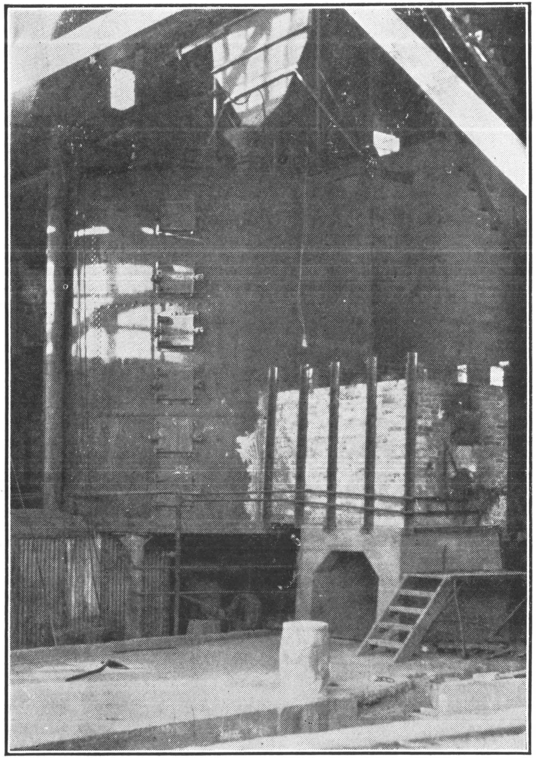
RELATIVE MERITS OF CONDENSER SYSTEMS.

In considering the relative merits of these condenser systems one should have in mind the criteria by which they are to be judged, or, in other words, the requirements that should be fulfilled by an ideal system. The efficiency of the condenser as a cooling device may be judged by the amount of heat which it dissipates, and the final temperature of the escaping gas as related to the area and volume of the condenser system. Obviously, the smaller the condenser the easier it will be to clean, and the less the likelihood of leaks remaining undetected. The efficiency of the condenser in collecting the condensed mercury may be judged by the loss of mercury from the stack.

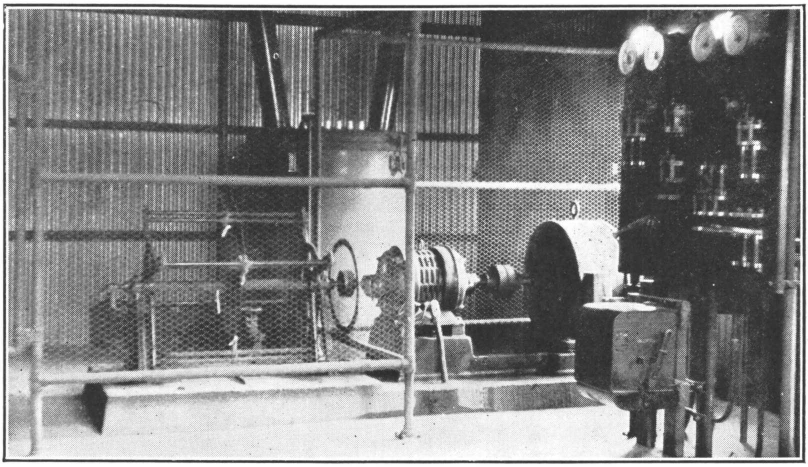
Next in importance to the satisfactory performance of these basic functions come the first cost of the condenser and the cost of maintenance and operation. A condenser should occupy as small an area as possible in order that a concrete floor, or other provision for recovering any mercury that may escape from the system, can be provided at reasonable cost, and should be so arranged that the mercury will collect at a few points only. A sectionalized construction is highly desirable because one unit at a time may be cut out for cleaning or repairs without interrupting the operation of the plant.



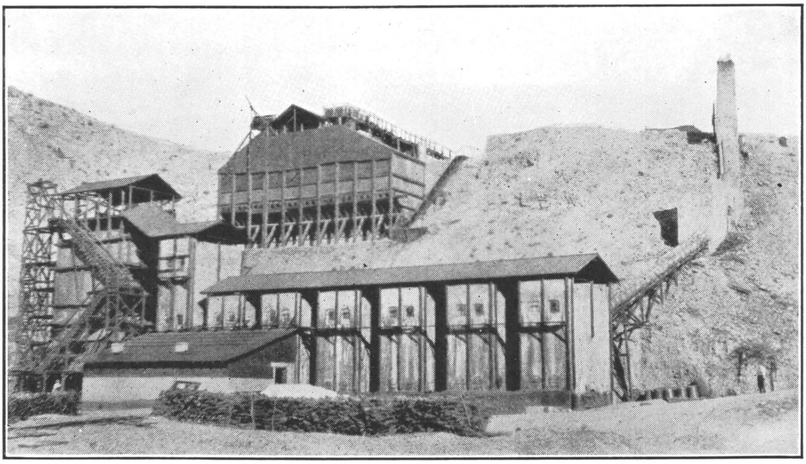
A. OIL BURNER AND FIRE BOX OF THE ROTARY KILN, CHISOS MINE, TEXAS



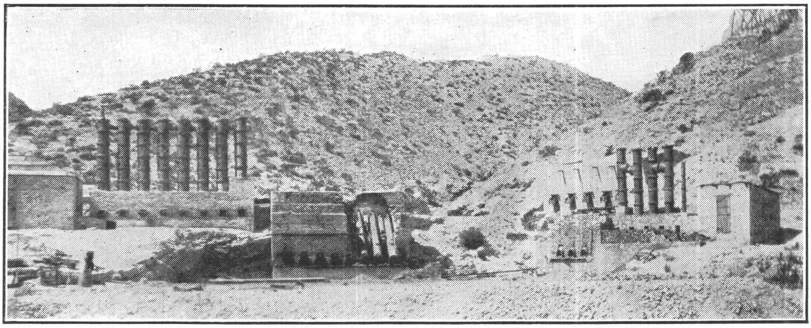
B. HERRESHOFF FURNACE AND EXTERNAL FIRE BOX, SENATOR MINE, CALIFORNIA



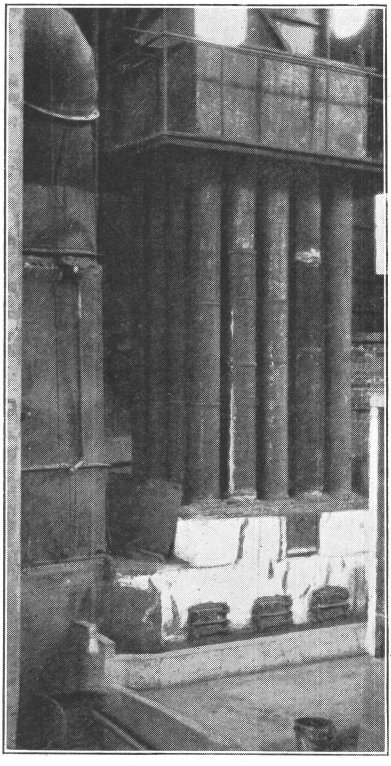
A. TRANSFORMER, MOTOR-GENERATOR SET, AND RECTIFIER FOR ELECTROSTATIC PRECIPITATION BY THE COTTRELL PROCESS AT NEW ALMADEN, CALIF.



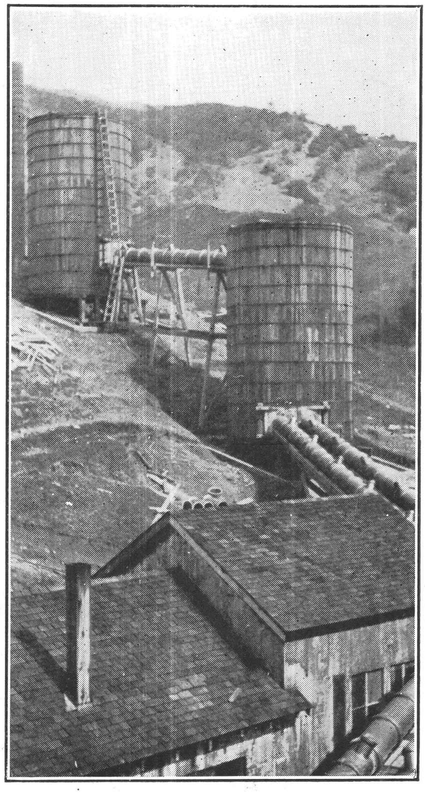
B. SCOTT FURNACE AND BRICK CONDENSERS AT THE BIG BEND MINE, TEXAS



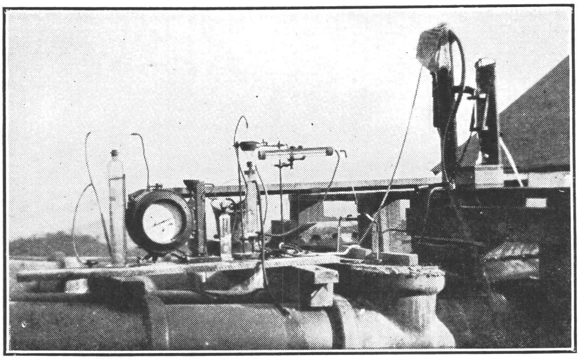
A. LARGE INCLINED RETORTS AND VERTICAL TILE-PIPE CONDENSERS AT THE MARISCAL (ELLIS) MINE



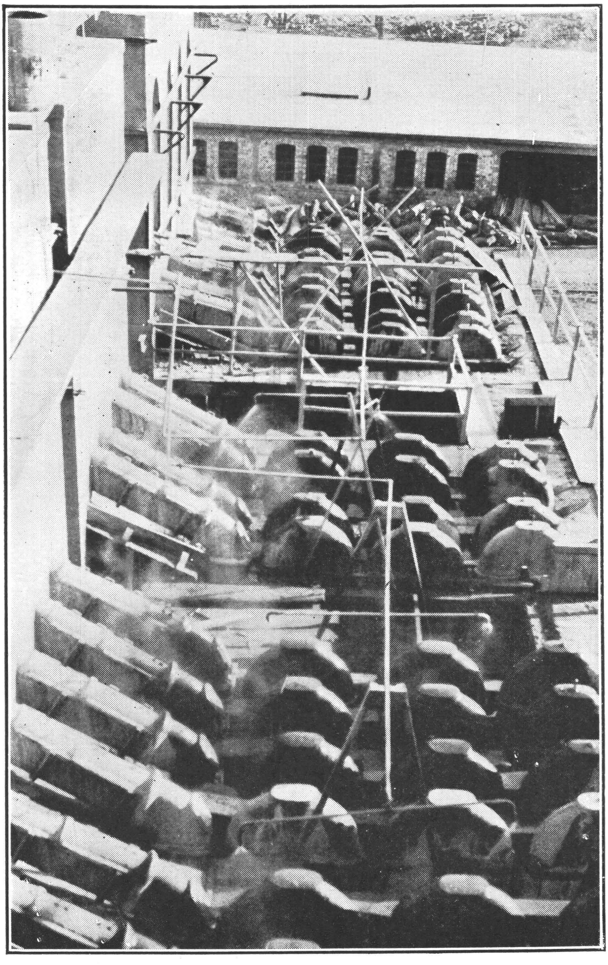
B. IRON-PIPE CONDENSER USED AFTER COTTRELL PRECIPITATION AT THE SENATOR MINE, CALIFORNIA



C. TANK-AND-TILE CONDENSER AT THE CLOVERDALE PLANT, CALIFORNIA, SHOWING METHOD OF CONNECTION



A. FUME-SAMPLING (LEFT) AND GAS-SAMPLING (RIGHT) APPARATUS IN OPERATION ON TILE-PIPE COOLING UNIT, OAT HILL PLANT, CALIFORNIA



B. BATTERY OF CERMAK CONDENSERS AT ABBADIA SAN SALVATORE, ITALY (VIEW FROM ABOVE)

All parts of the condenser should be easily accessible for cleaning, inspection, and repairs, and the condenser should be able to take care of fluctuations in the volume and temperature of the gas stream coming from the furnace, and of changes in atmospheric conditions. The durability of the condenser structure and the ease with which worn out or damaged parts may be replaced are also important considerations.

In comparing the various condenser systems shown in Plate XIX one should keep in mind how conditions at the several plants differ with respect to the factors that influence the process of condensation. These factors include the volume and composition of the gas stream, and especially its content of water vapor and the prevailing atmospheric temperature. Because of the differences among these factors an exact quantitative comparison of the various condenser systems with respect to rate of heat transference is impossible, and the best that can be done is to compare performances in a qualitative way. A summary of the available data on the several systems appears in Table 14. The estimated quantities are indicated; all others are based on actual measurements. The technique employed in measuring gas volume, determining stack losses, etc., has been discussed in detail elsewhere.[78]

DETERMINATION OF MERCURY CONTENT.

Plate XXIII, *A*, illustrates the apparatus used in determining the mercury content of the gas at the stack and at the other points along the condenser system.

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[78] Duschak, L. H., and Schuette, C. N., Fume and other losses in condensing quicksilver from furnace gases: Tech. Paper 96, Bureau of Mines, 1918, 29 pp.; Min. and Sci. Press, vol. 117, 1918, p. 315.

TABLE 14.—*Comparison of seven condenser systems.*

	New Idria.			Oceanic, Nos. 1 and 2 Scott, brick and wooden chambers.	Big Bend, Scott, brick chambers (regular Scott condenser).	Oat Hill, Scott, glazed tile and brick chambers.		Cloverdale, rotary, glazed tile and redwood tanks.
	No. 1 rotary, redwood tank and glazed tile.	No. 1 Scott (old), stone and brick chambers and wooden barrels.	No. 1 Scott (new), redwood tanks and barrels.			Entire system.	First chamber and zigzag only.	
Gas volume, cubic meters at stack conditions:								
Per 24 hours	40,040	69,800	97,430	62,800	[a] 30,000	24,048	26,400	[a] 40,000
Per ton of ore [b]	560 (57)	1,160 (60)	1,400 (70)	700 (90)	670 (45)	600 (40)	660 (40)	570 (70)
Per kg. of Hg. [b]	144 (277)	116 (600)	118 (825)	370 (170)	149 (202)	264 (91)	290 (91)	126 (318)
Mercury loss in grams per cubic meter; stack gas at 0° C. and 760 mm.:								
Total	0.1935	0.029	0.085	0.047		0.0300	0.0633	
Vapor loss	.1630	(0.059)	0.052	0.034		0.0060	0.0460	
Mist loss by difference	.0305		0.033	0.013		0.0240	0.0173	
Total loss per 24 hours, kg	7.87	2.01	8.18	2.94		0.722	1.674	
Loss in per cent of Hg., input	2.84	0.33	0.99	1.75		0.79	1.84	
Temperature of gas entering condenser, °C	245	185	264	134	153	158	158	238
Mean stack temperature, °C	53.7	39.4	37.7	31.9	46	11.7	36	50
Difference between atmospheric and stack temperature, °C	24	12	24	8	8	5	27	39
Cooling area, square meters	982.6	1,006.9	1,577.3	2,370	790.8	571.5	279.4	487.1
Condenser volume, cubic meters	879.1	489.0	1,098.5	1,815	452.0	127.5	56.2	305.7
Ratio $\frac{\text{cooling area in square meters}}{\text{gas volume, cubic meters per minute}}$	35.4	20.7	23.3	54.4	38.0	34.3	15.3	17.6
Ratio $\frac{\text{condenser volume, cubic meters}}{\text{gas volume, cubic meters per minute}}$	31.7	10.36	16.3	41.6	21.6	7.63	3.03	10.94
Number of changes in direction	8	50	25	25–12	35	21	15	4
Number of changes of gas velocity	6	48	24	30–10	32	11	5	4
Shortest path through condenser system, meters	145	261	253	165–166	137	115	61.6	91.5

[a] Estimated.
[b] Figures in parentheses indicate approximate daily tonnage of ore and daily production of quicksilver, in kilograms.

