CHAPTER V.—AMALGAMATING MILLS.

RELATIVE AMOUNT OF ORE MILLED.—A considerably larger amount of ore is treated by the various processes depending on amalgamation than is smelted or leached. In California amalgamating mills altogether outrank the smelting establishments, but are supplemented in many cases by lixiviation works. In Nevada the Eureka smelting district is the principal exception to the rule, 78,807 out of the total 346,331 tons treated in the whole state during the census year having been worked at Eureka. In Colorado, however, less ore was milled than smelted during the same period, the respective totals being 131,948 tons milled, as against 179,564 tons smelted. In Montana, out of a total of 86,576 tons reduced during the census period, 71,896 tons were worked in amalgamating mills. Oregon, Dakota, and Wyoming ores are treated by amalgamation. Until lately all the ores in Idaho were milled, but the discovery of important base-metal districts has given an impulse to smelting operations. Mills take precedence in Arizona. In general, it may be said that, wherever practicable, owing to the peculiar economic conditions prevailing in the far West, the amalgamation process is adopted in preference to smelting from motives of economy in first cost of plant and in actual treatment, though, on the other hand, the percentage of precious metals saved by the mills is usually much lower than the results reached in smelting works.

CAPACITY OF MILLS.—The quantity of the ore which can be treated per month varies from 50 tons to several thousand, according to the quality of the ore and the size of works. Among large mills the Homestake gold mill has a capacity of 325 tons and the California silver mill 380 tons in twenty-four hours. Some of the mills are kept steadily at work from one year's end to the other, but many of them are only running for a few weeks at a time.

OWNERSHIP.—The large mills are usually owned and operated by mining companies, though a few custom mills of considerable size are reported, and on the Comstock the greater number of mills are owned by companies which are engaged exclusively in milling. Of the custom mills, nearly all were built by mining companies, and have passed, from failure of the mines or other causes, into private hands.

TENOR OF ORE TREATED.—The range in the assay value of the ore treated is very great, extraordinary facilities and a large plant permitting ore to be worked in some places at a profit which under other conditions could only be treated at a loss. The physical character of the ore also varies greatly, the quartz in some districts being extremely coherent and tough and in others friable. Some ores are so soft that 10 tons per 24 hours can be crushed with a single stamp. The predominating color in chloride or oxidized ores is a dull reddish yellow, and in sulphuret ores various shades of gray.

The ore treated by silver mills contains nearly every known silver mineral, the specimens collected in connection with the present census embracing an almost complete list of even the rarer species. What are known as free ores, if from near the surface and exposed to atmospheric agencies, contain chloride or horn-silver (cerargyrite) and more rarely native silver, or sulphide (argentite) if from below the water line, although the latter mineral is also found at or near the surface. In ores which require a preliminary roasting stephanite, tetrahedrite, polybasite, pyrargyrite, proustite, etc., are found; in fact, the ores containing silver, and also sulphur, copper, lead, antimony, arsenic, or zinc, are met with in almost endless variety. The most common gangue is quartz, with or without calc-spar and other minerals. As exceptions may be mentioned White Pine, Nevada, where the gangue is siliceous limestone; El Dorado cañon, where it is calc-spar and chlorite; and Silver Reef, Utah, where it is sandstone. Fluorspar and barite, as well as many other minerals, occur occasionally as the gangue of lodes. In some places the association of minerals is a very remarkable one; for instance, in the Spanish Belt district, Nevada, cinnabar is found in silver-bearing tetrahedrite, and at the Hackberry mine, Arizona, free sulphur was found in the oxidized chloride ore of a silver mine.

Gold is generally found in the ore in a free state, though frequently it is so intimately mixed with iron sulphurets that it is difficult of amalgamation. At other times the gold is so extremely fine that it is carried off on the water before it can be amalgamated. It is often supposed that a sulphuret of gold is found in some mines, and that "coated" gold, or gold which cannot be amalgamated in the ordinary way, exists is a matter of fact; but there is little authority for the belief that sulphur is chemically combined with gold in these ores.

FORCE EMPLOYED.—A 20-stamp wet-crushing silver mill, running 24 hours per day, generally needs the following crew of men:

2 rock-breaker men.
2 battery feeders.
2 amalgamators.
2 amalgamators' helpers.
2 engineers (and in some cases 2 firemen).
1 roustabout.
1 foreman and assayer.

A somewhat larger mill would only require an increase of a hand or two, and a smaller mill, even a 5-stamp one, could not run with a much smaller force. The usual crew for a 20-stamp gold mill is five men.

As an example of the force required by a 20-stamp dry-crushing silver mill, the pay-roll of the Grand Prize mill, at Tuscarora, Nevada, may be quoted. This mill is run by a very strong crew. Some of the more recently constructed dry-crushing mills, which have more automatic appliances, such as continuous rotary driers in place of kilns, dispense with a few hands.

TABLE XCVIII.—FORCE EMPLOYED AT THE GRAND PRIZE 20-STAMP DRY-CRUSHING AND CHLORIDIZING MILL.

Class. (a)	Average number employed.	Length of shift.	Wages per shift.	Total number of hours' work per day.	Total wages per day.
Total	28	*Hours.*	....	336	$121 00
Amalgamators	2	12	$5 00	24	10 00
Amalgamators' helpers	2	12	4 00	24	8 00
Chloridizers	2	12	5 00	24	10 00
Chloridizers' helpers	2	12	4 00	24	8 00
Battery feeders	2	12	5 00	24	10 00
Engineers	2	12	5 00	24	10 00
Firemen	4	12	4 00	48	16 00
Melter and retorter	1	12	4 00	12	4 00
Dry-kiln men	6	12	4 00	72	24 00
Blacksmith	1	12	5 00	12	5 00
Laborers	4	12	4 00	48	16 00

a The assayer and foreman are included in staff.

In Tuscarora district there are two other mills, each of 10 stamps, the daily pay-rolls of which are added by way of comparison. Of these the Independence-Navajo is a dry-crushing and chloridizing-roasting mill, while the Lancaster, which runs on chloride ore, crushes dry, but amalgamates raw.

TABLE XCIX.—FORCE EMPLOYED AT THE INDEPENDENCE-NAVAJO 10-STAMP DRY-CRUSHING AND CHLORIDIZING MILL.

Class. (a)	Average number employed.	Length of shift.	Wages per shift.	Total number of hours' work per day.	Total wages per day.
Total	16	*Hours.*	....	192	$71 00
Battery feeders	2	12	$4 50	24	9 00
Dry-kiln men	4	12	4 00	48	16 00
Chloridizers	2	12	5 00	24	10 00
Chloridizers' helpers	2	12	4 00	24	8 00
Amalgamators	2	12	5 00	24	10 00
Engineers	2	12	5 00	24	10 00
Firemen	2	12	4 00	24	8 00

a The assayer and foreman are included in staff.

TABLE C.—FORCE EMPLOYED AT THE LANCASTER 10-STAMP DRY-CRUSHING AND RAW-AMALGAMATING MILL.

Class. (a)	Average number employed.	Length of shift.	Wages per shift.	Total number of hours' work per day.	Total wages per day.
Total	16	*Hours.*	....	192	$69 00
Engineers	2	12	$5 00	24	10 00
Firemen	3	12	4 00	36	12 00
Battery feeders	2	12	4 50	24	9 00
Amalgamators	2	12	5 00	24	10 00
Amalgamators' helpers	2	12	4 00	24	8 00
Drier tenders	2	12	4 00	24	8 00
Laborer	1	12	4 00	12	4 00
Sage-brush wheelers	2	12	4 00	24	8 00

a The assayer and foreman are included in staff.