COMPARISON OF MERCURY LOSSES.

A comparison of the quantity of mercury escaping from the various condenser stacks brings out the fact that the lowest losses for like volumes of gas were shown by the old condenser system used with the No. 1 Scott furnace at New Idria and by the system at the Oat Hill mine. The total mercury loss is larger at New Idria because of the larger gas volume, but the percentage loss is smaller because of the higher mercury content of the furnace charge at New Idria, where the daily quicksilver production was about 600 kg., as compared with 91 kg. at Oat Hill.

The figures given for the Oceanic plant are the combined values for the two Scott furnaces and condenser systems. The grade of ore was nearly the same as that at Oat Hill, and the tonnage and gas volume of the former is roughly double that of the latter, so that a fair comparison may be made. The actual mercury loss from the Oat Hill condenser is about one-fourth of that from the Oceanic, and the percentage loss is about one-half. The poorest showing is that of the condenser system of the No. 1 rotary at New Idria, where both the mercury carried away per cubic meter of gas and the percentage loss of mercury are higher than at any other.

COMPARATIVE COOLING ACTION.

The cooling action of the various condenser systems may be judged roughly by the temperatures shown in Plate XIX (p. 118). A closer comparison is obtained by reference to the curves given in Figures 11 and 12, which show the drop in temperature as related to condenser area and condenser volume, respectively. In all the systems atmospheric air is largely relied on for cooling, and the average atmospheric temperature is indicated on these curves by the horizontal line at the end of the curve for each system. A difference between the temperature of the gas escaping from the condenser stack and that of the atmosphere is shown by the vertical line rising from that horizontal line. Anyone examining the curves that show the relation between temperature and surficial condenser area can judge from the steepness of the curves the rates at which heat is dissipated per unit area by the parts of any given system. It is to be expected, of course, that the first part of each curve will show the steepest inclination, since it corresponds to the part of the system where the difference between the temperature of the condenser gas and of the outside air is largest.

Reference to the left-hand part of the curves, which corresponds roughly to condenser-gas temperatures above 100° C., shows that in general the most rapid temperature drops are in tile pipe, although brick or stone condensers show fairly rapid cooling. The part of the curves which, however, deserves closer scrutiny is that at the right, because a high recovery by a condenser system is not to be expected unless the gas stream is cooled until its mercury-vapor content is negligible. The best final cooling is shown by the Oat Hill condenser,

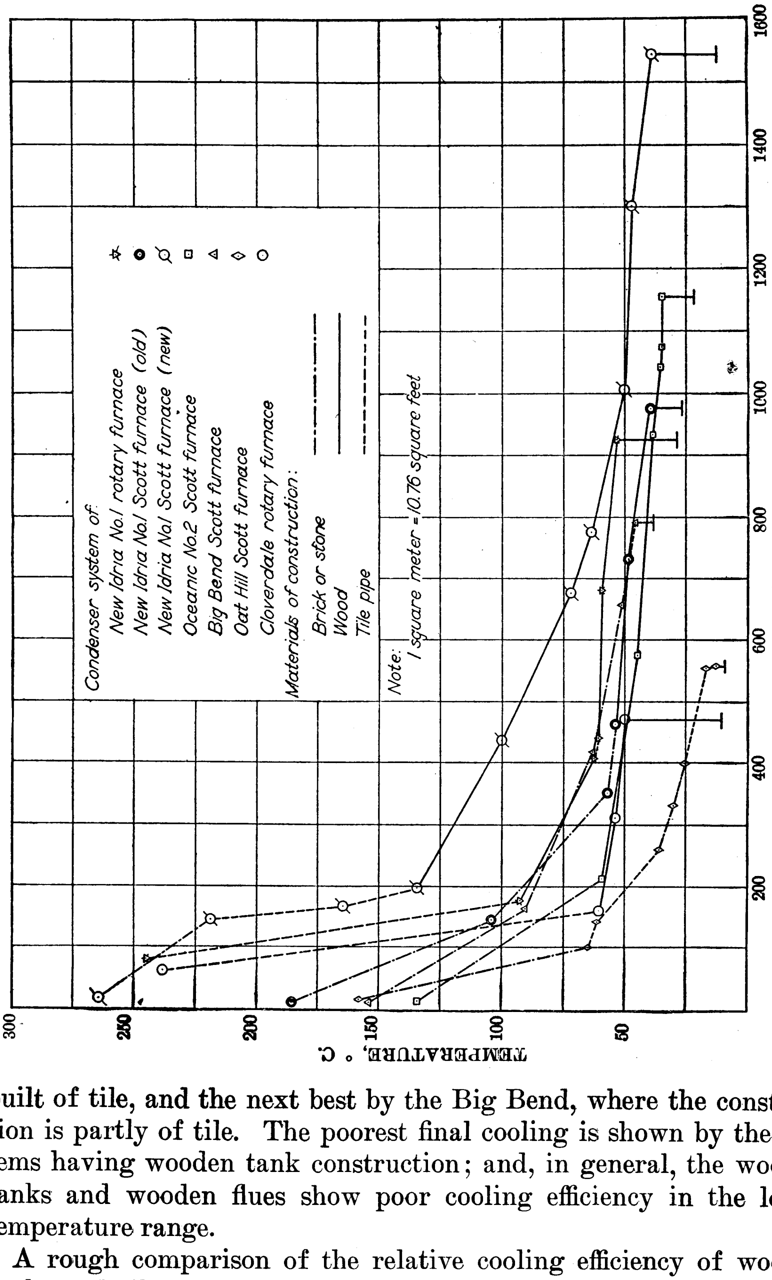

FIGURE 11.—Comparative curves showing relation of condenser temperatures to condenser area.

built of tile, and the next best by the Big Bend, where the construction is partly of tile. The poorest final cooling is shown by the systems having wooden tank construction; and, in general, the wooden tanks and wooden flues show poor cooling efficiency in the lower temperature range.

A rough comparison of the relative cooling efficiency of wooden tanks and tile may be made by examination of the Cloverdale and

Oat Hill systems. The atmospheric temperatures at the two plants were nearly the same, 11° and 9° C., respectively. The condenser gas enters the first Cloverdale tank at 61° C., and has a temperature of 50° C. at the base of the stack. The temperature of the gas on entering the zigzag part of the Oat Hill condenser is 62° C., and on leaving is 36° C. The area of the zigzag, however, is 117 square meters, compared with about 300 square meters for the two tanks of the Cloverdale system. Allowance, however, is to be made for

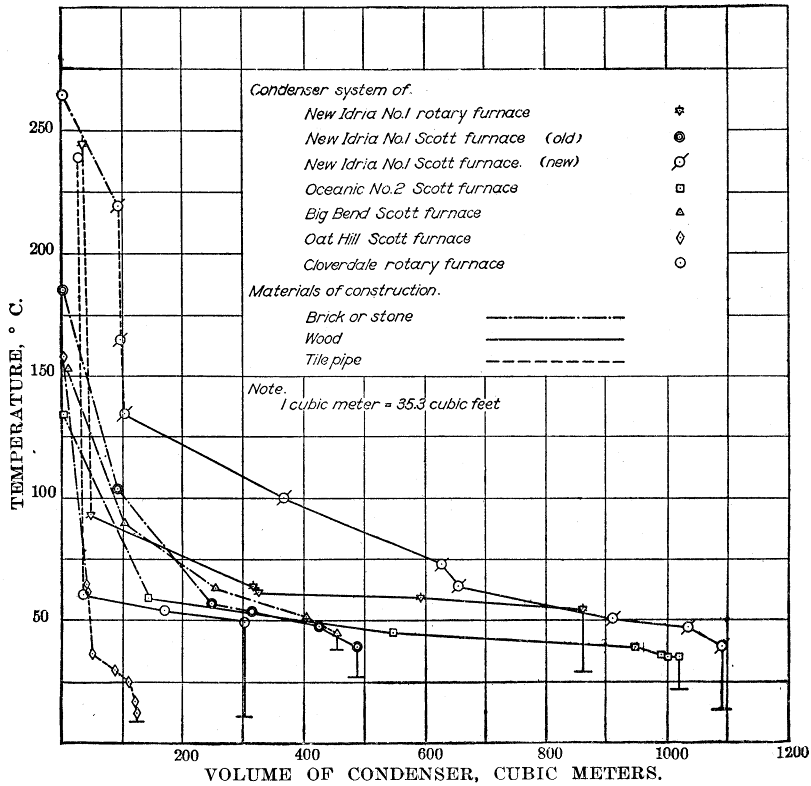

FIGURE 12.—Comparative curves showing relation of condenser temperatures to condenser volume.

the fact that the gas volume, and consequently the quantity of heat to be dissipated, is about 50 per cent greater at the Cloverdale plant.

A similar indication of the inefficiency of wooden construction for cooling in the lower temperature range is given by the condenser at the Oceanic mine. The two wooden chambers forming part of the condenser system attached to the No. 2 Scott furnace have a surface area of about 720 square meters and handle a gas volume roughly equal to that of the Oat Hill plant. In spite of this large area, however, the drop in temperature through these two chambers is only 20° C. Part of the rapid cooling shown by the tile pipe leading to the first wooden tank of the Idria rotary and the Cloverdale systems is due to a water spray which is introduced at the beginning of the line of tile to precipitate the dust that is not collected by the dust chambers.

The mixing of cold air with the condenser gas for the purpose of effecting rapid cooling does not seem particularly desirable. Initial rapid cooling of the gas stream is easily accomplished without it, and the dilution of the gas stream with a large volume of air has the disadvantage of increasing the volume of gas to be handled by the remainder of the condenser system, thus leading to a larger mercury loss from the stack. Cooling by the direct application of a water spray is effective as far as the cooling itself is concerned, but a large part of the mercury condensed in this way comes down very finely divided, thus introducing the danger of some of it being lost in the water that flows away; furthermore, any soot collected in this manner comes down in a condition which makes its handling difficult.

With reference to the use of wooden tanks or rectangular chambers, it is noteworthy that this form of construction is quite unsuitable for the part of the condenser system nearer the furnace. A great deal of difficulty was experienced in maintaining the first tank of the so-called new condenser system attached to the No. 1 Scott furnace at New Idria. The interior of the tank and the wooden supports for the top soon became charred, and it was necessary to erect an exterior wooden truss to keep the top from falling in. The temperature was not, in general, high enough to char the wood, so the action may have been due to acid vapor. In order to protect the wood and prevent the seams of the tank from opening, the surface had to be kept continually wet; and part of the cooling accomplished by this tank was, no doubt, due to water which leaked through the top and trickled down along the inside of the staves.

The experience with wooden chambers at the Oceanic plant has not been particularly satisfactory, and the cost of maintenance of these large tanks or rectangular chambers is high. Small redwood tanks, particularly when coated with pitch or similar material, show a much longer life. Wooden tanks with nearly horizontal bottoms were found rather difficult to clean, because the mercury and heavy soot collected on the bottom to the depth of several inches and was difficult to move with an iron hoe or rake, because these tools tend to float on the mercury. As a result the workmen sometimes actually had to enter the tanks—a dangerous procedure that is to be avoided if possible.

A comparison of the second section of the brick construction at Big Bend with the Oat Hill zigzag indicates that for intermediate temperatures the tile construction gives more effective cooling, although the difference is not very great. As suggested above, the difference in favor of the tile is largely a result of the greater thickness of the brick wall.

By referring to the ratios of condenser area and condenser volume to the gas volume, given in Table 14, the reader will see that the largest ratios do not correspond to the best cooling efficiency. Take the old and new condenser systems for the Scott furnace at New Idria, as an example. The new system, although it showed both a larger area and volume ratio than the old system, was less effective for cooling. The condenser showing the highest cooling efficiency, the one at the Oat Hill, has the smallest volume ratio, which indicates that volume alone is not a reliable index of the efficacy of cooling.

MIST LOSS FROM CONDENSERS.

As already pointed out, the ability of a condenser system to collect the condensed mercury is just as important as its efficiency in cooling the gas stream. The relative collecting ability of the several systems under discussion should be indicated by the so-called mist loss from the condenser stack, that is, the quantity of mercury carried away as suspended particles by a given volume of gas. As pointed out by the present authors in their paper on condenser losses,[79] the calculation of the proportion of the total loss due to the escape of particles of mercury is subject to considerable uncertainty, owing to the gas stream being apparently both undersaturated and supersaturated with respect to mercury vapor. The mist losses recorded in Table 14 were derived by deducting the calculated vapor loss (saturation at stack temperature being assumed) from the total loss as determined experimentally. This calculated mist loss is least for the Oceanic system, which shows the highest ratio of condenser volume to gas volume. On the other hand the next lowest mist loss is that of the Oat Hill condenser, which has the smallest volume ratio; evidently it is not alone the volume available for settling of the mercury droplets but the way in which this volume is disposed that determines the collecting power of a system. As pointed out in Technical Paper 96, several factors contribute to the collection of suspended particles from a gas stream. Various methods of baffling, by causing changes in the velocity and the direction of the gas stream, help to throw down suspended particles. The whole subject has been treated at length by Goodale[80] and

[79] Duschak, L. H., and Schuette, C. N., Fume and other losses in condensing quicksilver from furnace gases: Tech. Paper 96, Bureau of Mines, 1918, 29 pp.

[80] Goodale, C. W. and Klepinger, J. H., The Great Falls flue system and chimney: Trans. Am. Inst. Min. Eng., vol. 46, 1913, p. 583.

Klepinger. In the Oceanic condenser system much of the precipitation of the fine suspended particles of mercury is probably a result of the slow settling of these particles under the force of gravity. In the Oat Hill system the collection occurs largely through mechanical action. The frequent changes of direction and the comparatively high velocity of the gases undoubtedly set up eddy currents which, owing to the narrowness of the gas passages, probably cause particles in suspension to impinge on and adhere to the condenser walls. In this connection it is of interest to note that the mean velocity of the gas stream through one of the large settling chambers of the Oceanic system is about 0.01 meter per second, whereas that in the Oat Hill system is 1.16 meters. In their comparison of the old and the new condenser systems used for the Scott furnace at New Idria the authors pointed out, in the report already mentioned, that the smaller loss shown by the old system was probably due in part to the greater baffling effect resulting from the larger number of changes in direction and velocity of the gas stream.

The condenser system connected with the Herreshoff furnace at the Senator mine at New Almaden, Calif., was equipped with apparatus for the electrical precipitation (Cottrell process) of the suspended particles of mercury. Such apparatus effectively prevents the loss of any mercury particles in suspension, but since this loss, as shown by Table 14, is comparatively small, it is an open question whether special provision of this sort pays.

CONCLUSIONS.

The conclusion to be drawn from a consideration of these various condenser systems is that with respect to the efficiency of both cooling and collecting, the tile-pipe construction is the best. This type of condenser will be considered in some detail on pages 129 to 135. Brick condensers, particularly for that part of the system next to the furnace, show fairly good cooling and collecting efficiency, but have the disadvantage of absorbing some mercury, and unless special precautions are taken, a certain loss of mercury will take place through the bottom of the condenser. This fact is attested by the large amount of mercury collected from the ground beneath two of the old condenser systems in California. Wooden tanks or chambers do not show high efficiency as coolers, but a series of small wooden tanks or chambers, or a large chamber with some baffling device, might be used to advantage at the end of the condenser system for collecting suspended mercury.

Cast-iron condensers have been used to some extent in California, and when the ore is low in sulphur and consequently the amount of acid formed is small, they have shown good life. Compared on the basis of equal cooling areas, cast-iron construction is six to ten times as expensive as glazed-tile construction, and the use of special acid-proof alloys would be even more expensive. The initial cost of a condenser system for a 40 to 50 ton furnace, based on cost figures collected from a number of plants, is estimated as follows:

Estimated cost of a 40 to 50 ton furnace.