WAGES PAID.—The wages for mill employés range from $2 to $5 for twelve-hour shifts, and seem to be in proportion to the wages paid miners in the district where the works are situated. As a rule, mill employés receive about 10 per cent. more than miners, the difference in the length of their shifts (which for mill hands is almost invariably twelve hours) entitling them to more pay. In one or two districts mill hands were paid less than miners, however. Chinese are employed to some extent, and receive from $1 to $2. They work generally as laborers. Foremen are paid from $4 to $10 per day. The scale of wages paid in the Comstock mills is appended:

TABLE CI.—RATE OF WAGES OF DIFFERENT CLASSES OF EMPLOYÉS IN THE COMSTOCK MILLS.

Class.	Wages per shift.
Agitator men	$3 50
Amalgamators	4 00
Blacksmiths	5 00
Blanket sweepers	3 00 to 3 50
Carpenters	5 00 to 6 00
Chargers	3 50 to 4 00
Driers	4 00
Engineers	5 00 to 7 00
Feeders	3 00 to 4 00
Firemen	4 00
Foremen	5 00 to 6 00
Laborers	3 00 to 4 00
Masons	6 00
Oilers	3 50 to 4 00
Panmen	4 50 to 5 00
Retorters	3 50 to 4 00
Refiners	4 00 to 4 50
Tankmen	3 50 to 4 00
Teamsters	3 50
Watchmen	3 50 to 4 00
Woodmen	3 50

Illustrative tables are appended, showing the classification of the employés in the Comstock mills according to occupation and nationality. Of the 339 men whose nationality is recorded, 8 are classed with the staff and are omitted from the table of occupations.

TABLE CII.—NUMBER OF MEN OF DIFFERENT CLASSES EMPLOYED IN THE PRINCIPAL COMSTOCK MILLS, EXCLUSIVE OF STAFF: 1880.

Mill.	Number of men employed.	Amalgamators.	Acid makers.	Battery feeders.	Blacksmiths.	Blanket sweepers.	Bluestone makers.	Bullion refiners.	Carpenters.	Cooks and waiters.	Engineers.	Firemen.	Foremen.	Laborers.	Lead burners.	Machinists.	Metal roasters.	Millwrights.	Oilers.	Retorters.	Tankmen.	Teamsters.	Watchmen.
Total	a331	27	2	3	10	17	1	3	5	5	22	6	5	140	1	3	5	1	7	3	50	9	6
Brunswick	40	4	....	....	2	4	....	....	1	....	....	....	....	9	....	....	....	....	2	2	14	1	1
California	72	7	....	....	1	8	....	....	1	....	4	3	3	33	....	2	....	....	2	....	6	....	2
Excelsior	12	....	....	....	....	....	....	....	....	1	2	....	1	8	....	....	....	....	....	....	....	....	....
Franklin	15	2	....	....	2	2	....	....	1	....	....	....	....	8	....	....	....	....	....	....	....	....	....
Lyon	46	....	2	....	1	....	1	....	....	3	....	3	1	23	1	....	5	....	....	....	....	6	....
Mariposa	18	3	....	....	....	....	....	....	....	....	3	....	....	7	....	1	....	....	....	....	4	....	....
Morgan	33	2	....	....	2	1	....	....	1	....	3	....	....	9	....	....	....	....	3	1	10	1	....
Omega	12	....	....	....	....	....	....	3	....	....	....	....	....	7	....	....	....	....	....	....	....	....	2
Scorpion G. and S. M. Co. (b)	26	4	....	3	....	2	....	....	....	1	5	....	....	7	....	....	....	1	....	....	2	1	....
Trench	26	3	....	....	1	....	....	....	....	....	3	....	....	4	....	....	....	....	....	....	14	....	1
Woodworth	31	2	....	....	1	....	....	....	1	....	2	....	....	25	....	....	....	....	....	....	....	....	....

a In addition to the number of men here classified there were employed at other mills, which were either idle or running but a short time, 66 men, as watchmen, laborers, etc. The total amount paid as wages during the census year by the Comstock mills was $372,767 19.

b Includes Boston and Douglas mills.

TABLE CIII.—NATIONALITY OF MEN EMPLOYED IN THE PRINCIPAL COMSTOCK MILLS, INCLUDING STAFF: 1880.

Mill.	Number employed.	Americans.	Azores islands.	Belgians.	Canadians.	Channel islands.	Chinese.	Danes.	English.	French.	Germans.	Irish.	Italians.	Newfoundland.	Norwegians.	Nova Scotians.	Portuguese.	Russians.	Scotch.	Slavonians.	Swedes.	Swiss.	Welsh.
Total	339	115	1	1	36	1	20	12	13	1	16	77	2	2	2	4	10	1	5	2	5	3	1
Brunswick	41	17	....	....	7	....	....	1	2	....	2	10	....	....	....	....	....	....	....	2	....	....	....
California	72	20	....	1	5	....	....	....	2	....	5	28	1	....	....	1	2	1	2	....	2	2	....
Excelsior	12	3	....	....	2	....	1	....	....	....	2	2	....	....	....	....	2	....	....	....	....	....	....
Franklin	16	7	....	....	1	....	....	3	....	1	1	2	....	....	....	....	....	....	1	....	....	....	....
Lyon	48	17	1	....	5	1	14	....	3	....	1	3	....	....	1	....	2	....	....	....	....	....	....
Mariposa	18	6	....	....	3	....	....	2	....	....	2	4	....	....	....	1	....	....	....	....	....	....	....
Morgan	34	5	....	....	2	....	....	5	4	....	3	8	....	....	1	1	....	....	2	....	2	1	....
Omega	13	3	....	....	5	....	....	....	....	....	....	4	....	....	....	....	....	....	....	....	....	....	1
Scorpion G. and S. M. Co. (a)	26	14	....	....	....	....	1	....	....	....	....	5	....	....	....	1	4	....	....	....	1	....	....
Trench	27	10	....	....	5	....	....	....	1	....	....	9	....	2	....	....	....	....	....	....	....	....	....
Woodworth	32	13	....	....	1	....	13	1	1	....	....	2	1	....	....	....	....	....	....	....	....	....	....

a Includes Boston and Douglas mills.

RELATION OF LABOR TO PRODUCT.—An analysis of the results reached in 160 mills shows that for every ton of ore crushed the labor of one man for 3.9 hours, costing $1 34, is required. For the separate establishments the range is very great, depending upon the character of the ore and the treatment and upon the efficiency of the machinery, while wide variations are found in the averages for the different states and territories. The most economical process in point of labor, as in other details, is the treatment of free-milling gold ores by large wet-crushing mills; the most expensive, that of reducing base silver ores, which require chloridizing-roasting as a preliminary to pan amalgamation. Dakota, where the ores worked are exclusively of gold, and California and Oregon, where such ores largely predominate, are therefore the localities where the greatest saving in labor is effected; while Arizona, Nevada, and Montana show a much greater expenditure of labor, as would be expected, in view of the character of their ores. In Colorado and Idaho the different classes of mills are nearly balanced, and the labor employed in crushing a ton of ore closely approximates the average for the whole country. It should be observed that while in tonnage the gold mills take the lead, the silver mills treat ore of a higher grade, so that a comparison based upon the bullion product would show somewhat different results.

To produce a dollar in gold bullion costs 8 cents for mill labor in Dakota; in Nevada, though over five times as much work is required per ton treated, the cost for mill labor per dollar of silver bullion produced is 7 cents, or practically the same proportion. The following table shows the average number of hours' work and cost of labor per ton of ore crushed in the principal mining states and territories, and is based on the treatment of nearly a million and a half tons:

TABLE CIV.—AMOUNT AND COST OF LABOR IN AMALGAMATING MILLS.