Brick or masonry chambers	\$10,000 to \$15,000
Wooden tank-and-tile condenser	4,500 to 5,500
Tile-pipe cooling unit with one tank or chamber	2,500 to 3,000

In actual practice, except where much acid is formed, the maintenance charges on a brick or masonry condenser are probably the lowest, although sooner or later the entire bottom of the condenser has to be replaced; and frequently the entire condenser system is torn down and the material passed through the furnace in order to recover the mercury that has been absorbed. This retention of mercury and possible loss through the condenser bottom more than offset the low maintenance cost. With respect to the other two systems, maintenance on the tank-and-tile construction is considerably higher than on a condenser construction largely of tile.

In regard to compactness and ease in recovering the mercury, the tile construction is preferable; moreover it can be easily sectionalized so that one part of the system can be cleaned or repaired without interrupting the operation of the plant.

THE TILE-PIPE CONDENSER.

In view of the favorable showing made by the condenser system at the Oat Hill mine, this type of construction deserves further consideration. There is nothing particularly new in the idea of using chemical stoneware or glazed tile for quicksilver condensers. The Cermak condensers, which are constructed of specially designed stoneware pipe, have been the standard in Italy and Austria for some time (Pls. XXIII, *B*, p. 121, and XXV, *A*, pl. 140). One condenser of this type is in use at the Chisos plant in Texas. Glazed sewer tile has been used at a few plants in the United States, but until the tank-and-pipe condenser construction was tried out at New Idria, the brick or masonry condenser was standard in this country. In studying the condenser problem, it seemed to the authors that the use of ordinary glazed sewer tile for condenser construction had not received enough attention. Glazed tile is relatively inexpensive, and, as already pointed out, it possesses the requisite properties of condenser material—relatively good thermal conductivity, necessary mechanical strength, and good resistance to mercury and dilute sulphuric acid. Glazed tile of large diameter have been used to some extent for connections between brick condenser-chambers and for flues leading to the stack. Tile of large diameter costs considerably more per unit of area than smaller pipe, the price of 24-inch pipe, for example, being nearly twice that of 12-inch pipe. The larger sizes are also less strong mechanically, not only because of the larger diameter but because mechanical strain and imperfections are more likely to occur in molding and burning of large pipe. These various considerations led to the idea of using pipe of moderate diameter in condensers.

THERMAL RESISTANCE.

One of the first points that arose was the ability of glazed sewer tile to withstand the thermal shock that might result from changes in temperature of the gas stream leaving the furnace, and also the unevenness of intermittent wetting of the outside of the pipe in connection with water cooling or the "hosing down" of the condenser system. This point was investigated experimentally. The authors are indebted to W. C. Riddell, formerly of the bureau staff, for assistance in making these tests, which were briefly as follows: A brick fire box was constructed with a tile-pipe flue leading from it to a stack. A stream of hot gas was obtained by burning illuminating gas in the brick chamber, and a perforated water pipe was arranged above the string of tile so that the latter could be quickly quenched with cold water when desired. A number of experiments were made with strings of 8-inch and 12-inch sewer tile, purchased locally. Results showed that both the 8-inch and 12-inch tile were unaffected after a gas stream at a temperature of 200° C. had been allowed to pass through until thermal equilibrium had been established, and the tile was then flushed with water at a temperature of 15° to 18° C. Some cracking occurred in a similar experiment with gas temperatures varying from 250° to 300° C. The 8-inch tile resisted thermal shock at the higher temperatures somewhat better than the larger tile, and in general, the lengths that were hard-burned cracked more readily than those softer burned. It was concluded from these tests that tile at least up to 12 inches in diameter could safely be used for quicksilver furnace gases having a temperature of 250° C., and that by selection of the lighter burned tile the temperature range could be extended to 300° C. It is to be noted that the thermal shock applied to the tile in these tests is more severe than is likely to be received in practice. Moreover, when cracking did occur, the tile was not shattered; and after a single fine crack had developed a considerably more severe thermal shock could be applied without causing further cracking.

Through the cooperation of Murray Innes, of San Francisco, owner of the Oceanic quicksilver mine, San Luis Obispo County, Calif., arrangements were made for a series of small-scale tests of a tile-pipe condenser. An experimental condenser unit designed to handle about 20 per cent of the gas from one of the 45-ton Scott furnaces at the plant was constructed of 6-inch tile, arranged in seven 20-foot rows placed one above another in a vertical plane. It furnished a cooling area of about 21 square meters. The condenser was connected to the furnace through an opening in the exit pipe, and as the condenser was below the exit-pipe level a small blower was used to move the gases. A launder was arranged above the cooling unit so that the latter could be flushed with water if desired. The furnace gas entered the cooling unit at a temperature of 112° to 160° C. and was cooled to 30° C. In general, this cooling was accomplished in the first half or first two-thirds of the cooling unit. Calculation showed that a heat transference was realized of 20 to 28 kg.-cal. per square meter of surface per 1° C. temperature difference per hour. This result was considered particularly satisfactory in view of the fact that the furnace gas carried considerable carbonaceous soot, which soon coated the interior of the cooling unit. It will be noted that the heat transfer realized agrees with the values given on page 117.

THE COOLING UNIT.

The next step was to design a cooling unit for a large furnace. Because a supply of cold water adequate for the condenser is not available at a number of quicksilver plants, it was thought better to base the design on cooling by atmospheric air. For this purpose, in line with the discussion on page 117, a value of 3 for the heat-transference constant was assumed. Taking the gas stream from the Scott furnace at New Idria as an example, the authors showed on page 115 that the heat to be dissipated by the condenser system amounted to about 5,000 kg.-cal. per minute or 300,000 kg.-cal. per hour. In this case, the mean temperature difference between the condenser atmosphere and the outside air amounted to 80° C. We have, therefore, $\frac{300{,}000}{3\times 80}=1{,}250$ square meters as the area theoretically required to cool the gas stream to 40° C., corresponding to a little more than 4,000 linear feet of 12-inch tile. This calculation is only a rough approximation, since the process of heat transference is more complicated than that indicated by the above simple formula. More than half of the total heat to be dissipated from the above gas stream was due to the water that had to be condensed, and it was shown on page 112 that this particular condenser atmosphere did not become saturated with water until it attained a temperature of 61.5° C. Consequently, the release of this large amount of latent heat would not begin until that temperature was reached, and at that point the temperature difference between the condenser atmosphere and the cooling medium—here the atmospheric air—was much less than the average temperature difference assumed above. On the other hand, the rate of heat transfer to a solid surface is much more rapid from a condensing vapor than from a permanent gas, and this higher rate tends to offset the small temperature gradient of the part of the condenser system in which condensation of the water vapor actually occurred.

Through the further cooperation of Mr. Innes a condenser system, based on a design proposed by the authors, was built in connection with the 40-ton Scott furnace at the Oat Hill mine. Plate XXVI (p. 141) is a view of this system. The cooling unit consisted of five superimposed lines of 12-inch tile, arranged in two sections of two strings each, so that all parts were readily accessible for inspection and repairs. The two adjacent rows of tile were supported by two 4 by 4 inch wooden stringers on the outside and one 4 by 6 inch stringer between the rows, which in turn were supported by 8 by 8 inch wooden posts. Crosspieces and cleats were so arranged that very few nails or bolts were required, and the structure was practically self-supporting without these. A wooden launder was placed above each pair of cooling units so that the tile could be flushed with cooling water if desired. This launder had lips reaching to the center line of each string of tile, and holes were bored in the sides of the launder just above these lips so that the flow of water could be controlled by wooden pegs if necessary. Plate XXIV is a diagram of the system.

The entire cooling unit was built on a concrete floor that drained to a series of settling boxes so that any mercury escaping from the condenser would be collected. A drain for collecting the mercury was placed in each string of tile close to the brick chamber which formed the connection with the furnace exit pipes. An iron slide damper was placed in each string next to this brick chamber so that the gas stream could be shut off from one or more of the strings for cleaning or repairs. Beyond the cooling unit some brick chambers that remained from an old condenser system were used as settling chambers, and connection was made from the last of these to the base of the stack by four strings of 12-inch tile. All joints between the tile were sealed with Portland cement mortar. The various openings at the end of the strings of tile forming the cooling unit and in the tees placed elsewhere in the system were closed with wooden lids that were luted in with clay or mud.

The essential data in regard to the area and volume of the condenser, the volume of gas treated, and the cooling accomplished are given in Table 14. The condenser normally operated under natural

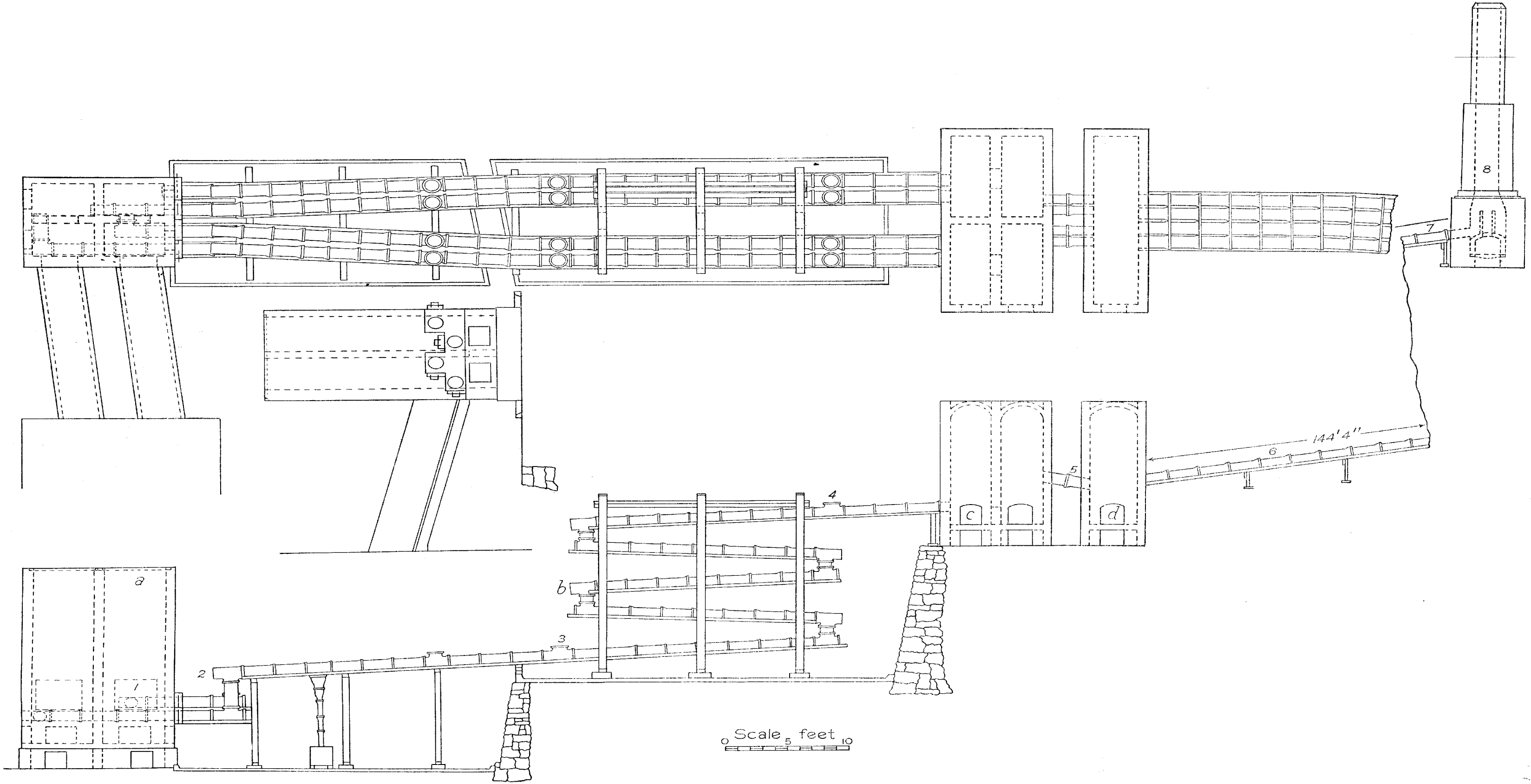

DIAGRAM OF THE PIPE CONDENSER SYSTEM, OAT HILL MINE, CALIFORNIA draft, but a small fire box was placed in the base of the stack so that the gas stream could be slightly reheated when necessary. A flow of water of about 19 liters (5 gallons per minute) proved sufficient to maintain a film over the outside of the cooling unit. With this small flow, the cooling effect probably was due as much to the evaporation of the water as to the heat actually removed by the water that flowed away. As the requisite cooling was obtained without the use of water, the arrangement for water cooling was held in reserve for use when needed. It is evident that if water cooling had been adopted as a regular practice a smaller cooling unit would have sufficed. In Table 14, with the data for the complete condenser unit are given corresponding values for the system up to the point where the cooling unit enters the first brick chamber. The reader should note that the performance of this part of the condenser system in cooling the gas stream and collecting the mercury compares favorably with that of the other system discussed, and that the volume-to-area ratios are much less than for any other system. This indicates that dispensing with the remainder of the system would cause only a slight loss of efficiency.

CLEANING.

The condenser system was cleaned once a month. The free mercury and soot were first hoed from the small brick chamber next the furnace, and then from the successive strings of tile in the cooling unit. In this cleaning the slide damper was closed, the wooden doors were removed in order, beginning at the top, and the mercury and soot were hoed and washed down progressively to the lowest pipe, from which the material dropped through a vertical drain pipe into the collecting box below. The mercury and soot from these boxes were removed to the soot pans and treated in the usual manner. Fully 95 per cent of the mercury was obtained from the cooling unit itself and a little from the first brick chamber. The brick chambers beyond the cooling unit and the flues running to the stack were cleaned at six-month intervals and yielded very little metal, a result that confirms the statement made above, that this part of the system was not essential.

The furnace and condenser system first began operation early in 1919, and when this paper was written had operated altogether about two years with satisfactory results. Some cracking of the tile had occurred, due presumably to the action of the acid water on the Portland cement used for the joints. This cracking, however, had not been serious and no tile had been replaced. The leakage of air into the condenser system as determined by gas analyses ranged from nothing to 10 per cent. A leakage of 50 per cent has been noted with other forms of condensers, and the tile condenser was found to be much freer from leaks than any of the other forms of construction investigated.

GENERAL DISCUSSION OF THE TILE-PIPE CONDENSER.

Of the various forms of condenser construction here considered there seems little question that the tile-pipe construction is the best as to both first cost and general performance. The tile construction meets all of the general requirements of a satisfactory condenser system mentioned early in this chapter. It can be arranged in a compact unit, with all parts easily accessible for inspection, cleaning, and repairs, and the removal of mercury is more easily accomplished than from any other form. The results with the tile-pipe condenser show that a large condenser volume is not necessary. The tile construction also provides effective cooling surface and the baffling effect needed in collecting the condensed mercury.

During the tests of the first experimental tile condenser at the Oceanic plant it was observed that a larger proportion of free-running mercury collected in the first part of the experimental unit than was obtained from the regular condenser system of the Oceanic plant. Apparently the tile condenser offers some advantage in treating a gas stream carrying in suspension a considerable amount of carbonaceous soot, as at the Oceanic furnaces. The good contact between the gas stream and the condenser surface results in rapid cooling and gives the mercury vapor a certain opportunity to condense directly upon the surface of the tile, whereas the light carbonaceous soot is carried farther along by the gas stream. The steep temperature gradient in the layer of gas next to the condenser surface also favors the diffusion of mercury vapor from the hotter part of the stream into the cooler layers next to the condenser surface and thus aids the separation of mercury from carbonaceous soot. Obviously, the lower the temperature of the condenser surface the better the opportunity for the separation to take place. This condition is obtained by water cooling, for the tile when flushed with a moderate volume of water takes on a temperature closely approaching that of the cooling water.

The use of specially made chemical-stoneware pipe, as in European practice, offers no particular advantage and is considerably more expensive. The variety of shapes of ordinary glazed tile that are available in the American market makes possible the fabrication of almost any form of tile condenser desired.

The condenser system at the Oat Hill plant was built originally for water cooling, which determined the tile arrangement adopted.

Several other arrangements are, however, possible; and in general the tile-pipe construction has a flexibility that makes possible its adaptation to the topography. A simple and in some ways more convenient design is one in which the strings of tile are placed vertically instead of nearly horizontally. With vertical tile the connections at the bottom could be easily arranged in the form of small brick or concrete chambers each having a clean-out door and a small mercury drain. The top connections could be made as in the Oat Hill system by the use of standard tees and short connecting pieces. This arrangement would be easier to clean than the Oat Hill system, and water cooling in the form of sprays could be applied, although the removal of the cooling water would not be quite as simple as with the Oat Hill arrangement. A condenser arranged in this way has been used at the Black Butte mine in Oregon and also at the Mariscal, formerly the Ellis, mine in Texas.

The Mariscal condenser illustrated in Plate XXII, *A* (p. 120), is built of pipe of large diameter and is used with a battery of inclined retorts of fairly large capacity. One advantage of the vertical arrangement is that the water and mercury, which usually condense in largest quantity in different parts of the system, can be drawn off separately and the necessity of handling a large quantity of wet, dirty mercury is avoided. Some acid-proof cement, such as is used in acid works, could be used to advantage instead of the Portland cement used for the joints in the Oat Hill condenser.

Although the results with the Oat Hill condenser indicated that the brick chambers beyond the cooling unit were hardly necessary, it would probably be desirable in general to provide settling chambers between the cooling unit and the stack. Brick or concrete construction could be used for this purpose, but small circular wooden tanks placed horizontally at a slight inclination are probably the most satisfactory and least expensive. Experience with tanks of this sort at New Idria demonstrates that if they are coated inside and outside with pitch, asphalt, or similar material, they last for many years. Tanks up to 6 or 8 feet in diameter give little trouble from opening of the seams, and if they are placed at a slight inclination, as suggested, the mercury drains to one end, from which it can be easily drawn off. A manhole in one end gives easy access for cleaning. A large tank or wooden chamber with wooden strips for baffles might be used, but small tanks seem preferable. By the use of small tanks this part of the system, as well as the gas cooler, can be separated into two or three units, so that one section at a time can be shut down for clean-up and repairs without interrupting the operation of the plant.

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CONDENSER OPERATION.

The essential features that require attention in connection with the operation of a condenser system are the maintenance of proper draft and effective cooling, and frequent inspection for leaks. Unless the operation of the furnace is subject to unusual variations, the temperature distribution along the condenser system will remain constant within small limits, subject to variations of atmospheric temperature. It is advisable to place a few thermometers, including one at the base of the stack, at various points along the condenser system, so that changes in temperature can be quickly detected. With a tile-pipe condenser resort can usually be had to water-cooling to reduce excessive temperature. When only a small quantity of water is available this can be effectively applied to horizontally arranged pipe by dripping it along the top of the pipe. Pieces of moistened burlap may be placed on the pipe to distribute the water over the surface; with this arrangement the cooling is effected largely through the evaporation of the water.

Condenser systems are ordinarily operated with natural draft. When the condenser is above the furnace level, and particularly when it is placed on a hillside with a gradual ascent to the stack, natural draft will usually suffice under all conditions. Stacks of brick, glazed tile, and wood coated with pitch or similar material are used in this country. These stacks are usually of small diameter and not more than 30 to 50 feet high. A separate stack is provided for each set of condensers. European practice favors the use of a single large and considerably taller stack for all of the condenser units at a given plant. (See Pl. XXVII, *A*, *B*, *C*, p. 140.) The low temperature to which the condenser gas should be cooled makes desirable the providing of as much elevation as possible in order to obtain the maximum effect from natural draft. As the velocity of the gas stream in the condenser system is low, the difference between the interior and the atmospheric pressure is usually small. In order to avoid the loss of mercury through the escape of condenser gas, the pressure within the system, particularly near to the furnace, should be slightly less than atmospheric. A slight leakage of air into the system does not increase the gas volume enough to involve any danger of increased loss from the stack, and a considerable inward leakage is preferable to the escape of even a small amount of condenser gas.

At some plants a wooden exhauster of the type of the Guibal fan is used either continuously or to increase the draft through the system during the clean-up or when repairs are being made. The fan should be near the base of the stack or at the beginning of the flue leading to the stack. For experimental purposes the authors designed a simple exhauster of this type, consisting of a heavy cast-iron hub slotted to hold the ends of hardwood vanes. The ends of these vanes formed a dovetail corresponding to the opening in the hub, so that they were held firmly in place against the action of centrifugal force. The cast-iron hub was long enough to extend beyond the impeller housing, and consequently the shaft was not exposed to the gas within the exhauster. The wooden housing and impeller blades were thoroughly saturated with an asphalt-base paint. This exhauster showed good efficiency, and the same design could be used for larger-size blowers. The only materials exposed to the gas are the large cast-iron hub and the wooden blades and housing, and the simple construction of the impeller makes replacement of broken or worn-out blades easy. The exhauster showed under test a capacity of 600 cubic feet per minute at 0.5-inch water gauge and at 1,000 revolutions a minute. A practical trial showed that the fan running at 1,400 revolutions could handle the entire gas stream from a 50-ton rotary furnace plant.

At some plants an auxiliary fire is used to assist the draft, particularly when atmospheric conditions are unfavorable. This fire should be at or close to the base of the stack, and the arrangement at the Oat Hill plant is as good as any. The air admitted to the fire box should be carefully limited to the amount required for the combustion of the fuel, so that the maximum reheating of the stack gas will be assured.

The point at which the maximum collection of mercury is made in the condenser system will depend on the concentration of mercury vapor in the furnace gas, on the cooling efficiency of different parts of the condenser system, and on the baffling provided for collecting the condensed mercury. The method of calculating the amount of mercury vapor that condenses in any given temperature range has already been discussed. As is to be expected, the point at which the greatest amount of mercury is obtained from the condenser system is somewhat beyond the point where the maximum condensation takes place. With the old condenser system of the Scott furnace at New Idria, for example, it is calculated that about 70 per cent of the mercury should condense in that part of the condenser system where the temperature ranges from 125° to 100° C. About 30 per cent of the production was actually obtained there, and an additional 60 per cent from the temperature range of 100° to 55° C.

COLLECTING VESSELS FOR MERCURY.

Suitable vessels should be provided for receiving the mercury after it drains from the condenser system. Heavy cast-iron pots are often used for this purpose, but when acid water accompanies the mercury they are slowly attacked and have to be replaced from time to time.

If such water has to be handled a pot of acid-resisting iron, such as corrosiron or duriron, may be used to advantage. Iron pots, particularly the larger sizes that hold several tons of mercury, may be set in concrete. A concrete setting is not only a convenient method of support but helps the pot to withstand the pressure of the large amount of mercury and serves to retain the metal if the pot cracks. Satisfactory pots can also be made of a rich concrete mortar, well finished on the interior; launders of this same material are suitable for collecting the mercury at a series of points or for conveying it to a central collecting vessel. Redwood gutters, milled in one piece, give excellent service as launders, as they are not affected by the acid, and being of one piece have no cracks through which mercury can escape. Reduction of the handling of mercury to a minimum is desirable, and when the elevation of the condenser system permits, iron pipes may be used to convey the metal to the quicksilver room. If iron pipes are used, each collecting pot should be provided with a gooseneck, so that mercury alone will flow into the collecting pipe and the water will overflow from the lip of the pot into a launder leading to the settling boxes.

The settling boxes may be made of wood, preferably of slot-and-wedge construction, or of concrete. Their bottoms should be slightly inclined to insure the mercury collecting at one point, and each box should have several partitions so arranged that the water flows alternately under and over them to prevent the loss of any finely floured mercury or mercurial soot, which may tend to float on the surface of the water. The question of loss of mercury in the water flowing away from the condenser system has been discussed in another paper by the authors.[81] Data gathered since 1918 on the nature and magnitude of the losses agree with those already published. When considerable water escapes from the condenser system, and particularly when water is used freely for the periodic clean-up, adequate settling boxes should be provided to prevent the loss of mercury in suspension.

The usual practice is to clean the condenser system, or at least that part of it in which the larger part of the mercury condenses, once a month. When the condenser system is arranged, as at the Oat Hill plant, so that one section may be shut down at a time without interrupting plant operation, cleaning the condenser is a comparatively simple matter, and there is little danger of the workmen being exposed to mercury-laden gases. With the older type of condensers the operation of the furnace is usually suspended during the

[81] Duschak, L. H., and Schuette, C. N., Fume and other losses in condensing quicksilver from furnace gases: Tech. Paper 96, Bureau of Mines, 1918, 29 pp.; Min. and Sci. Press, vol. 117, 1918, p. 315.

clean-up, and the exhauster is started or, if used regularly, is speeded up so that a strong indraft will prevail through the doors that are opened to permit cleaning. Under these conditions it is desirable to get the cleaning up done as soon as possible. To insure rapid work and no delays, the men employed should be thoroughly familiar with the operation and should be properly instructed in advance. The special buckets and tools that are to be used should be inspected and everything made ready before the actual clean-up begins. The clean-up starts at the furnace end of the condenser system, the most difficult and dangerous work being at that end. In progressing from the furnace the work becomes easier as the men tire; also the last part of the condenser system is not cleaned as often as the first part, and the clean-up can be stopped where convenient. The soot is removed as far as possible in a dry condition by suitable hoes, and often a jet of water is used for the final cleansing. When water is to be used the settling boxes to which the water flows should be inspected just before the clean-up begins to make sure that the spaces under the alternate partitions are unobstructed.

Mercury and mercurial soot may be safely handled in iron or heavy enamelware buckets, but no other metal should come in contact with either. The mercury may be stored in iron or concrete containers; for the soot, tight wooden boxes may also be used.

The amount and composition of the soot obtained at different plants varies widely, as already pointed out (see Table 5) according to the nature of the ore and the fuel and the skill employed in running the furnace. When the ore contains considerable pyrite, or when carbonaceous soot and tarry matter are found in the furnace, 50 per cent or more of the mercury produced may be originally collected in the form of soot. On the other hand, the Scott furnace with the new condenser system at the New Idria plant yielded in a month practically no soot and only a few bucketfuls of floured mercury when producing 600 to 700 flasks of mercury per month. Owing to the continual disintegration of the condenser material under the action of acid the soot obtained from brick or stone condensers often contains 10 per cent or more of mineral matter.

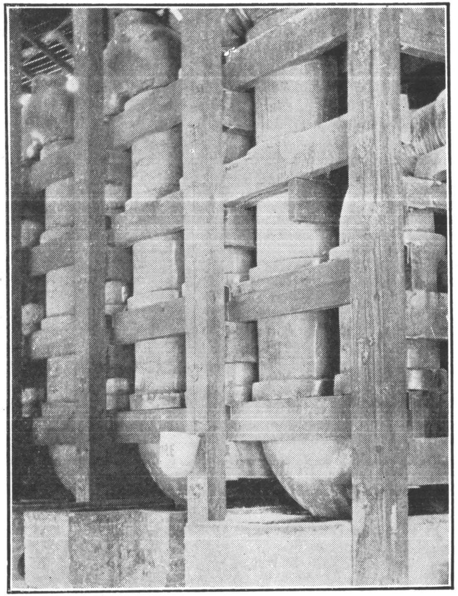
TREATMENT OF MERCURIAL SOOT.

The formation and composition of mercurial soot have already been discussed, and some typical analyses are given in Table 5. With respect to treatment, soot may be divided into two classes, one consisting essentially of globules of free mercury that are prevented from coalescing by the presence of foreign matter, such as ore dust and carbonaceous soot or mineral matter from the condenser system; and another, soot in which part of the mercury is held in chemical combination, as the sulphide or sulphate. The problem of treating the first type of soot is essentially a mechanical one, the object being to cause the mercury globules to coalesce and flow away from the foreign material.

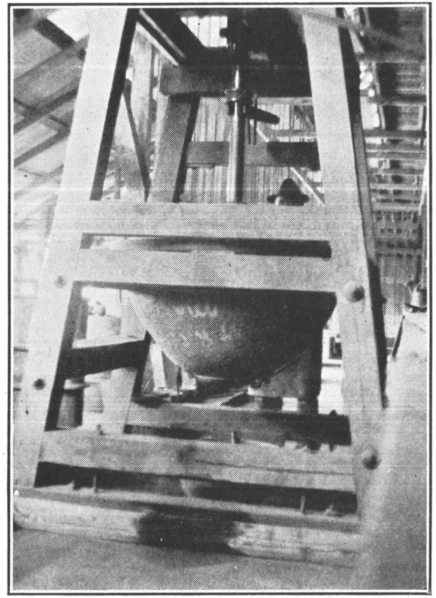
Treatment of the first type of soot depends upon mechanical agitation, which may be effected either by raking or hoeing by hand on an inclined surface, or by a mechanical device, such as that illustrated in Plate XXV, *B* (p. 140). When the soot is moist, and particularly when acid is present, the addition of a little burned lime often facilitates the separation of the mercury. As the addition of lime causes a considerable increase in the volume of soot, the amount should be carefully regulated to the minimum which will produce the desired result. Warming of the soot and alternate wetting and drying also promote this separation. So-called steam pans are sometimes used. These are made of cast iron with a steam jacket below. The pan is inclined so that the mercury will drain to one corner, while a gooseneck is provided for the escape of the mercury as it separates. This type of pan is convenient for warming the soot, and for the alternate moistening and drying mentioned above. An iron pan of this sort may also be heated by hot air from a small fire box, but in this case care must be taken to prevent heating to a point where the mercury will begin to volatilize and expose the workmen to the danger of mercurial poisoning. A well-finished inclined concrete surface may also be used for this purpose, but when the amount of soot is large the use of a mechanical device saves labor. The soot machine here illustrated consists of a large cast-iron pot with a horizontal propeller that lifts the soot. A propeller with two blades was found to give better results than one with three, and a fairly steep pitch is desirable. The propeller was driven at about 60 revolutions per minute by a 5-horsepower oil engine. The opening at the bottom of the pot is covered by a saucer-shaped iron door provided with a lip for the overflow of the mercury. When the treatment of a given batch of soot is completed, this door is swung downward and the residue discharged through the bottom.

The residue from mechanical treatment, which may still contain as much as 20 per cent of free mercury, is either returned to the furnace or set aside for treatment in a retort. When soot of the first type contains only a small amount of mercury, which may be the case when considerable ore dust passes over into the condenser system from the furnace, or when the quantity of such soot is small, the material is usually returned directly to the furnace.