State or territory.	Number of mills.	Tons treated.	Total number of men employed.	Foremen.	Amalgamators.	Day laborers.	Other workmen.	Total number of hours' work done during the year.	Total sum paid on labor account, exclusive of staff.	Average number of hours per ton crushed.	Average amount paid for labor per ton crushed.
Total	160	1,467,478.75	2,046	101	302	657	986	5,745,785	$1,973,906 00	3.915	$1.345
Arizona	11	15,946.40	137	3	23	64	47	200,340	71,717 00	12.563	4.497
California	33	419,883.50	301	13	76	147	65	1,078,742	291,577 60	2.569	0.694
Colorado	28	140,117.35	197	23	22	57	95	579,574	146,024 00	4.137	1.042
Dakota	22	506,238.00	259	13	35	147	64	825,001	251,759 00	1.629	0.497
Idaho	7	16,301.25	59	1	8	9	41	70,128	22,017 00	4.278	1.343
Montana	13	47,801.00	144	9	17	62	56	373,746	191,609 00	7.819	4.008
Nevada	33	237,004.75	742	34	96	147	465	2,063,813	739,149 00	8.686	3.111
Oregon	5	12,437.00	27	2	6	14	5	41,684	10,250 00	3.352	0.824
Utah	8	71,[illegible]59.50	180	3	19	10	148	512,757	249,804 00	7.216	3.515

POWER.—The proportion which the horse-power of the engine bears to the stamp seems to be as follows:

For gold mills, from 2 to 3 horse-power.
For wet-crushing silver mills, from 5 to 6 horse-power.
For dry-crushing silver mills, from 6 to 7 horse-power.

These figures are only approximate, as the data in regard to the horse-power of engines in different mills are no doubt in many cases incorrect, and the horse-power is not only governed by the size of machinery, but also by the number of stamps, etc., a small mill requiring proportionately more horse-power to the stamp than a large one.

The pressure of steam ordinarily maintained in the boilers is about 80 pounds; but in some few mills it is 90 pounds, and even as high as 100 pounds. Very few boilers are run with a pressure of less than 70 pounds.

Water-power is also used in many mills, and where practicable is much more economical than steam. Some mills are fitted with both steam-engines and water-wheels, the former being brought into use when the latter are unavailable, as in case of drought or freezing. The water-motors are turbines of the Leffel or Knight patterns, overshot wheels, or more rarely hurdy-gurdy wheels.

FUEL.—The fuel used in quartz-mills is generally wood, and it is of many different kinds and qualities. In some mills in Nevada sage-brush is used under the boilers. The leading kinds of wood are: mountain mahogany, weighing from (*a*) 3,200 to (*b*) 4,400 pounds per cord; mesquit, weighing from 2,500 to 3,500 pounds; nut-pine, with very variable weight, depending on seasoning, size, and shape of sticks, etc., may be put down at between 2,000 and 3,300 pounds; sugar-pine, from 2,000 to 2,300 pounds; cedar, 1,800 to 2,500 pounds; cottonwood, 1,500 to 2,300 pounds. As near as can be determined 156 pounds of sage-brush are equal to 100 pounds of good cedar. The averaging of different weights of the same kind of wood is impossible, as the weight of a given kind of wood depends upon the locality of its growth (which affects its grain), the seasoning, and the closeness with which it is corded.

It is almost impossible, on account of the different character of the wood burned and the differences of method and machinery, to determine the quantity of fuel consumed under the boiler in working a ton of ore. For gold mills the consumption is estimated at from one-tenth to one-fifth of a cord per stamp, while for wet-crushing silver mills three-quarters of a cord is probably the maximum, in which case the wood must needs be of a very poor kind and quality or the machinery very inefficient; the average is thought to be less than half a cord. In roasting-mills, which are always dry-crushing, about one-third of a cord is consumed to the stamp. This does not include the wood necessary for roasting, which will be mentioned later.

The fuel consumed in retorting can safely be estimated at from one-fifth to one-third of a cord to the 1,000 pounds of amalgam.

WORKING RESULTS.—The question of percentage extracted is one in regard to which the expert is obliged to depend entirely upon the statement of the mill man, and in many cases the percentage claimed is higher than the circumstances under which the ore was worked warrant. Sometimes the percentage extracted is calculated from the assay value of the tailings, a method which rarely, if ever, gives true results; often it is merely determined by guess, leaving a very liberal margin in favor of the mill man's skill.

Where silver ore is roasted, probably from 80 to 90 per cent. is extracted on the average, though a few mills have been known to work up to 96 per cent. In so-called free-milling ore generally not much over 70 per cent. is extracted, although mills in White Pine, Nevada, and Silver Reef, Utah, have worked ores to 85 per cent., and on the Comstock ore has been worked to above 80 per cent.

In gold mills there is a wide difference in the percentage extracted, but it rarely exceeds 80 per cent., and often falls much below it.

It is not possible to fix any definite figure for the average assay value of tailings and slimes. The value varies exceedingly according to richness of ore and mode of treatment.

COST OF TREATMENT.—The cost of milling gold ores in California varies from 54 cents to \$2 50 per ton. Where water-power is to be had, ores can be milled very cheaply, at a cost of not more than \$1; and where steam-power is necessary, exceptional facilities may reduce the expenses of working to those of the average water-power mills. In other states and territories it is as a general thing somewhat more expensive to reduce gold ores; but in the Black Hills region of Dakota the large gold mills are operated very cheaply, in spite of high freights and other disadvantages, because of the simplicity of the process, the scale upon which operations are carried on, and the economy in labor which is affected by the use of automatic labor-saving devices.

The cost of milling silver ores in wet-crushing is from \$4 to \$8; in dry-crushing from \$6 to \$12. With ores which require roasting it is from \$12 to \$28. There are, of course, circumstances where these figures are exceeded, and sometimes it may be possible to work ores at less cost than the lowest prices given. The great difference in ores, in the facilities for working, and in the methods of extracting the precious metals, render it impossible to make a general statement as to the cost of the process.

REDUCIBILITY OF VARIOUS ORES.—Those ores can be most cheaply worked which contain silver as chloride without base metals or carry gold unmixed with sulphurets. As instances of particularly free-milling silver ores

a This weight is from data of the Eureka and Palisade railroad.
b This weight is from data of the Richmond Mining Company.

those of White Pine, in Nevada, and Silver Reef, in Utah, may be mentioned. Next in order come those ores which, although they contain chloride of silver, are mixed with lead (generally in the form of carbonate) and various copper, arsenic, and antimony minerals. The ore of the Comstock, although it contains but little base metal, may be classed under this head, as most of its silver is in the form of sulphide and a part of the gold is contained in the iron pyrites. It mills from 70 to 80 per cent. The last and most difficult class of ores to work comprises the many complex sulphides of silver and other elements; and although it is almost invariably necessary to roast these ores, some of them (such, as are composed of quartz, silver sulphide, and a little iron and copper pyrites) can be roasted and worked at a cost not much in excess of that of the baser free-milling ores, and certainly to better advantage, as at least 90 per cent. of the assay value can thus be extracted. But these ores are very rarely met with, and by far the larger class is that in which the ore either does not contain enough sulphur to render the chloridation complete or contain large quantities of the sulphides of zinc, antimony, and arsenic, and can therefore be roasted only at the expense of much time, care, and money. Such are those of Morey, in Nevada, and many in Arizona. Frequently mistakes in deciding upon the proper plant and process to work ores are made, as it is often a delicate matter to decide whether an ore can be best milled or smelted, and, if milled, whether it requires roasting. The mining country is covered over with reduction works which are failures, owing to the wrong method having been chosen to work the ores. Such disappointments are of rare occurrence nowadays, more prudence and experience being brought to bear on this class of investment than was formerly the case.

TERMS OF CUSTOM MILLS.—The prices charged for reducing ores are generally one-quarter to one-half as much again as the actual cost, or even more, except with such milling companies as own mines, which they lease to miners, expecting to make their profit out of the "tribute" and not out of the mill.

Usually gold mills do not guarantee any percentage. In silver mills, where the ore is worked raw, the guarantee is from 70 to 80 per cent. of the assay value. Sometimes a "clean up" is given; that is, after the ore is worked, the bullion product is gathered together and turned over to the ore owners, who pay the charges for working. When ore needs roasting, the mill generally guarantees 80 per cent.; sometimes, as in Austin, 80 per cent. on such ore only as assays over a certain fixed value. Where a percentage is guaranteed, the tailings, slimes, etc., are looked upon as belonging to the mill company; but when ore has been worked on the "clean up" plan they are generally regarded as belonging to the mining company, unless there has been a previous agreement in reference to them. In purchasing ores, the percentage of the assay value offered is paid after deducting the price of milling.

When custom ore is worked the mine almost always pays for the hauling of the ore to the mill. Ore is usually transported by wagon or on pack animals, and sometimes by a tramway or a railroad.

TENOR OF BULLION.—The quality of bullion produced is, of course, very varied. In gold mills the fineness usually runs from 0.600 to 0.850 in gold (the maximum reported was 0.920), silver constituting most of the balance of the bullion. In silver mills the bullion has a much wider range of fineness, running from 0.050 (as was the case with a little of the bullion at the Meadow Valley mill, in Pioche) up to 0.999. Among instances of fine silver bullion may be mentioned that of the Comstock and of White Pine, in Nevada, and that of Silver Reef, in Utah. In these places it does not often run below 0.990, and often as high as 0.999. The quality of bullion depends generally upon the character of the ore, but sometimes upon the method of working. For instance, some base ores, in which most of the silver is in the form of chloride, can be made to produce bullion 0.900 fine and upward by amalgamating cold or rather not hot, or by using the minimum of salt and little or no bluestone. With such ores it is much better to keep the bullion at from 0.600 to 0.700; for, although the silver of the chloride is the first metal amalgamated, the sulphide silver minerals remain, unamalgamated, with the base-metal compounds.

The only base metals which are found to any extent in the amalgam are lead and copper. Iron in some quantity is occasionally found, but its presence is owing to a mechanical mixture of particles worn from the battery and pans, and not to any chemical process. Sulphurets containing zinc, antimony, arsenic, and some other metals also enter into the amalgam in the same way, but by adding quicksilver to the amalgam and washing in a clean-up pan these sulphurets and the iron can be removed.

DISPOSITION OF BULLION.—Bullion is usually shipped to the New York or San Francisco markets through Wells, Fargo & Co. and other express companies. The charges vary from ½ to 2½ per cent., the discount being governed by the market value of silver and the quality of the bullion.

STAMP-BATTERIES.

WET AND DRY CRUSHING.—For gold ore the wet-crushing battery is exclusively used, and for free-milling silver ore it is also generally employed. In some cases, as at Eberhardt, in White Pine, and the Lancaster mill, at Tuscarora, dry-crushing is practiced on the ground that a much higher percentage is saved, owing to the avoidance of loss of finely divided horn-silver in the slimes, which is apt to occur when the ore is crushed wet. This is highly probable with chloride ore, for, although the slimes are very thin, the earthy material in them seems to be richer in silver than the ore itself. There are also some ores which, when crushed, produce so much slime that it is impossible to keep a reasonable percentage of pulp in the mill. For such ores dry-crushing is preferable; but for other free-milling ores wet-crushing is by far the best, as more ore can be crushed, there is less wear and tear on the battery per ton, and the hands are not subjected to that inconvenient and unhealthy dust which everywhere pervades the atmosphere of a dry-crushing mill, in spite of attempts made to get rid of it. To a certain extent the advantages of dry-crushing and wet-crushing for free-milling ores are combined in the plan adopted at Bodie and elsewhere of pumping back the water from the settling-tanks into the battery. When it is intended to roast ore, it is always crushed dry.

WHERE BUILT.—The machinery of most of the mills in operation on the Pacific coast has been constructed in San Francisco; the plant of some mills, however, has been brought from the East, and of the latter Chicago has furnished a large proportion. Recently the work of the eastern founderies which turn out mining and milling machinery has improved greatly in quality; and, aided by the cheapness of labor and of material, they have been able to compete with San Francisco and other Pacific coast establishments, even at a relative disadvantage in point of freight charges. As a rule, the types of machinery in vogue have originated and have been developed in works in the neighborhood of the mines, and these establishments have shown much enterprise in adopting new improvements. Latterly, however, since the manufacture of mining and milling machinery has grown to be an industry of importance in the East, novelties and improvements are to be found in the product of the eastern works. Innumerable worthless inventions have also issued from all points at which mining machinery is manufactured.

SIZE OF MILLS.—Among the large mills may be mentioned the following: In California, the Empire (80-stamp), in Amador county, and the Plumas-Eureka (88-stamp), in Plumas county. In Nevada, the California (80-stamp), at Virginia City. In Dakota, the Homestake old (80-stamp) and new (120-stamp) mills, in Whitewood district, and the Father de Smet (80-stamp), in Lost Placer district. In Colorado, the Black Hawk (125-stamp), of Gregory district. In Arizona, the Central Arizona (80-stamp) mill, at Vulture, Maricopa county. There is, on the other hand, a steam battery of two stamps at Globe, Arizona, and a Huntington double-stamp battery at Queen's River, Idaho, while a single-stamp battery, the Kendall, is being introduced for prospecting tests. The largest mills, as gauged by the number of stamps only, are as a rule gold mills, though there are several 40- and 60-stamp silver mills, beside the California (80-stamp), above mentioned.

The most common size for a mill seems to be 20 stamps; it is also a very convenient size, especially for a roasting mill. The size, of course, ought to be governed by the quantity of ore available; but this is not always the case, for mills have been built where there was no ore, and often where the crushing capacity was vastly in excess of any probable output of the adjacent mines.

NUMBER OF STAMPS PER BATTERY.—Five is the usual number of stamps per battery, though some of the older mills have but four. The Gold Stripe mill, in Plumas county, California, has six. Millmen are commonly of the opinion that an even distribution of pulp in a mortar can only be secured with an odd number of stamps, and of such numbers five is on several accounts the most convenient.

WEIGHT OF STAMPS.—In new mills the usual weight of stamps is from 700 to 800 pounds each, though great variations still occur in practice. As will be seen from the subjoined tables, the range in weight is from 300 to 1,050 pounds. The average for 280 mills reported on, having 5,367 stamps, is 689 pounds. The heaviest batteries, as a rule, are to be found in Nevada, and the lightest in Alabama and Georgia. Of the important mining states Colorado appears as the locality where light stamps are most in vogue.

TABLE CV.—STAMP BATTERIES: NUMBER AND WEIGHT OF STAMPS.