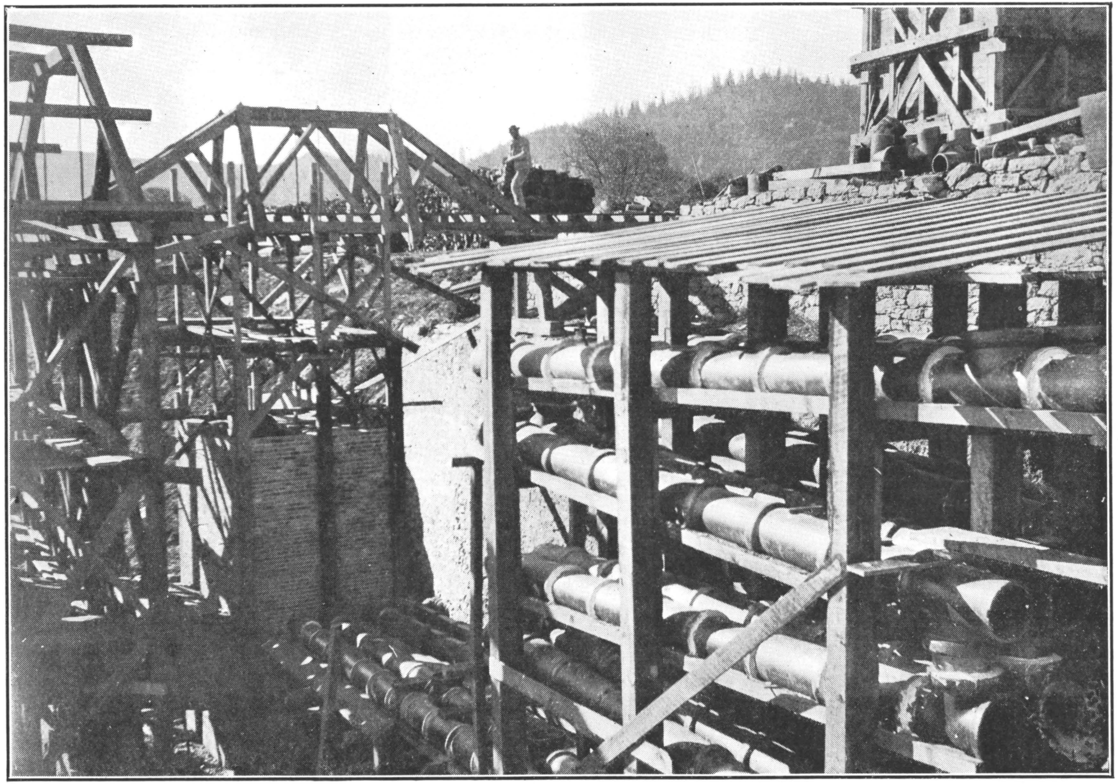
With the installation of the rotary furnaces and tank-and-pipe condenser systems at New Idria, another type of apparatus was installed for treating the dust washed down by the water sprays, and also for handling the soot that was flushed down in cleaning up the



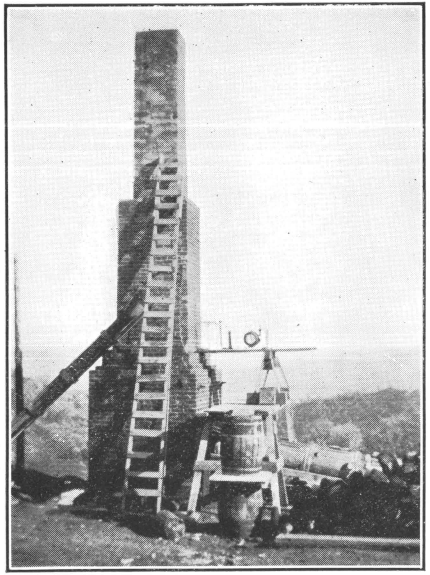
A. CERMAK CONDENSERS FOR THE ROTARY FURNACE AT THE CHISOS PLANT, TEXAS, SHOWING THE WATER SEAL AND MERCURY COLLECTING BOXES AT THE BOTTOM



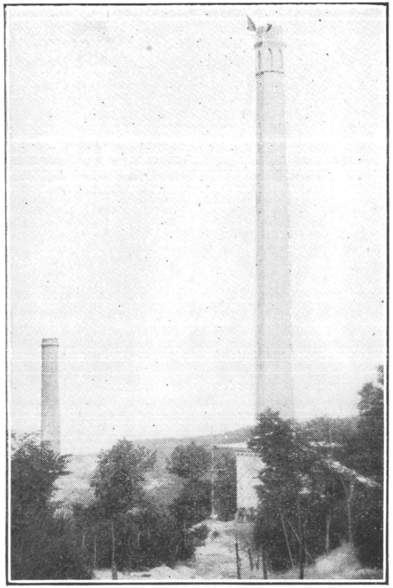
B. MACHINE FOR TREATING MERCURIAL SOOT AT THE CHISOS PLANT, TEXAS



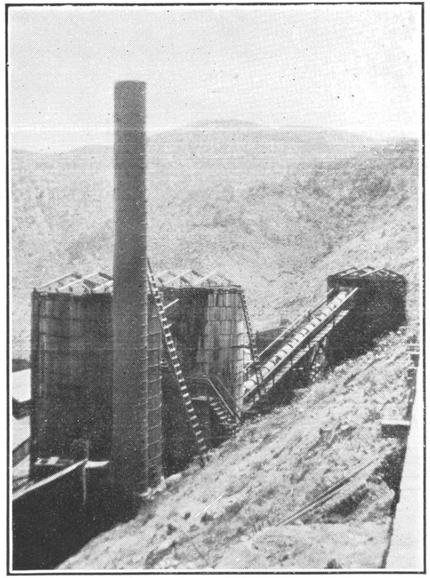
TILE-PIPE CONDENSER AT THE OAT HILL PLANT, SHOWING (LEFT) BRICK EXIT PIPE FROM SCOTT FURNACE AND SMALL BRICK CHAMBER



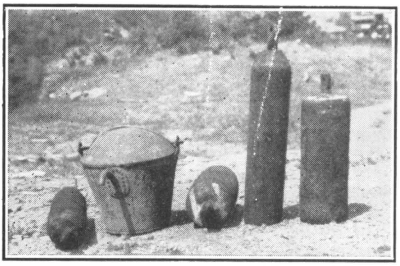
A. STACK OF THE OAT HILL PLANT, CALIFORNIA



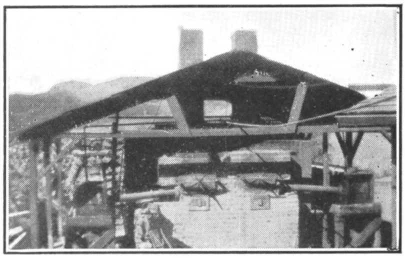
B. STACK OF THE ABBADIA SAN SALVATORE PLANT, ITALY



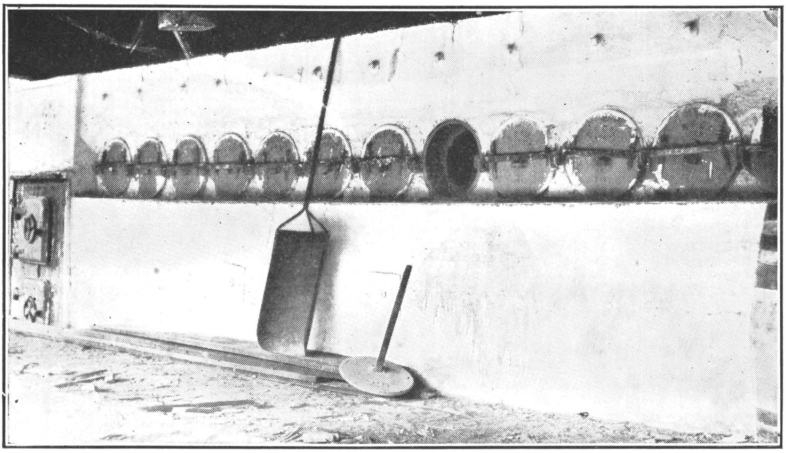
C. STACK OF THE NEW IDRIA NO. 1 ROTARY PLANT, CALIFORNIA



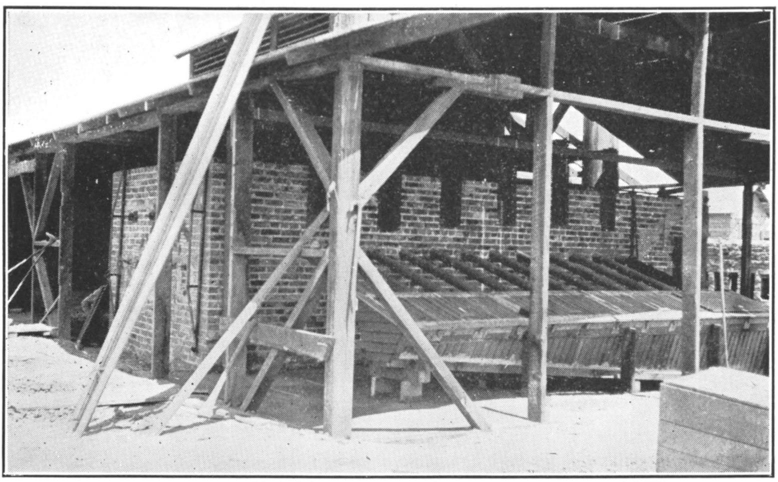
A. TWO TYPES OF QUICKSILVER FLASKS AND WEIGHING BUCKET



B. TWO D RETORTS AND WATER-COOLED CONDENSER FOR TREATING MERCURIAL SOOT



C. FRONT VIEW OF JOHNSON-McKAY 12-PIPE RETORT



D. REAR VIEW OF A CONDENSER; SHOWS PIPES AND COLLECTING TROUGH condenser system. The condenser product was flushed into a concrete pot, whence the mercury flowed out through a gooseneck in the bottom; the water carrying soot in suspension overflowed into a launder, whence it was conveyed to a tank 14 feet in diameter and 4 feet deep, in which was a revolving shaft with arms for agitation. This tank had an outlet at the bottom and a second one about 3 feet from the bottom. The metallic mercury and heavy sludge settled and at the end of a run were taken out in buckets and treated on the steam-heated soot pans. The overflow from the upper opening passed to an Overstrom concentrating table, which recovered some free mercury; the middling product returned to the furnace. Soot, with the addition of enough lime to neutralize the free acid, has also been treated in a small ball mill, the discharge from which passed to a Senn pan-motion amalgamator. The ball mill was effective in separating the mercury, but the amalgamator was not found particularly satisfactory and was discarded.

In treating soot of the second type a certain amount of free mercury can usually be obtained by one of the procedures outlined above, but the residue containing not only free mercury but mercury compounds may still hold the major portion of the mercury collected by the condenser. This material is usually treated in a D retort, such as illustrated in Plate XXVIII, *B* (p. 141). The practice of returning this material to the furnace is not usually advisable, as the foreign substances which were responsible for the original formation of the soot are thus returned to the furnace cycle. Another objection lies in the delay involved in recirculating the mercury through the furnace and condenser systems. In treating soot of this type sufficient lime should be added to take care of any free acid and the sulphur compounds present. Wood ashes may also be used for this purpose. When retorting is practiced it is customary to store the soot or residue from mechanical treatment until sufficient has accumulated for a retorting campaign.

PREPARING QUICKSILVER FOR SHIPMENT.

The clean mercury recovered from the condenser system and from treatment of soot and residues is usually collected in large cast-iron pots or sheet-iron tanks. It should be bottled for storage or shipment as quickly as possible. A suitable arrangement of iron pipes makes it possible to convey the mercury automatically from the various points of production about the plant to the collecting pots in the quicksilver room. Ordinarily the mercury requires no preliminary treatment prior to bottling for shipment. The surface of pure mercury should remain bright in contact with the air. Formation of a film or scum shows that the mercury contains an amalgam of some base metal, such as zinc, which oxidizes slowly on exposure to air. Such contamination can be removed to a considerable degree by agitating the mercury in contact with air. This agitation is easily done by bubbling air through the mercury. The oxide of the base metal will collect on the surface as a scum, entraining considerable mercury, and this material can be easily skimmed off and treated as ordinary soot. The practice of filtering the mercury through chamois skin has been practiced at some plants, but this is not necessary if the mercury storage pots or tanks are provided with bottom outlets, as no dirt can remain within the mass of mercury. At New Idria an iron ring was floated in the collecting pots and the scum and water in the ring dipped out, after which the ring formed the rim of a clean surface of mercury.

As mentioned on page 1, the standard container for quicksilver in the United States is a flask holding 75 pounds avoirdupois. Three types of flasks have been used in this country, the oldest being of wrought iron and a later one of pressed steel. Recently a lighter flask has been made from sections of mild-steel tubing with ends autogenously or electrically welded. The flask is closed by a screw plug, and a little red lead in oil is used to make the seal mercury tight. Plate XXVIII, *A* (p. 141), shows the two types of flasks and a container used for weighing out mercury.

In this country a special bucket or scale pan (Pl. XXVIII, *A*) is ordinarily used in weighing out the mercury. For most convenient weighing the mercury-storage vessels should be placed at a slight elevation so that the metal can be tapped off directly into the counterpoised vessel on the scales. A volumetric device for measuring the mercury has been described by Sterner-Rainer.[82] The writers[83] have been informed that this device has been perfected by the addition of a compensator to take care of temperature variations, so that it is simple and accurate in operation.

The marketing of quicksilver is described by H. W. Gould in a paper printed in the Engineering and Mining Journal-Press under date of December 9, 1922.

RETORT FURNACES.

INTRODUCTION.

The retort furnace was the earliest device used for the extraction of quicksilver from the ore, and the use of retorts for this purpose

[82] Sterner-Rainer, Roland, Der derzeitige Stand des Quecksilberhüttenwesens in Europa: Oest. Ztschr., Berg.-Hüt., Jahrg. 62, 1914, pp. 529–563. Translation by C. N. Schuette, Production of quicksilver in Europe: Chem. and Met. Eng., vol. 19, 1918, pp. 721–727; vol. 20, 1919, pp. 32–35 and 82–84.

[83] Private communication from H. W. Gould, who has visited the European quicksilver mines.

has continued to the present day. According to Brelich[84] the entire quicksilver production of China is obtained by means of retorts, and, in general, they are widely used where methods are primitive.

In America the retort furnace is used for two purposes, namely, the treatment of selected ore and the recovery of quicksilver from the intermediate products of the reduction works, such as mercurial soot. According to Ransome[85] 8 per cent (2,868 flasks) of our domestic quicksilver production in 1917 was obtained by the direct treatment of ore in retorts. In view of the high labor and fuel cost involved in this operation it is somewhat surprising to find that retorts are still so extensively used for ore treatment. It has been rather common practice in this country to use the retort furnace in the early stages of the development of a prospect. This has the advantage of furnishing an early source of income for use in further development and in equipping the mine and the reduction works. It is, however, to be regarded as a shortsighted policy, and the ultimate profits would usually be greater if the mine could at the outset be developed far enough to justify the erection of a furnace.

A general discussion of retort furnace practice has been given by Egleston,[86] Booth, Forstner, and Bradley.

DESIGN AND CONSTRUCTION OF RETORT.

Quicksilver retorts are generally made of cast iron, having either a circular or **D**-shaped cross section. The retorts are mounted singly or several together in a furnace, so arranged that the hot gases from the fire box will circulate around all sides of the retort. It is common practice to protect the retort by fire brick or fire-clay tile, so that the flames from the fire box will not impinge directly upon the cast iron. The retort should be so supported that it is exposed to little mechanical strain.

Three types of retort furnace are now in use, namely, a furnace holding from one to three large **D** retorts, a furnace with from 1 to 12 circular pipe retorts, and a furnace having from 1 to 8 large circular retorts mounted at an angle. The first two types are used for both soot and ore, and the last type for ore only.

A typical installation of a double **D** retort furnace is illustrated in Plate XXVIII, *B* (p. 141), which shows the general arrangement,

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[84] Bralich, Henry, Chinese methods of mining quicksilver: Min. and Sci. Press, vol. 90, 1905, pp. 386–406.

[85] Ransome, F. L., Quicksilver in 1917: Mineral Resources of the United States, U. S. Geol. Survey, 1917, p. 373.