County and district.	Number of mills reported.	Number of stamps.	Maximum weight of stamps.	Minimum weight of stamps.	Average weight of stamps.
ALABAMA.					
CLEBURNE CO.			*Pounds.*	*Pounds.*	*Pounds.*
Third	1	8	400	350	375
TALLADEGA CO.					
T. 19, R. 6	1	8	300	300	300
ARIZONA.					
MARICOPA CO.					
Globe	2	7	1,000	750	821
MOHAVE CO.					
Cedar Valley	1	5	750	750	750
Hualapai	1	5	950	950	950
Owens	2	30	850	850	850
PIMA CO.					
Arivaca	2	30	750	350	483
Tombstone	3	45	900	750	850
PINAL CO.					
Globe	1	5	900	900	900
Pioneer	2	25	750	750	750
ARIZONA—Continued.					
YAVAPAI CO.			*Pounds.*	*Pounds.*	*Pounds.*
Cherry Creek	1	10	850	850	850
Peck	1	10	850	850	850
Pine Grove	1	10	750	750	750
Tiger	1	10	950	950	950
....	1	10	800	800	800
CALIFORNIA.					
AMADOR CO.					
Amador City	2	80	800	750	775
Jackson	1	30	750	750	750
Plymouth	1	80	750	750	750
Sutter Creek	1	40	640	640	640
CALAVERAS CO.					
Washington	1	10	600	600	600
Independence	1	10	500	500	500
Mokelumne Hill	1	60	650	650	650
EL DORADO CO.					
Placerville	1	20	750	750	750
Springfield	1	10	650	650	650

TABLE CV.—STAMP BATTERIES: NUMBER AND WEIGHT OF STAMPS—Continued.

County and district.	Number of mills reported.	Number of stamps.	Maximum weight of stamps.	Minimum weight of stamps.	Average weight of stamps.
CALIFORNIA—Continued.					
INYO CO.			*Pounds.*	*Pounds.*	*Pounds.*
Russ	1	30	600	600	600
LASSEN CO.					
Hayden Hill	3	20	650	600	625
MARIPOSA CO.					
Mariposa Estate	1	40	750	750	750
MONO CO.					
Blind Spring	1	5	750	750	750
Bodie	6	120	800	530	749
NEVADA CO.					
Grass Valley	1	50	850	850	850
Nevada City	2	88	750	750	750
PLACER CO.					
Colfax	1	10	800	800	800
PLUMAS CO.					
Genesee Valley	1	10	750	750	750
Indian Valley	4	96	765	700	738
Quartz township	1	88	800	800	800
Seneca	1	5	650	650	650
SAN BERNARDINO CO.					
Clark	2	15	750	600	650
SAN DIEGO CO.					
Cargo Muchacho	1	20	950	850	900
Julian	1	5	650	650	650
SHASTA CO.					
Pittsburgh	1	10	900	900	900
SISKIYOU CO.					
Quartz Valley	1	10	600	600	600
Sawyer's Bar	1	32	650	650	650
South Fork Salmon	1	32	650	650	650
TUOLUMNE CO.					
Confidence	1	40	750	750	750
Soulsbyville	1	15	750	750	750
COLORADO.					
BOULDER CO.					
Central	1	25	*a* 450	*a* 450	*a* 450
Gold Hill	1	10	750	750	750
Grand Island	1	15	650	650	650
Ward	2	70	650	450	593
	1	5	750	750	750
CLEAR CREEK CO.					
Griffith	2	30	750	650	700
Idaho	1	25	650	650	650
Morris	1	12	500	500	500
Upper Union	1	10	500	500	500
CUSTER CO.					
Hardscrabble	1	20	650	650	650
GILPIN CO.					
Enterprise	8	223	550	400	490
Eureka	1	20	550	550	550
Gregory	6	350	600	550	570
Nevada	8	114	600	500	536
LAKE CO.					
California	1	15	750	750	750
RIO GRANDE CO.					
Summit	1	10	530	530	530
SUMMIT CO.					
Avalanche	1	15	525	525	525
SAN JUAN CO.					
Animas	1	10	750	750	750
DAKOTA.					
LAWRENCE CO.					
Bear Butte	2	20	700	650	675
Lost Placer	1	80	750	750	750
Whitewood	20	740	800	550	692
GEORGIA.					
CHEROKEE CO.			*Pounds.*	*Pounds.*	*Pounds.*
Third and Second	1	5	400	400	400
	1	5	400	400	400
COBB CO.					
	1	10	500	500	500
FORSYTH CO.					
Third and First	1	5	650	650	650
HALL CO.					
Ninth	2	15	1,050	625	908
HARALSON CO.					
Eighth	1	10	400	400	400
LINCOLN CO.					
Goshen	1	8	300	300	300
LUMPKIN CO.					
Twelfth and First	9	145	500	450	473
Thirteenth	1	10	450	450	450
McDUFFIE CO.					
Republican	1	5	550	550	550
MERIWETHER CO.					
Lutherville 1281	1	10	350	350	350
OGLETHORPE CO.					
Simston	1	10	650	650	650
WHITE CO.					
Third Land	1	20	450	450	450
Fourth Militia	1	10	700	700	700
WILKES CO.					
Bussey	1	10	750	750	750
178	1	5	450	450	450
	1	5	525	525	525
IDAHO.					
ALTURAS CO.					
Bonaparte	1	20	630	630	630
Middle Boisé	4	40	650	650	650
Queen's River	1	2	850	850	850
Rocky Bar, or Bear Creek	1	10	650	650	650
BOISÉ CO.					
Banner	1	20	650	650	650
Elkhorn	1	5	700	700	700
Gambrinus	1	10	650	650	650
Granite	2	85	800	700	729
Mammoth	2	16	700	550	625
Shaw's Mountain	1	10	550	550	550
LEMHI CO.					
Yankee Fork	1	20	825	825	825
OWYHEE CO.					
Carson	6	75	900	628	674
Wagontown	2	20	650	650	650
MAINE.					
HANCOCK CO.					
Sullivan	1	10	750	750	750
MONTANA.					
BEAVER HEAD CO.					
Bald Mountain	1	6	650	650	650
Bannock	1	5	650	650	650
	1	6	750	750	750
DEER LODGE CO.					
Flint Creek	2	30	750	650	683
McClellan Gulch	1	5	600	600	600
Silver Creek	1	40	650	650	650
Summit Valley	8	93	750	350	675
JEFFERSON CO.					
Cedar Plains	1	10	600	600	600
Elkhorn	1	5	650	650	650
	1	20	650	650	650

a Estimated.

TABLE CV.—STAMP BATTERIES: NUMBER AND WEIGHT OF STAMPS—Continued.

County and district.	Number of mills reported.	Number of stamps.	Maximum weight of stamps.	Minimum weight of stamps.	Average weight of stamps.
MONTANA—Continued.					
LEWIS AND CLARKE CO.			*Pounds.*	*Pounds.*	*Pounds.*
Silver Creek	1	35	750	600	675
Stemple	3	25	700	650	670
MADISON CO.					
Hot Spring	1	5	650	650	650
Mineral Hill	1	10	800	800	800
Silver Star	2	24	668	550	539
NEVADA					
ELKO CO.					
Columbia	1	10	1,010	1,010	1,010
Tuscarora	3	40	800	750	787
ESMERALDA CO.					
Black Mountain	1	10	800	800	800
Columbus	2	40	900	900	900
Esmeralda	1	8	400	400	400
Oneota	1	4	650	650	650
Pine Grove	2	20	750	750	750
Wilson	1	10	800	800	800
EUREKA CO.					
Cortez	1	10	700	700	700
HUMBOLDT CO.					
Mount Rose	2	20	950	950	950
Sierra, or Dun Glen	1	10	750	750	750
Winnemucca	2	20	800	750	775
LANDER CO.					
Lewis	1	15	850	750	800
Reese River	1	20	1,000	1,000	1,000
LINCOLN CO.					
Eldorado	1	15	700	700	700
Ely	1	30	750	750	750
Pahranagat	1	10	*a* 750	*a* 750	*a* 750
NYE CO.					
Morey	1	10	800	800	800
Tybo	1	20	700	700	700
Union	1	40	850	850	850
STOREY, LYON, AND ORMSBY COS.					
The Comstock	14	469	984	750	830
WHITE PINE CO.					
Cherry Creek	1	20	*a* 800	*a* 800	*a* 800
Ward	1	20	750	750	750
White Pine	1	30	750	750	750
NEW MEXICO.					
DOÑA AÑA CO.					
Hillsborough	1	10	550	550	550
GRANT CO.					
Silver Flat	2	20	750	600	675
NORTH CAROLINA.					
GASTON CO.					
....	1	40	750	750	750
GUILFORD CO.					
....	1	5	850	850	850
NORTH CAROLINA—Cont'd.					
MECKLENBURG CO.			*Pounds.*	*Pounds.*	*Pounds.*
Capp's Hill	1	10	650	650	650
Sixth	1	10	750	750	750
MOORE CO.					
....	1	10	750	750	750
OREGON.					
BAKER CO.					
Burnt River	2	10	750	600	675
Conner Creek	1	20	600	600	600
Rye Valley	1	5	900	900	900
....	1	20	650	650	650
GRANT CO.					
Granite	1	10	800	800	800
UTAH.					
JUAB CO.					
Tintic	2	42	750	750	750
SALT LAKE CO.					
West Mountain	1	20	650	650	650
SUMMIT CO.					
Uintah	1	40	850	850	850
TOOELE CO.					
Camp Floyd	1	20	750	750	750
Rush Valley	1	5	*a* 750	*a* 750	*a* 750
UTAH CO.					
Tintic	1	10	750	750	750
WASHINGTON CO.					
Harrisburgh, or Silver Reef.	4	30	850	650	758
VIRGINIA.					
BUCKINGHAM CO.					
Curdsville	1	5	750	750	750
CULPEPER CO.					
....	1	10	550	550	550
FAUQUIER CO.					
....	1	10	650	650	650
LOUISA CO.					
Coohoo	1	10	650	650	650
STAFFORD CO.					
Hartwood	1	10	750	750	750
WYOMING.					
CARBON CO.					
Douglass	1	10	640	640	640
SWEETWATER CO.					
California	5	50	850	350	610
Miners' Delight	3	50	800	750	770

a Estimated.

TABLE CVI.—STAMP BATTERIES: NUMBER AND WEIGHT OF STAMPS—RECAPITULATION BY STATES AND TERRITORIES.

State or territory.	Number of mills reported.	Number of stamps.	Maximum weight of stamps.	Minimum weight of stamps.	Average weight of stamps.	State or territory.	Number of mills reported.	Number of stamps.	Maximum weight of stamps.	Minimum weight of stamps.	Average weight of stamps.
			Pounds.	*Pounds.*	*Pounds.*				*Pounds.*	*Pounds.*	*Pounds.*
Total	280	5,367	1,050	300	689	Maine	1	10	750	750	750
						Montana	26	319	800	350	608
Alabama	2	16	400	300	337	Nevada	43	901	1,010	400	820
Arizona	19	202	1,000	350	781	New Mexico	3	30	750	550	633
California	43	1,031	950	500	733	North Carolina	5	75	750	350	710
Colorado	34	985	750	400	559	Oregon	6	65	900	600	681
Dakota	23	840	800	550	697	Utah	11	167	850	650	703
Georgia	26	288	1,050	300	508	Virginia	5	45	750	550	661
Idaho	24	283	900	550	674	Wyoming	9	110	850	350	686

Total weight of stamps, 3,701,204 lbs.=1,850$\frac{1204}{2000}$ tons.

WEIGHT OF SHOES AND DIES.—The weight of the shoe bears a certain relation to the weight of the stem, tappet, and boss. For a stamp which weighs 800 pounds the shoe should weigh about 125 pounds and the die say 100 pounds. Some millmen advocate a very thick die; but the one great objection to this plan is that the loss of crushing power when the level of the die surface is worn much below the discharge more than compensates for the gain in time and material in changing dies less often, and if the thick dies are set high at first the breakage of screens is much above the usual amount.

MATERIAL OF SHOES AND DIES.—Cast iron is still much more frequently used than steel for shoes and dies. Thus, of 253 mills reported, only 23 use steel exclusively and 11 both steel and iron for this purpose, or 34 use steel altogether or in part, while in 219 mills iron is the material employed. Among the best equipped mills, however, the proportion of steel shoes and dies is undoubtedly higher than is shown by these figures, for among the 34 mills using steel are some of the finest of the recently constructed works.

On the first introduction of steel for this purpose great expectations were aroused among millmen as to its superiority, but, unfortunately, many of the earlier castings sent out proved defective, and a reaction of opinion set in. The difficulty with the first steel shoes and dies seems to have been that, although they outlasted the iron shoes in the ratio of 2 to 1, many of them wore cup-shaped and chipped; but this may have been owing to the fact that the outside of steel shoes and dies, on casting, cooled more rapidly than the inside, and thereby became harder. The steel shoes and dies at present manufactured are, however, of much better quality, and usually wear very evenly, the improvement being due to a better knowledge of the requirements and greater care in casting.

The question of steel *versus* iron for shoes and dies is purely one of relative economy, and not of theoretical merits of one metal above the other. Steel lasts longer than iron, but costs proportionately more; on the other hand, a less frequent adjustment of drop is required with the more slowly-wearing material. In districts remote from railroad communication, and where the freight charges are high, steel is more economical than iron; for the first cost of material becomes a matter of less moment when the expense of transportation reaches from 3 to 5 cents per pound, as is sometimes the case. If it requires two sets of iron shoes and dies to do the work of one set of steel ones, evidently it will cost twice as much in freight charges for this item to keep a mill supplied with the former.

In many districts there are founderies, sometimes connected with mining works, which recast the worn shoes and dies, thus utilizing the old iron, but have not the facilities for producing perfect steel castings. This consideration, no doubt, has weight in the choice of material. There is also no market for the old steel, while scrap-iron, on the other hand, can be readily disposed of in most localities. In dry-crushing mills, running on very hard rock, steel is generally preferred.

The Table CVII shows the prevalent practice in the several states and territories, the reports from 253 works being grouped under three heads—mills using cast iron only, mills using steel only, or mills using both iron and steel for shoes and dies.

TABLE CVII.—STAMP BATTERIES: CHARACTER OF SHOES AND DIES.

State or territory.	Number of mills reported, with character of shoes and dies specified.	Number of mills using iron shoes and dies.	Number of mills using steel shoes and dies.	Number of mills using both iron and steel shoes and dies.
Total	263	219	23	11
Alabama	2	2		
Arizona	17	12	4	1
California	43	31	8	4
Colorado	34	30		a 4
Dakota	23	22	1	
Georgia	26	24	2	
Idaho	20	18	2	
Maine	1	1		
Montana	26	25	1	
Nevada	24	19	3	b 2
New Mexico	3	3		
North Carolina	5	5		
Oregon	6	6		
Utah	9	8	1	
Virginia	5	4	1	
Wyoming	9	9		

a One mill uses steel dies and iron shoes, and one mill iron dies and shoes of both steel and iron.
b One mill uses iron dies and steel shoes.