[86] Egleston, Thos., The metallurgy of silver, gold, and mercury in the United States. Vol. 2, 1890, pp. 808–813; Booth, F. J., The reduction of quicksilver ore: Min. and Sci. Press, vol. 93, 1906, p. 570; Forstner, Wm., Quicksilver resources of California: California State Min. Bur. Bull. 27, 1903, pp. 197–205; Bradley, W. W., Quicksilver resources of California: California State Min. Bur. Bull. 78, 1918, pp. 210–218.

and in Plate XXIX, which shows the sections of the D retort at Oceanic mine. The retorts are supported throughout their entire length by fire-clay tile. Two stacks are provided, so that by means of the dampers the heat of each retort can be closely controlled.

The flame from the fire box passes back through a central flue, thence through passages in the end wall to the flues under the retorts. The use of fire-clay tile to protect the closed end of the retort from the sweep of the hot gases adds to the life of the retort.

The particular installation here shown is used for treating soot. It has a capacity of from 750 to 1,000 pounds of a mixture of soot and lime per 24 hours, with a fuel consumption of one-half to three-fourths cord of wood. The retort charge is put in shallow iron pans, of which there are two for each retort, and the retorts are charged every 12 hours. The condenser system shown in the photograph consisted of two wooden boxes with projecting sides and ends. A shallow layer of water covered the top of each box; it served to keep the boxes tight and aided cooling.

This retort furnace would probably have a capacity of one-half to three-fourths ton of ore in 24 hours, with about the same fuel consumption mentioned above. Retorts of this type, when well mounted in the way illustrated and handled with reasonable care, will last a long time. A life of two years is not unusual.

A typical installation of a 12-pipe retort, frequently known as the Johnson-McKay retort, is shown in Plate XXVIII, *C* and *D* (p. 141).

The individual pipes are 12 inches in diameter and about 6½ feet long. The condenser pipes are 3 inches in diameter and about 8 feet long, being set into the retorts with rust joints. The retort pipes are closed by an inner and outer lid, both of which are luted with a thin clay or mud mixture. The use of the inner lid prevents the mercury vapor from reaching the outer lid and condensing, as it would be likely to do otherwise. This type of retort is charged with a large scoop, shown in the photograph, and discharged with a hoe.

A 10-pipe retort of this type used for soot treatment had a capacity of about 850 pounds in 24 hours, with a fuel consumption of half a cord of wood (about 1 ton). This retort was charged only on day shift, and the fires were banked at night. It could treat probably double this amount of soot with about twice the fuel consumption. When used for ore treatment, a bench of 12 pipes will handle from 1½ to 2 tons of ore in 24 hours, with a fuel consumption of from 1 to 1½ cords of wood. In this case one pipe is charged every hour, two complete cycles being made in 24 hours.

An installation of the large inclined retorts has been shown in Plate XXII, *A* (p. 120). These retorts are used for treating ore, and a battery of 8 pipes showed a capacity of 12 tons in 24 hours with a consumption of 3.5 cords of wood. The individual pipes are charged every 6 to 8 hours; as only half of the contents is removed before each charging, continuous operation is approached. Another interesting feature is the admission of a controlled amount of air in the lower end of each pipe. This increase in the gas volume leaving the retorts makes necessary the use of a more extensive condenser system than is required for the other types. The gas stream from the individual pipes is collected in a manifold leading to the condensers; the design of the latter is well shown in the photograph.

When the material charged to a retort contains much sulphur, lime must be added; otherwise mercuric sulphide, and not free mercury, will be deposited in the condenser. The use of lime and other substances for this purpose has been discussed previously in connection with the roasting of quicksilver ores.

It will be evident that the fuel consumption of a retort furnace is much higher per unit of ore treated than for direct-fired furnaces. The cost of construction and operation per unit capacity is also considerably higher, and as previously indicated, the use of retorts for material that can be otherwise treated is not economical practice. In treating high-grade material, the retort has the advantage that the volume of gas evolved is very small, which consequently permits the use of a very simple type of condenser. Furthermore, the retort and condenser can be made gas tight, and thus obviate any danger of leakage.

The operation of a battery of retorts involves a fair amount of disagreeable manual labor, and the operator is exposed to greater danger of mercurial poisoning than he is at most other furnaces. In spite of its disadvantages in the treatment of ore, the retort will doubtless continue in use as in the past in connection with the development of prospects. As an adjunct to furnace plants, the retort holds a regular place in quicksilver metallurgy, but, as indicated in discussion of the soot problem, the production of soot can, no doubt, be considerably decreased by approaching the problem in the right way. Development in that direction will doubtless result in a diminished use of retorts, even for this latter purpose.

CONCENTRATION AND WET TREATMENT OF QUICKSILVER ORES.

INTRODUCTION.

It was natural that the application of methods for the concentration and wet treatment of ores of copper, zinc, etc., should stimulate interest in applying these processes to the winning of quicksilver. Attempts to concentrate cinnabar ore were made as early as 1871, and proposals to treat quicksilver ores by a hydrometallurgical process date back to nearly this same time. A comprehensive review of these subjects has been given by Bradley.[87] It may be stated in general that the attention given to these subjects, and particularly to concentration, is quite out of proportion to the results that have been so far achieved, and if the subject had been carefully examined from an economic standpoint, in advance of practical attempts to treat quicksilver ores by milling operations, it is likely that a good deal of time and money might have been saved.

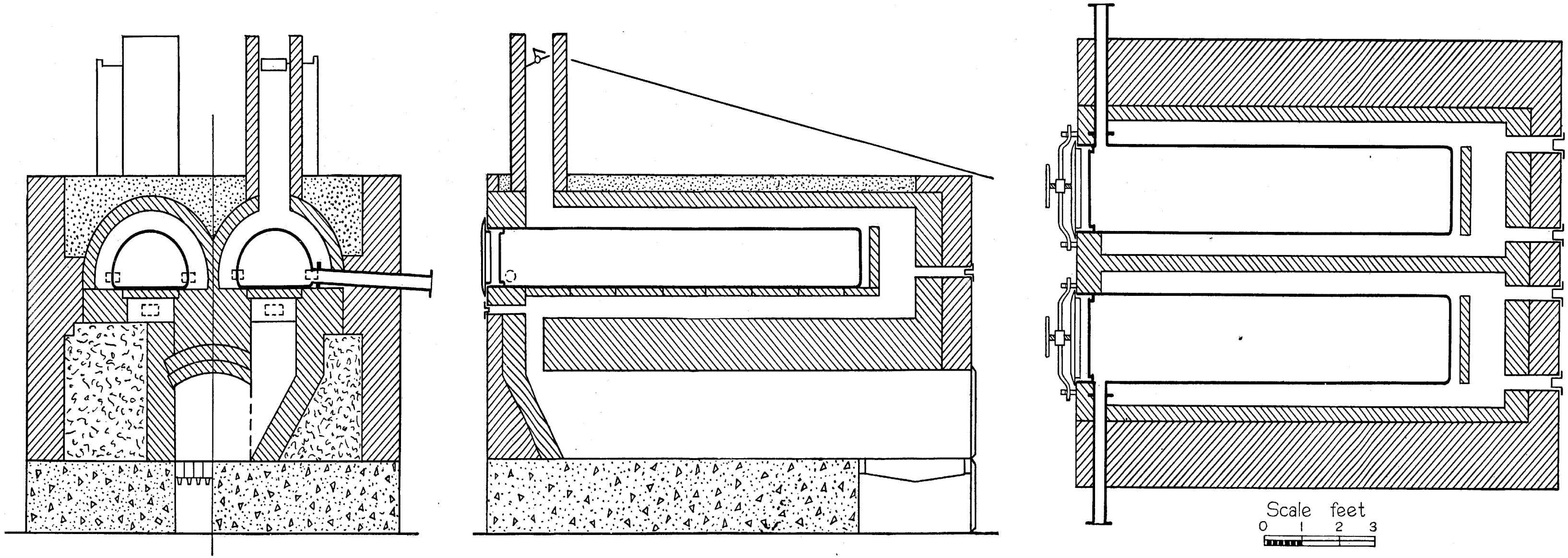
CONCENTRATION OF QUICKSILVER ORES.

Comparatively little attention had been given to concentration prior to the outbreak of the World War, but following this the subject of concentrating cinnabar was actively discussed. It was thought that concentration might be applied to advantage in securing a rapid increase in quicksilver production in connection with regular furnace-plant operation. A number of mills were constructed in California, but most of them were poorly designed, hastily erected, and not well operated. The majority of the attempts were short lived and were unsuccessful commercially. These various undertakings are reviewed in detail by Bradley.[87]

Several plants were visited by the authors of this bulletin in company with Thomas Varley, ore-dressing engineer for the Bureau of Mines, in charge of its experiment station at Salt Lake City, Utah. It was observed that, in general, the mechanical condition of the concentrating machinery was poor, that the material was not properly classified before being passed to the tables, and that the types of concentrating tables in use had not been well selected. Correction of these defects should have materially improved results, but would not have altered the fundamental economic situation.

The only large mill that was operated for any extended period was that at the New Idria mine, San Benito County, Calif. This mill was designed to treat about 150 tons of ore per day, and the actual daily tonnage of the original mill ranged from 125 to 200. The ore going to the mill consisted of minus 1-inch material with a relatively large proportion of fines. It was fed directly to a 7 by 6 foot Allis-Chalmers ball mill with a screen discharge which acted as a trap for coarse material. The ball-mill discharge was passed without desliming to hydraulic classifiers, and thence to six double-decked Deister-Overstrom tables. These tables yielded a finished concentrate carrying about 10 per cent of mercury, a middling product which was returned to the ball mill for regrinding, and a tailing which was discarded. Accurate tonnage and assay figures are not

[87] Bradley, W. W., Quicksilver resources of California: California State Min. Bur. Bull. 78, 1918, pp. 286–352.



SECTIONS OF D RETORT, OCEANIC MINE, CALIFORNIA available, but it is doubtful whether the recovery exceeded 60 per cent. One of the principal sources of loss was the slime. Cinnabar slimes readily, but no provision was made in the mill for recovering the mineral from the fine material which passed over the tables and went into the tailings. (The mill was later improved in certain respects: A trommel with $\frac{1}{8}$-inch openings was placed in the circuit ahead of the ball mill, the minus $\frac{1}{8}$-inch material being passed to a Richards pulsator jig, which recovered the coarse cinnabar as a hutch product, thus preventing it from being slimed in the mill. The jig tailing, together with the oversize from the trommel, passed to the ball mill and thence to a six-spigot Richards pulsator classifier. The upper deck of the Deister-Overstrom tables was removed. With these changes the mill treated about 300 tons of ore in 24 hours, yielding 10 tons of concentrate averaging 11 per cent mercury. The loss in the tailing was somewhat reduced, and the recovery increased to about 66 per cent.)

Obviously this low recovery as compared with what can be accomplished by direct furnace treatment would render the milling operation uneconomical under ordinary conditions. It did, however, serve the purpose of materially increasing the quicksilver output during a period of high prices and when quicksilver was much needed for war purposes.

A small mill at the Oceanic mine, San Luis Obispo County, Calif., described by Heberlein,[88] produced about the best results which have been obtained in quicksilver concentration. The ore, which carried approximately 0.3 per cent mercury, yielded a concentrate carrying 5 per cent mercury, with a recovery of 80 per cent. The operating cost is given as 50 cents per ton. A Huntington mill was used for fine grinding in order to reduce the sliming of the cinnabar to a minimum.

A number of tests were made during the war period on the application of flotation to quicksilver ore, but practical development in this direction was largely prevented by the complicated situation with regard to flotation patents and the royalties that were being asked. Bradley,[89] Stowell and Coghill, and others have shown that cinnabar can be readily floated, and the use of flotation in connection with the milling operations referred to above would undoubtedly have materially increased recoveries.

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[88] Heberlein, C. A., The mining and reduction of quicksilver ore at Oceanic mine, Cambria, Calif.: Trans. Am. Inst. Min. Eng., vol. 51, 1915, p. 110.

[89] Bradley, W. W., Work cited, pp. 300–320; Stowell, E. G., and Coghill, Will H., Experimental flotation of low-grade quicksilver ore: Min. and Sci Press, vol. 120, 1920, p. 117.

CONCENTRATION EXPERIMENTS.

In order to obtain first-hand information as to the results which might be expected in the concentration of quicksilver ores on tables and by flotation, samples of certain ores were sent to the Bureau of Mines experiment station at Salt Lake City, Utah, where tests were conducted by Thomas Varley. The laboratory equipment used had been especially designed with a view to obtaining test results which could be closely duplicated in actual practice. The following description of a few representative tests is given by Mr. Varley:

CONCENTRATION AND FLOTATION TESTS OF NEW IDRIA ORE.

Ore from the New Idria mine, San Benito County, Calif., may be taken as representing a clean siliceous ore. The cinnabar occurs along fissures, and in general is easily released by crushing.

The ore, which assayed 0.76 per cent mercury, was crushed to 20-mesh size and classified as shown below:

Classification of ore.

Material.	Weight, grams.	Assay, per cent Hg.
+100	3,720	0.73
−100	1,250	.84
	4,970	

The +100-mesh product was passed over a Deister-Overstrom concentrating table, with results as follows:

Concentation tests of + 100-mesh product.

	Weight.	Hg.	Fe.	SiO_2.	Hg.	Total Hg.
	Grams.	*Per cent.*	*Per cent.*	*Per cent.*	*Grams.*	*Per cent.*
+100 table concentrates	170	11.10	13.90	48.20	19.0	70
+100 table tailings	3,530	.23			8.2	30
Totals and averages	3,700	.73			27.2	100

The table tailings from the above test were ground to −100 mesh and treated by flotation in a Janney laboratory machine, using the equivalent of 1 pound of reconstructed pine oil and 2 pounds of sulphuric acid per ton:

Flotation of table tailings.

	Weight.	Hg.	Hg.	Total Hg.
	Grams.	*Per cent.*	*Grams.*	*Per cent.*
Flotation concentrates	75	5.40	4.05	48.3
Cleaner tailings	1,075	.14	1.50	100
Reject tailings	2,370	.12	2.82	33.7
Totals and averages	3,520	.23	8.37	100.0

The —100-mesh product was treated by flotation as in the test above. Results were as follows:

Flotation of —100-mesh product from fresh ore.

	Weight.	Hg.	Fe.	Insoluble.	Hg.	Total Hg.
	Grams.	*Per cent.*	*Per cent.*	*Per cent.*	*Grams.*	*Per cent.*
Flotation concentrates	32.5	26.30	13.90	27.60	8.60	81.3
Cleaner tailings	115	.67			.80	7.6
Reject tailings	1,100	.11			1.08	11.1
Totals and averages	1,247.5	.84			10.48	100

Calculation of extraction.

	Hg., grams.
+100-mesh	27.20
—100-mesh	10.48
	37.68
+100 table concentrates	19.00
Table tailings flotation concentrates	4.05
—100 flotation concentrates	8.60
	31.65

$$\frac{31.65}{37.68}=84.1 \text{ per cent extraction.}$$

The treatment of this same ore by straight flotation yielded the following result: New Idria ore, ground to —100 mesh and treated in a Janney laboratory flotation machine using the equivalent of 1 pound of reconstructed pine oil and 2 pounds of sulphuric acid per ton.

Straight flotation tests of New Idria ore.

	Weight.	Hg.	Fe.	SiO_2.	Hg.	Total Hg.
	Grams.	*Per cent.*	*Per cent.*	*Per cent.*	*Grams.*	*Per cent.*
Flotation concentrates	14.5	21.40	12.5	34.3	3.10	87.6 } 95.3
Cleaner tailings	70.0	0.38			0.27	7.7 }
Reject tailings	415.0	0.04			0.17	4.7
Totals and averages	499.5	0.76			3.54	100.0

There is 87.6 per cent of the mercury extracted in concentrates, and on the assumption that 70 per cent of mercury in the cleaner tailings could be recovered by regrinding and re-treating, the extraction would equal 93 per cent.