The iron in use is nearly always a mixture of white and gray iron in different proportions, and is sometimes chilled and sometimes not. Of 39 mills reporting the use of chilled-iron shoes and dies specific description was received from 25, as follows, without any attempt at classification further than to quote the wording of the reports:

Gray	1	White	15
Mottled	1	White double chilled	1
Nearly mottled	1	Hard (white?)	1
White mottled	1	Common cast	2
White surface-chilled	2		

No details were received from 14 other mills reporting the use of chilled iron for this purpose. In 143 reports, in which it was not stated whether the castings were or were not chilled, the following varieties are included:

Very soft gray	1	100 parts white mottled, 25 gray, and 15 wrought iron	1
Very soft (gray?)	1	One-third No. 1 (gray) and two-thirds scrap	4
Soft gray	1	Hard white	5
Gray	3	White Scotch pig	2
Mottled	12	Woodstock white charcoal cold blast	1
Common cast	12	Car-wheel iron	1
Mottled nearly white	1	Scrap	3
White mottled	13	Hard scrap	1
Common cast low white	1	Scrap and pigs	2
Nearly white	22	Franklinite (mixed?)	8
Medium white	1		
White	47		

MATERIAL OF TAPPETS, ETC.—Steel has been introduced as the material for tappets, cams, bosses, etc., as well as for shoes and dies. It has been found to work excellently for these purposes.

LEVEL OF THE DIE BELOW DISCHARGE.—In gold mills where amalgamation is carried on in the battery the level of the die-surface is kept from 2 to 6 inches below the discharge in order to amalgamate as much gold as possible in the mortar. By means of a movable slide at the bottom of the screen the level of the discharge can be kept the same as the die wears out.

In silver mills which crush wet it is intended to keep the die-surface about 1 or 2 inches below the level of the discharge; but as the die wears down its surface is gradually lowered, and this decreases the crushing power of the battery. It is the aim of the mill man to set his die as high as possible, without endangering his screens, so as to utilize most of his die to the best advantage. In dry-crushing silver mills this is also the case, except that there is a greater loss of power in the dry than in the wet-crushing battery when the die-surface is much below the level of the discharge.

GUIDES.—Many varieties of guides have been experimented on, but the prevailing practice is to use plain wooden ones. The wood chosen is generally oak, though other hard woods answer the purpose; sometimes soft wood is employed, because of the difficulty of obtaining other kinds. At the California mill, Virginia City, both oak and hickory are used, and heart of maple, when obtainable, is recommended by some millmen. At Austin, Nevada, the cast-iron guides have square holes, which are filled with pine blocks with the grain vertical and bored out to the proper size for the stem. These blocks can be replaced at any time, and only one stamp need be hung up. At the Buffalo mill, at Atlanta, Idaho, guides of yellow pine and fir, the only woods in the district, having given much trouble by wearing out very rapidly, the superintendent, to avoid the necessity of transporting solid guides of hard wood from a distance, hit upon a similar expedient. Oak keys, four for each bearing and with the grain vertical, were mortised into the fir guide-frames, with the bearing surface turned concave to fit the stem. These keys are replaced by new ones without disturbing the frames or removing a bolt.

Some of the cast-iron guides in use are lined with babbitt-metal, and at one mill in Utah a lining of rawhide is adopted. In dry-crushing batteries babbitt-metal and brass are cut by the quartz dust. In one Nevada mill brass guides are used with part of the batteries, while iron is used in the rest; but the result of this trial, side by side, of the two styles is not reported. One mill in Colorado and one in Georgia have upper guides of wood and lower ones of iron. The following table shows the proportionate number of mills fitted with the different kinds of guides, from which it appears that in 244 of the 257 mills reported wooden guides are used exclusively:

TABLE CVIII.—STAMP BATTERIES: CHARACTER OF GUIDES.

State or territory.	Number of mills reported, with character of guides specified.	Number of mills using wooden guides.	Number of mills using iron guides.	Various.
Total	257	244	9	4
Alabama	2	2		
Arizona	19	18	1	
California	43	42	1	
Colorado	34	33		1
Dakota	23	23		
Georgia	26	24	1	1
Idaho	20	20		
Maine	1	1		
Montana	26	26		
Nevada	25	21	3	1
New Mexico	8	8		
North Carolina	5	5		
Oregon	6	6		
Utah	10	9		1
Virginia	5	5		
Wyoming	9	6	3	

CAMS.—Double-armed cams are now used almost exclusively, but in a few of the older mills single-armed cams are still found. The advantage of the double-armed cam is obvious; the cam-shaft, which is subject to an enormous strain, can be run at one-half the speed necessary to obtain the same number of drops per minute with single cams. The attempt to carry this principle still further by adopting three-armed cams has not been successful, as the tappet has not room, in falling, to clear the rising cam-tips safely, while the latter could not be shortened without assuming a less advantageous curve.

MAXIMUM DROP.—The maximum drop obtainable is not the same in silver and gold mills, as the stamps of gold mills should drop farther than those of silver mills, in order to produce the splash which assists in battery amalgamation. In gold mills the maximum drop obtainable is from 12 to 20 inches, and in silver mills from 9 inches upward.

WIDTH OF MORTAR.—The width of the mortar at the bottom of the screen varies from 10 to 14 inches and depends upon the hardness of the ore. At the California mill, where the quartz is very friable, the width of the mortar is only 10½ inches, although the stamp weighs nearly 1,000 pounds and the shoe has a large diameter. Such a mortar could hardly be used for hard quartz, as the screens would be continually in danger. Of course the narrower the mortar the faster the battery will discharge.

DOUBLE AND SINGLE DISCHARGE.—In gold mills where battery amalgamation is practiced the discharge is always single, as one screen will discharge the ore as fast as it is amalgamated. In wet-crushing silver mills, where the ore is hard, it is also usually single, as the battery will discharge from a single screen nearly as fast as the ore is crushed. At the California mill, although the ore is soft, a single discharge is used; in other places, as, for instance, Silver Reef, Bodie, and Tombstone, a double discharge gives better results than a single one. Comstock men think there is no advantage in a double discharge, as, no matter how soft the ore, the water will carry it out of the battery as soon as crushed. This, however, is doubtful. In Silver Reef, where 10 tons to the stamp are crushed per diem, it is unquestionably advantageous to use a double discharge, and it is extremely doubtful whether power is not always saved and the percentage of slimes diminished by this construction. Local conditions must decide whether the saving effected is counterbalanced by the inconvenience of a double set of sluices and an increased consumption of screens.

For dry-crushing mills a double discharge is necessary, for the ore will not screen nearly so fast as it is crushed. The main point in adjusting a dry battery to crush properly is to make the splash of the pulp as regular and as high as possible, and this is done by dropping the stamps in the proper order, so that the material being crushed shall be evenly distributed in the mortar, and by keeping as little of it as possible under the stamps at one time. Where there are 3 inches or so of ore under the stamps there is no chance for the splash; very little pulp passes through the screens, and the battery is soon choked and fills up.

SCREENS.—There are four kinds of screens in use for quartz-mills: steel and brass wire, and slot and needle-punched sheet-iron. A tin screen is mentioned in the schedule of a gold mill in California, but the advantages are not apparent. For gold mills the punched-slot screen is the one mostly used, though steel wire is sometimes met with; in wet-crushing silver mills steel and brass wire and needle or slot-punched iron are all used. The preference seems to be for the brass wire, though steel is now replacing it. Punched screens are not so much used as formerly. At the California mill they use a horizontal-slot punched screen, which is said to correspond to a No. 50 wire. This may be the case when the screen is new, but after it is slightly worn, as the tank sand shows, it certainly lets through much coarser pulp. These screens are said to work exceedingly well, and are the invention of the superintendent of that mill. In dry-crushing mills the brass and steel wire screens are used.

In gold mills, from No. 5 to 9 slot is the common size; in silver mills, wet- and dry-crushing, from No. 40 to 60 wire screens, though the usual number is 40. At Bodie (wet-crushing) they use as low as No. 20, and at the Manhattan and Eberhardt & Aurora (dry-crushing) they use No. 60. No account of any concave screens is included in the census schedules.