CONCENTRATION AND FLOTATION TESTS OF GOLDBANKS ORE.

The second ore tested was from the Goldbanks mine near Winnemucca, Nev. This is a rather unusual type of ore, consisting of a breccia recemented with silica, in which a good deal of the cinnabar is contained in the cementing silica. The ore is hard and appeared to present a particularly difficult problem in concentration owing to the difficulty of releasing the cinnabar and the danger of sliming it during fine grinding.

The ore, which assayed 3.78 per cent mercury, was crushed through 20 mesh and screened on a 100-mesh screen.

Classification of ore.

Material.	Weight.	Assay, Hg.
	Grams.	*Per cent.*
+100	7,705	3.40
−100	2,270	5.33
	9,975	

The +100-mesh product was treated on a Deister-Overstrom concentrating table, as follows:

Concentration of +100-mesh product.

Material.	Weight.	Hg.	SiO_2.	Hg.	Total Hg.
	Grams.	*Per cent.*	*Per cent.*	*Grams.*	*Per cent.*
Table concentrates	360	24.90	66.50	90.0	36.0
Table tailings	7,150	2.22	----------	159.0	64.0
Totals and averages	7,510	3.40	----------	249.0	100.0

The table tailings from the above were crushed and treated in a small Janney flotation machine with the equivalent of 1.5 pounds of reconstructed pine oil and 2 pounds of sulphuric acid per ton. The results are shown below:

Flotation of table tailings.

Material.	Weight.	Hg.	SiO_2.	Hg.	Total Hg.
	Grams.	*Per cent.*	*Per cent.*	*Grams.*	*Per cent.*
Flotation concentrates	386	29. 20	59. 2	112. 3	70. 4
Cleaner tailings	715	2. 71		19. 2	11. 9
Reject tailings	6, 034	. 47		28. 3	17. 7
Totals and averages	7, 135	2. 22		159. 8	100. 0

The —100-mesh product was treated by flotation direct, using the same reagents as in above test, as follows:

Flotation of —100-mesh product from fresh ore.

Material.	Weight.	Hg.	SiO_2.	Hg. in sample.	Total Hg.
	Grams.	*Per cent.*	*Per cent.*	*Grams.*	*Per cent.*
—100 flotation concentrates	357	30. 90	54. 50	110. 0	91. 0
—100 cleaner tailings	783	0. 89		7. 0	5. 8
—100 reject tailings	1, 123	0. 36		4. 0	3. 2
Totals and averages	2, 263	5. 33		121. 0	100. 0

Calculation of extraction.

	Hg. grams.
Table concentrates	90. 0
Table tailings, flotation concentrates	112. 3
—100 flotation concentrates	110. 0
	312. 3
Heads	378. 0
In tailings and cleaner tailings	65. 7

$$\frac{312.3}{378} = 82.6 \text{ per cent extraction.}$$

Assay of combined concentrates=28.2 per cent Hg.

On the assumption that 50 per cent of the mercury could be recovered by retreating the cleaner tailings, the percentage of extraction would be 86 per cent.

This ore was also treated by straight flotation with the following results. Goldbanks ore was crushed to —100 mesh and treated in a

Janney laboratory flotation machine with the equivalent of $1\frac{1}{2}$ pounds of reconstructed pine oil and 2 pounds of sulphuric acid per ton of ore. The results are given below:

Straight flotation tests of Goldbanks ore.

Material.	Weight.	Hg.	Hg.	Total Hg.
	Grams.	*Per cent.*	*Grams.*	*Per cent.*
Flotation concentrates	53	31.30	16.55	84.8
Cleaner tailings	162	1.06	1.72	8.8
Reject tailings	285	0.44	1.25	6.4
Totals and averages	500	3.78	19.52	100.0

There was 84.8 per cent of the Hg extracted in concentrates, and on the assumption that 65 per cent is extracted by re-treating the cleaner tailings, the percentage of extraction would equal 90.5 per cent.

Goldbanks ore was crushed to —150 mesh and treated by flotation as in the above test.

Flotation of —150-mesh product from fresh ore.

Material.	Weight.	Hg.	Hg.	Total Hg.
	Grams.	*Per cent.*	*Grams.*	*Per cent..*
Flotation concentrates	52.5	30.60	16.05	82.1
Cleaner tailings	153.0	1.36	2.40	12.3
Reject tailings	278.0	.40	1.10	5.6
Totals and averages	483.5	3.78	19.55	100.0

The extraction was 82.1 per cent of the Hg in the concentrates, and on the assumption that 65 per cent of the Hg in the cleaner tailings could be recovered by re-treatment, the extraction would equal 90.1 per cent. Finer crushing does not seem to improve the extraction.

CONCENTRATION TESTS OF SULPHUR BANK ORE.

A third type of ore tested was from the Sulphur Bank mine, Lake County, Calif. This ore is unique among quicksilver ores in that it contains free sulphur and sulphuric acid. The cinnabar exists in a very finely divided condition mixed with sulphur, also with the product resulting from the action of the acid on the country rock. The free sulphur content is variable, and in the sample tested amounted to about 20 per cent. Attempts to concentrate this ore by flotation were unsuccessful owing to the presence of the free sulphur, and no results worth reporting in detail were obtained, but the results obtained by table concentration are given below.

The ore was crushed to —20 mesh, and divided into two sizes for table concentration, as follows:

Classification of ore.

Material.	Weight.	Assay.
	Grams.	*Per cent.*
+80	1,778	
−80	1,258	
	3,036	0.27

The products were tabled separately on a Deister-Overstrom table with the following result:

Concentration of +80 and —80 mesh products.

Material.	Weight.	Hg.	Hg.	Total Hg.
	Grams.	*Per cent.*	*Grams.*	*Per cent.*
+80 table concentrates	85	3.50	3.00	66.5
+80 table tailings	1,685	.09	1.51	33.5
Totals and averages	1,770	.255	4.51	100.0
−80 table concentrates	33.5	8.60	2.88	74.0
−80 table middlings	263.00	.17	.45	11.3
−80 table tailings	940.0	.06	.56	14.7
Totals and averages	1,236.5	.313	3.89	100.0

Calculation of extraction.

	Grams Hg.
+80 table concentrates	3.00
—80 table concentrates	2.88
	5.88

Total grams in heads=8.40.

$$\frac{5.88}{8.40}=70 \text{ per cent extraction.}$$

Assay of combined concentrate=5.00 per cent Hg.

GENERAL DISCUSSION OF CONCENTRATION.

The tests just given show that as to technical details reasonably good results can be obtained by the use of flotation or a combination of gravity, concentration, and flotation. The results with the Goldbanks ore, which represents an exceptionally "tight" ore with a hard gangue, were better than might have been expected. The Sulphur Bank ore is exceptional, and no similar deposit of any importance is known.

No experimental data are available in regard to the concentration of a cinnabar ore high in pyrite, and it is an open question what results would be obtained by differential flotation.

It is evident from the experimental results quoted that a properly designed and operated mill would yield considerably better recoveries than those actually realized in the large-scale operations described above. In spite of this, it is extremely doubtful whether there is much of a field for concentration in quicksilver metallurgy. One obvious disadvantage is that the mill product requires further treatment before quicksilver is obtained. A jig or table concentrate can be readily dewatered, dried, and treated either in a retort or by incorporation in a furnace charge, as was the practice at the New Idria plant. A flotation concentrate, on the other hand, is likely to be difficult to handle. At New Almaden, where flotation was practiced for a short time, the concentrate was like mud, and gave considerable trouble in drying and subsequent treatment.

Viewed broadly, the simplicity of the furnace treatment of quicksilver ore and its low cost, combined with the fact that a finished marketable product is usually obtained in a single operation, leave little opportunity for the economical application of concentration. In other branches of metallurgy the successful application of flotation to the treatment of low-grade ores has been made possible by the very large available tonnage of this class of ore, which has in turn made it possible to operate mills with a capacity of several thousand tons of ore a day. If corresponding tonnages of low-grade quicksilver ore were available, there would be more reason for considering the application of concentration. A number of hypothetical cases have been studied by the authors, and only one was found in which the combination of a milling and furnace plant showed a larger profit than that which could be obtained with furnace equipment alone. The assumed conditions leading to this result were similar to those existing during the war—an exceptionally high price for quicksilver during a short period. In this case the advantage of milling part of the ore lay in the fact that a mill of large capacity could be constructed at a lower unit cost than a furnace plant.

In general, for the conditions as they exist in the quicksilver industry in this country, the higher recovery that can be obtained with a furnace plant more than offsets any slight saving in the cost of ore treatment through milling. Apart from the question of the relative cost at which quicksilver can be produced by a combination of milling and furnace operations as compared with furnace treatment alone, the permanent loss of recoverable mineral in milling can not be ignored. It has been shown that a properly designed and operated furnace plant can make a surprisingly high recovery from low-grade ore, and unless it can be demonstrated that milling will yield a distinctly higher return than furnace treatment, there is no warrant for one of our important natural resources being unavoidably wasted in milling operations.

HYDROMETALLURGICAL TREATMENT OF QUICKSILVER ORES.

It has long been known that mercuric sulphide is soluble in a solution of sodium sulphide, and the idea of utilizing this fact as a basis for treating cinnabar ore dates back many years. The chemical behavior of mercuric sulphide with solutions of sodium sulphide has recently been carefully investigated by Allen and Crenshaw.[90] Data in regard to solubility of mercuric sulphide at 25° C. in solutions of sodium sulphides of various strengths are given in Abegg's Handbuch.[91]

Mulholland [92] proposed to make use of this reaction for treating a low-grade cinnabar ore, the mercury to be recovered from solution either by the addition of zinc hydrate or by the electrolysis of the sulphide solution. Thornhill [93] describes the successful application of sodium sulphide leaching for the recovery of artificial mercuric sulphide from mill tailing, and the possible application of this process to ore treatment. The mercury was recovered from solution by precipitation on metallic aluminum. Bradley [94] describes some related experiments on various California ores. The writers have also had access to the reports covering private tests of a number of quicksilver ores by H. G. S. Anderson, who was associated with E. B. Thornhill at Cobalt, Ontario. The experimental data indicate, in general, that good extractions can be obtained. The process, however, has not been applied on a commercial scale, and judgment in regard to its economic possibilities is based on hypothetical calculations. This analysis indicates that wet treatment will probably be more expensive than furnace treatment with no compensating advantages. The considerations are, in general, the same as those presented above in connection with concentration. Exceptional conditions might exist under which wet treatment would compare favorably with direct furnace treatment, but it is doubtful whether there is much opportunity for this process in quicksilver metallurgy.

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[90] Allen, E. T., and Crenshaw, J. L., The sulphides of zinc, cadmium, and mercury; their crystalline forms and genetic conditions: Am. Jour. Sci., vol. 34, 1912, pp. 341–396.

[91] Abegg, R., Handbuch der anorgainscher Chemie. Bd. 2, Abt. 2, Leipzig, 1905, p. 632.

[92] Mulholland, C. A., Treatment of low-grade cinnabar ores: Australian Min. Standard, vol. 45, 1911, p. 565.

[93] Thornhill, E. B., Wet method of mercury extraction: Min. and Sci. Press, vol. 110, 1915, p. 73; The recovery of mercury from amalgamation tailing, Buffalo Mines, Cobalt: Trans. Am. Inst. Min. Eng., vol. 52, 1916, p. 165.

[94] Bradley, W. W., Quicksilver resources of California: California State Min. Bur. Bull., 78, 1918, p. 321.

HEALTH HAZARDS IN THE EXTRACTION OF QUICKSILVER.[95]

By R. R. Sayers.

INTRODUCTION.

The danger of the mercurial poisoning of men employed in the mining and reduction of quicksilver ores has long been recognized, and has received considerable attention from quicksilver operators. The betterment of any conditions that call for improvement must obviously depend upon a thorough understanding of the subject, and fortunately the attention that has been given to mercurial poisoning makes it possible to discuss the subject in a comprehensive way.

There is probably no industry, trade, or art in which quicksilver is used but has produced some cases of mercurial poisoning. This is true of the mining and reduction of quicksilver ores, where the hazard has long been recognized. Cases occur when the ore contains free mercury or the more soluble salts and when the workings are underground and are poorly ventilated; but there are far more cases among the men employed about the reduction works.

Very little free quicksilver has been found in recent years in the ores mined in the United States, so that the mercurial poisoning of miners is of subordinate importance. The chief danger is in connection with the reduction of the ore in retorts and furnaces. The hazard at retort operations is the greater, because the mercury is evolved in the retort in the form of a concentrated vapor, and a small leak may permit the escape of a large amount of mercury. When the retort is opened for recharging, it may still contain a certain amount of mercury vapor, and some may still be escaping from the charge. Unless the operator thoroughly understands this condition and takes proper precautions, he may easily receive enough mercury to cause poisoning.

It may not be out of place here to refer to the danger from fumes when retorts used for distilling of quicksilver from amalgam at gold or silver mines are opened. When retorting is finished, and the cover of the retort is detached, some mercurial fumes always arise from the inside of the retort, probably being entangled in the bullion left therein. When the cover has been lifted off, nobody should remain near by for several minutes until the vapors have dissipated.

[95] Sayers, R. R.; Mercury poisoning: Bureau of Mines Reports of Investigations, Serial No. 2354, May, 1922, 6 pp.

In furnace operation there is the ever-present danger from the leakage of mercury-laden gas from the furnace and condenser, but this can be minimized by a proper regulation of draft. In a well-regulated furnace plant the only serious hazard that can not be entirely avoided is in connection with the cleaning of the condenser system where men unavoidably come in contact with mercury and mercurial soot. Another source of danger present even in a well-regulated plant is the dust that accumulates on the furnace and condenser; also about the furnace and condenser buildings. Samples of dust examined by the authors of this bulletin were found to be rich in quicksilver.

In brief, the avoidance of mercurial poisoning depends upon proper control of the workmen. The conditions leading to this poisoning are well understood, and there is no inherent difficulty in rendering the reduction of quicksilver ores an entirely safe operation. From a practical standpoint the operator is frequently confronted by the serious difficulty of obtaining a class of labor intelligent and cleanly enough to observe the necessary precautions. The small size of the majority of quicksilver plants in the United States also presents economic difficulties in providing change rooms, wash rooms, lunch rooms, etc., such as one finds in other metallurgical plants, lead smelters, for example, where a similar health hazard exists. The intermittent character of quicksilver mining in the past and the small margin of profit on which plants are compelled to run during periods of low prices—if they run at all—also makes it difficult for the operator to obtain a desirable class of workmen. The reputed danger in quicksilver operations is also a factor to be considered, and a plant that gave adequate attention to the health hazards of the industry would no doubt find it easier to obtain the right kind of workmen.

The writers of this bulletin have noticed certain instances of dangerous practices for which there is no excuse. At one plant the man who bottled the quicksilver used his hand as a dipper in adjusting the quantity of metal in the scale pan. As quicksilver was bottled almost every day he exposed himself to a condition that would unavoidedly lead to chronic mercurial poisoning. At another plant the man who drew the spent ore and did the firing at the Scott furnace smoked a corncob pipe, which he invariably laid on one of the furnace braces when doing actual work. In view of the high mercury content of the dust in and about the furnace he was doubtless taking regular doses of mercury through the mouth. A little more intelligent attention to this subject on the part of those in charge of the operation of quicksilver reduction works should go a long way toward eliminating mercurial poisoning.

GENERAL INVESTIGATIONS INTO MERCURIAL POISONING.

GENERAL EFFECTS.

In 1863 W. V. Wells[96] visited the mines at Almaden, Spain, which were worked by convicts. He reported that the men at the furnaces suffered severely, being able to work only about one week out of four, after which they had to be transferred to places where the exposure was less. He says that "pale cadaverous faces and leaden eyes are consequences of even these short periods." Mercury, like lead, when absorbed in small amounts at intervals daily, for instance over an extended period, will cause symptoms of poisoning. The poisoning is due to the fact that mercury is eliminated from the body more slowly than it is absorbed. The frequency and severity of cases of mercurial poisoning are far less with present methods of mining and recovery of the metal than they were formerly. Further improvement is possible, however, and much can be done by both the workmen and the operators by following the proper precautionary measures, some of which are described in this chapter.

Cases of industrial mercurial poisoning occurring about mines and reduction works are usually chronic with occasional development of acute symptoms when the workmen are exposed to undue amounts of mercury vapors, dust, or soot. The cardinal symptoms[97] of the disease are stomatitis (inflammation of the mouth), often with some salivation, tremors, and a peculiar timidity (erethism). These symptoms are accompanied by organic degenerative changes in various organs of the digestive system, the circulatory system, and the kidneys.

AGGRAVATING CAUSES.

Insufficient or improper ventilation of the working places, with failure to maintain sufficient draft to prevent the escape of mercury vapor from furnaces, condensers, and retorts, at once creates a condition likely to cause poisoning. Uncleanliness of the workmen appears to be no small factor. Men who do not wash their hands well before eating or putting anything into the mouth, who do not care for their teeth, who fail to rinse the mouth well before eating and drinking, who do not take a hot cleansing bath at the end of each day's work, and do not change their work clothes for street clothes after work develop mercurial poisoning more often than those who take such precautions. Taylor[98] states that of those who came under

[96] Bates, Mrs. L. W., Mercury poisoning in the industries of New York City and vicinity. 1912, p. 13.

[97] Teleky, Ludwig, Mercury poisoning; in Kober, G. M., and Hanson, W. C., Diseases of occupation and vocational hygiene, 1916, p. 130.

[98] Taylor, J. G., Chronic mercurial poisoning, with special reference to the danger in hatters' and furriers' manufactories: Guy's Hospital Reports. London, vol. 55, 1901, pp. 171–190.

his observation the men that took proper precautions scarcely ever suffered.

There is wide individual variation in susceptibility to the poisoning, which may be due to the fact that the tissues of some persons are better able to store mercury in an innocuous form than the tissues of other individuals. So far as searched, the literature does not indicate that there is a racial susceptibility, but most cases recorded occur in the white race, due probably to the fact that better records are kept for that race, also to the fact that more white men are employed in the industry than those of other races. Habits of excess, especially the use of alcohol and tobacco, seem to increase the susceptibility of individuals and to increase the severity of the symptoms.[99] Teleky[1] says that women, children, and tuberculous or cachetic individuals must be regarded as the most susceptible. Individuals coming under these classes should not be employed in any industry where poisoning by mercury is a hazard. In Belgium no one under 16 years of age is allowed to be employed where there is risk of mercurial poisoning.[2]

IMMEDIATE CAUSES.

The immediate cause of industrial mercurial poisoning is the absorption and retention of small quantities of the metal or one of its many compounds for an extended period of time. The absorption of 0.001[3] gram of mercury a day over an extended period is said to cause the disease. The process by which the mercury enters the body is not definitely known, but some authorities state that the mercurial salts combine with the tissue cells to form a mercury albuminate, soluble in water when sodium chloride is present. The conversion of metallic mercury to the oxide is supposed to occur before the albuminate can be formed. If this is true, a very fine division of the mercury tends to hasten its absorption through any of the portals of entry to the body tissue, since the surface of the mercury exposed to oxidation increases rapidly as subdivision occurs.

Quicksilver may enter the body through the skin, the gastrointestinal tract, or the respiratory tract. When applied to the skin it is more readily absorbed if the person is perspiring or if the mercury is impure and dirty. Orenstein[4] calls attention to the fact that

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[99] Taylor, J. G. Work cited, pp. 171–190.

[1] Teleky, Ludwig. Work cited, p. 130.

[2] Bates, Mrs. L. W. Work cited, 13 pp.

[3] Bangert, G. S., Occupational mercury poisoning: New York Med. Jour., vol. 107, 1918, p. 1179.

[4] Orenstein, A. J., Mercurial poisoning: Jour. Chem. Met. Min. Soc. of South Africa, vol. 20, May 1920, pp. 230, 231.

metallic quicksilver, combined with grease, is easily absorbed. He also says that 0.65 gram rubbed into the skin daily will produce mercurialism, and if the body be exposed daily to only a slightly larger quantity of mercury vaporized, poisoning will likewise result. Dry and clean mercury on a dry clean skin is probably absorbed only with difficulty.

Although comparatively large quantities of the metallic mercury can be taken at one time by mouth without causing death, small quantities often repeated will lead to chronic poisoning from the absorption and accumulation in the body tissues. This last is the condition met in those of uncleanly habits, also in those who use tobacco or eat at their working places near furnaces, condensers, and retorts. Some dust and fumes may be swallowed if the air is heavily laden, but under these circumstances the poison enters to some extent through the lungs and skin. Metallic mercury vaporizes at low temperatures, vaporization being noticeable at a temperature as low as 8.5° F., the amount vaporized increasing with the temperature. The vapors, fumes, and dust enter the body by all three portals of entry, as they settle on the clothing or skin, are breathed into the lungs, or are swallowed with food or other substances taken into the mouth.

Only a part of the mercury absorbed is retained in the body. It is slowly eliminated through the kidneys, the large intestine, and in the bile and saliva. In the urine and saliva it is found to be rather inconstant, occurring in individuals who have not been exposed to mercury for several months, and is sometimes absent in grave cases, but is not found in the feces of individuals who have not been recently exposed to the hazard. The onset, or attack, as well as the type of symptoms, depends upon the rate of absorption and elimination of the metal.

SYMPTOMS.

If the workman is being exposed to large quantities of mercury fumes, dust, or soot, the poisoning may be almost of a subacute type. At first there is only a foul breath, salivation, and a metallic taste in the mouth. This may be followed by receding of the gums, which become sore and swollen, with loosening of the teeth and even ulceration of the cheeks and gums. The skin generally becomes yellowish white, a hue similar to that found in some forms of malaria but not constant enough to permit it being called characteristic. Punctiform (dotted) hemorrhages of the skin of the back and arms are found in many furnace workers but in few miners. These hemorrhages are probably due to the evident action of mercury on the blood.

As only small amounts of mercury are ordinarily absorbed within any one day by a worker around the reduction plants, the start of the symptoms is usually slow and insidious. Giglioli [5,6] found in the cases he investigated at the mines of Abbadia San Salvatore that the men who were not physically fit, as the tuberculous, the alcoholic, and those suffering from malaria, often developed the initial symptoms after working less than a year. In addition to the symptoms given above, his patients had bleeding from the intestines, a feeling of nausea, colicky pains, and sometimes retching, especially on evacuation. Where the mercurialism was more slowly acquired, as in men of good constitution and of careful hygienic habits, Giglioli found that the following was the general course of the symptoms: First, a loss of appetite, stomatitis, and intestinal disturbances might occur, but not always; then, after a period varying from less than a year to two or three years, or even more, tremors developed. The tremors usually began with convulsive movements—known as " jerks "—during the night, interrupting the patient's sleep.

Tremors generally affect the upper limbs first, then pass to the legs. If the workman continues to expose himself to the mercury, the tremors grow worse and gradually extend to the whole body. The jaw becomes tremulous and the head shakes with a fine tremor somewhat similar to that found in very old people. The respiratory muscles may become involved, causing great difficulty in breathing. Mock [7] describes the tremor when well developed as being somewhat typical, and consisting of coarse contractions and jerks often accompanied by a fine tremor. This is termed an intention tremor, and is accentuated by the slightest effort or emotion. The stage of tremor is said not to be dangerous to life, but if exposure to the mercury is continued the brain may be affected, resulting in persistent headache, sleeplessness, and loss of memory, and the man is finally compelled to cease work. He may even then recover, but death is the more probable result.

One of the cardinal symptoms of mercurial poisoning is the peculiar timidity—similar to that shown by shy children—which develops in the sufferers, especially in the presence of strangers. They are uncomfortable and unable to do simple things of everyday life. Teleky [8,9] states that erethism occurs only in marked cases. These patients have periods of prolonged apathy with loss of memory, especially for recent events.

[5,6] Giglioli, Guido Y., [Contribution to the study of industrial mercurialism in the cinnabar basin off Mount Amiata (Siena)]: Il Ramazzini, 3 an., Jan. 1909, pt. 1, pp. 230–344.

[7] Mock, H. E., Industrial medicine and surgery. Philadelphia, 1920, p. 220.

[8,9] Teleky, Ludwig. Work cited, p. 130.

Although a paralysis occurs, it is not a frequent symptom, and neuritis, undoubtedly found in some cases, probably is even less frequent, while sensation and the reflexes remain normal.

SPECIFIC EFFECTS.

Although most authorities have found no specific changes in the nerve tissue, mercury has been extracted by chemical means from the brain of victims of mercurialism. The mercury seems to form a compound with the brain substance in such a way as to interfere with the centers controlling motion, but not to interfere with the centers controlling sensation.[10] Mercury has also been extracted from the tissues of the kidney and liver. According to Kunkel[11] there is a chronic diffuse inflammation of the kidney, which is larger than normal, smooth on the surface, has a grayish-white color, and is of a doughy consistence. If examined microscropically, the cells lining the secreting tubules—and in some cases the lining of the blood vessels—contain fatty globules, indicating fatty degeneration of these cells. The tubules themselves in some cases have been found to contain a chalky-like substance, probably calcium salts. The kidney substance is edematous with some of the fluid portion of the blood.

MacNider[12] states that in subacute forms in animals the liver was found to have degenerative changes, most marked in the midzone and periphery of the lobules. In the more chronic cases there was an increase in the connective tissue, making it much more abundant than normally found.

There are two forms of stomatitis, one of the subacute type, occurring in those who have worked in contact with mercury for a short time; the other, chronic in form, among those who have worked for a number of years.[13] In the subacute type of inflammation the mucous membrane of the cheeks and gums is redder than normally, is swollen, and in some cases shows ulcers on both the cheeks and gums. In the chronic type the gums recede from the teeth, exposing the roots. The teeth may become loosened, and, in severe cases, lose their enamel and appear wrinkled and broken. The salivary glands are infiltrated and swollen. The large intestine shows inflammation, sometimes with ulceration, corresponding to the stages described for the mucous membrane of the mouth.

In the blood there is a secondary anemia, similar to that found in malaria, the red blood cells, hemaglobin, and color index being below

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[10] Taylor, J. G. Work cited, pp. 171–190.

[11] Kunkel, A. J., Handbuch der Toxikologie. Jena. 1899–1901, pt. 1, pp. 122–148.

[12] MacNider, W. DeB., The occurrance of degenerative changes in the liver of animals intoxicated by mercuric chloride: Proc. Soc. for Experimental Biology and Medicine, vol. 16, 1918–19, pp. 82–84.

[13] Giglioli, G. Y. Work cited, pp. 230–344.

normal. Anemia is marked in cases that have existed for any great length of time and present in almost all cases to some extent.

POSSIBILITY OF RECOVERY.

Most patients suffering from industrial mercurial poisoning recover if they are removed during the early stages when there is stomatitis, or even after the development of tremor. Under these circumstances, recovery usually takes place within a few weeks, provided they are removed from exposure to mercury. Even when the tremor is severe and persistent, it will usually disappear ultimately, but often requires months, during which time patients should avoid contact with quicksilver in any form. If paralysis, delirium, insanity, or severe cachexia are present, the outcome is doubtful.[14]

Recovery in any case depends upon freedom of the patient from further exposure, and his avoidance of excesses of all kinds, especially alcoholic. As these patients are often in poor physical condition, they are very susceptible to intercurrent diseases, from which they often die.

CONCLUSIONS AND RECOMMENDATIONS.

The preventive phase of the treatment of industrial mercurial poisoning is much more important than the curative phase, and in general, if the following rules are observed, the number and severity of the cases will be materially decreased:

1. Working places should be ventilated, preferably by mechanical means, so that there is always an abundant supply of fresh air.
2. The adjustment of the draft at the furnaces and condensers should be such as to maintain negative pressure within them, thus preventing the escape of fumes and dust at all times.
3. The workers should be supplied with respirators, which should be worn whenever it is necessary to be exposed to mercury dust or fumes.
4. Change [15] and bathhouses, equipped with showers (1 for every 10 employees), washbasins or taps (1 for every 5 employees), and an individual locker for each employee should be provided. Each locker should be arranged in two compartments, one for work clothes and the other for street clothes.
5. Lunch rooms should be convenient to working places, and the eating or depositing of food should not be permitted except in the lunch room.

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14 Grieveson, E. R., Chronic mercurial poisoning: Western Canada Med. Jour., vol. 8, 1914, pp. 54–64.

15 Kendall, C. E., Change houses in Lake Superior region: Technical Paper 289, Bureau of Mines, 1923, 31 pp.

6. A physical examination should be made of all applicants for employment, and those addicted to drunkenness, those having tuberculosis, those in poor physical condition, and those under 18 years of age should not be permitted to work where they will be exposed to quicksilver in any way.

7. A physical examination of all employees should be made at least once every six months and before returning to work after any illness; and those suffering from mercurialism or those in poor physical condition should be removed from work exposing them to the poisoning.

8. Every employee should be told of the dangers of poisoning by mercury and instructed how best to avoid them. This should be done by a responsible person, especially designated for the purpose by the general manager. The employee should be further instructed how to avoid the dangers by the foreman of the work to which he is assigned.

9. As personal cleanliness and personal hygiene are important factors in preventing poisoning by mercury, the following instructions should be impressed upon each workman:

(*a*) Wash your hands carefully before eating or handling anything that will be placed in the mouth.

(*b*) Always rinse your mouth before taking a drink or eating.

(*c*) Drink water and milk plentifully.

(*d*) Never come to work without having eaten a substantial meal. With an empty stomach conditions are more favorable for the absorption of mercury into the body.

(*e*) Take a hot shower bath, using soap, at the end of each day's work.

(*f*) Change your ordinary clothing for work before going to the plant and your work clothing for ordinary clothing at the end of the shift. Work clothing should cover the body well and be tight fitting at the ankles, wrists, and neck. A close-fitting cap or hat should be worn. The work clothing should be washed frequently, at least once a week.

(*g*) When passing to leeward of fumes, hold your breath as much as possible. Pass to the windward wherever possible. When necessary to be in fumes or dust, wear a respirator at all times.

(*h*) After having been in fumes, dust, or soot, wash the exposed parts of the body, hands, and face, and rinse the mouth well at the first opportunity.

(*i*) Do not allow yourself to become constipated. Regular habit in this regard is much more important than cathartics; train yourself to be regular.

(*j*) Care of the teeth and gums is most important. They should be kept clean and healthy, and should be examined at least once every six months by a dentist, and his recommendations followed.

(*k*) If you feel at all sick, consult the doctor at once.

TREATMENT OF PATIENTS.

To treat industrial mercurial poisoning successfully it is imperative that the patient be removed from all possibility of further exposure to mercury. The basis of the active treatment is eliminative by stimulating the activity of the kidneys, the intestines, and the sweat glands. Magnesium citrate has been found a very satisfactory cathartic in these cases. Large quantities of water should be taken for its effect on both the kidneys and intestines. Hot baths, sweat baths, and other hydrotherapeutic measures are found to be beneficial. Iodides are usually prescribed, but, according to Schereschewsky, they are of doubtful value. The stomatitis and digestive disturbances should be treated according to the symptoms. Potassium chlorate solution and astringent mouth washes should be used. Massage is said to be of value in treating the tremors.

The general physical condition of the patient should be improved by an abundant diet of milk, eggs, cream, and other nutritious food.

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