The numbers by which punched screens are known correspond to those of ordinary sewing-needles, from which the punches are made. The width of the slots in slotted screens is equal to the diameter of the holes in a punched screen of the same number. The number of a wire screen is the number of meshes to the linear inch.

SELF-FEEDERS.—The automatic ore feeders employed are the Hendy, Tulloch, Stanford, Victor, and some others, and they seem to be used irrespective of whether the battery is a wet or a dry crusher. The accompanying table shows the number and the character of the self-feeders included in the plant of the mills reported and the extent to which these appliances have been introduced.

TABLE CIX.—PROPORTIONAL NUMBER OF MILLS IN WHICH AUTOMATIC FEEDERS ARE USED.

State or territory.	Number of mills reported.	Number using automatic feed.	Number fed by hand.	Unspecified.
Total	272	115	154	3
Alabama	2	....	2	....
Arizona	20	17	3	....
California	43	26	16	1
Colorado	34	5	29	....
Dakota	23	11	12	....
Georgia	26	1	25	....
Idaho	13	2	11	....
Maine	1	1	....	....
Montana	26	*a* 6	20	....
Nevada	44	31	11	2
New Mexico	3	3	....	....
North Carolina	5	....	5	....
Oregon	6	5	1	....
Utah	12	7	5	....
Virginia	5	....	5	....
Wyoming	9	....	9	....

a Including one mill in which, in addition to automatic machines, feeding by hand is also practiced.

An automatic quicksilver-feeder, for gold mills, acting on the principle of the ore-feeders, is a new feature. It is so connected with the cam shaft that each revolution of a large ratchet wheel, which can be regulated to revolve once in one, two, three, or four minutes, as desired, dips a small cup into a reservoir of quicksilver, and by a trip of its cam empties the contents into a pipe leading to the mortar. The feed is therefore regular and practically continuous.

PLATES.—Gold mills are usually provided with a copper plate, or with a silver-plated copper plate which has been amalgamated with quicksilver, for the purpose of catching the gold or the amalgam as it comes from the battery. This plate is called an apron, and is as wide as the discharge of the battery, being sometimes rectangular and sometimes tapering toward the sluice. Millmen seem to differ in regard to the respective virtues of copper plate and silver-plated copper plates. Some think that the silver plate, after amalgamation, is too "hard" to catch the amalgam; others (and these are the majority) prefer it, and its use is becoming more general. A silvered plate is much more readily amalgamated and kept clean.

BATTERY POWER.—As a rule, the whole mill is run by one engine or wheel. The California mill is an exception, for the battery of 80 stamps is situated a quarter of a mile from the pan mill, to which the pulp flows through a long sluice. The settling tanks are in the pan mill.

ROCK-BREAKERS.—Blake's (or some modification of it, for the patent has lapsed) is the rock-breaker in use in most mills. Some have none at all, the rock either being fine enough for the battery as it comes from the mine or being broken by hand when necessary. In a few mills Chinese are employed to break rock for the batteries. The following table shows the number of mills reported as having rock-breakers, and the number of the latter in use. It will be observed that the proportion is very slightly over one per mill, except in the case of the Dakota mills:

TABLE CX.—NUMBER OF MILLS REPORTED AS HAVING ROCK-BREAKERS, AND NUMBER OF ROCK-BREAKERS IN USE.

State or territory.	Number of mills reported as having rock-breakers.	Number of rock-breakers reported.
Total	128	164
Arizona	13	16
California	12	18
Colorado	13	18
Dakota	11	29
Idaho	11	11
Maine	1	1
Montana	8	8
Nevada	32	34
New Mexico	3	3
North Carolina	1	1
Oregon	2	2
Utah	17	19
Virginia	4	4

DRYING THE ORE.—In dry-crushing mills the ore is generally dried on cast-iron plates set over the flues of the roaster, where there is one; where no roaster is used, the plates are heated by a special fire. The Pacific rotary drier is in use in some mills, and consists of a revolving cylinder, through which the ore passes continuously at the same time that the heated air from a special fireplace passes through the cylinder in the opposite direction. The cylinder is set at a slight angle from the horizontal, and acts somewhat on the principle of the Howell furnace. This rotary drier seems to work very well, and certainly produces a great saving of labor, for in drying the ore by the ordinary method on plates it is necessary to turn it over frequently by hand. This mechanical process of drying, however, fails to utilize the waste heat from the roasting furnace, in case one is employed, which fact renders the rotary drier less profitable than it otherwise would be.

MORTAR CASTINGS.—Mortars are now almost invariably made in one solid casting. The practice of casting them in several pieces, to be bolted together, is justified only when great difficulties in transportation are to be overcome. If the only possible means of conveying machinery to a district difficult of access is by pack-train, the separate portions must not be heavier than from 300 to 350 pounds each. Foundery men, in supplying plant for remote districts, as, for example, Mexico and Central America, have shown great ingenuity in furnishing sectional castings. This plan is available in the case of many items of mining and milling machinery, but for mortars single castings are so much stronger that piecing should, if possible, be avoided. The strain upon a mortar, in case the mortar-bed settles in the least degree irregularly, requires corresponding rigidity in the casting; and the pieced mortars cannot be relied upon with confidence, notwithstanding the care taken to make them as efficient as the circumstances admit. Of some 250 mills from which data on this point were obtained not one had a sectional mortar.

DROP.—The length of the "drop" for dry-crushing batteries is usually from 6 to 8 inches, and in wet-crushing from 6½ to 9 inches. The drop for gold-mill batteries is greater than for silver, as it seems that more splash is required where amalgamation is carried on in the mortar than where it is only necessary to pulverize the rock. The greatest drop noted for silver mills is that of the Martin White battery, at Ward, Nevada, where a 750-pound stamp drops 9 inches ninety times a minute.

ADJUSTMENT OF DROP, SPEED, AND WEIGHT OF STAMPS.—Engineers have not been able to deduce any general law governing the length of drop and the number of drops per minute for a given weight of stamps and ore of a certain texture. The fact that in many batteries the weight of stamp, drop, and speed are not the best for the kind of ore that is to be crushed is admitted. It is, however, much easier to condemn the arrangement of a particular battery working on a given ore than it is to say exactly what drop and what number of drops would do the most work with the least proportionate wear and tear. It is a self-evident proposition that the heavier the battery, the greater its drop and the number of drops, the more ore will be crushed. Of course these have their limits in practice. If a 750-pound stamp, dropping 6 inches one hundred times per minute, crushes a given kind of ore at a certain rate, for a harder ore a heavier stamp would be required, or a greater drop, or more drops per minute, and perhaps all three, in order to crush the same amount of ore in a given time. Taking the wear and tear into account, it would seem that if the stamps be made heavier the drop, as well as the number of drops, should be decreased, in order to keep the wear and tear of a battery down to a minimum. The converse of this would be true of a lighter one.

In practice, however, these factors are governed by other considerations. Suppose a company has 15 tons of ore to be crushed in 24 hours and it is intended to put up a battery of 15 stamps, weighing 700 pounds each, to crush this ore; it would be much more advantageous to build a 10-stamp battery of say 1,000-pound stamps (granting that this battery would crush the given quantity of ore, which it would probably do), for the 10-stamp battery would cost about one-quarter less, and the amount saved in cost of plant would more than compensate for the proportionate difference in wear and tear between the two, even taking it for granted that, in point of fact, the wear and tear of the lighter battery would be less per ton of ore crushed.

It is considered a good rule to put up a heavy battery, no matter what may be the character of the ore to be crushed; to give that battery as great a speed as possible, say 100 drops per minute, and to give the stamps as long a drop as their weight and speed will permit. Of course there are some limits to be placed on weight, speed, etc. Stamps ought not to exceed 1,000 pounds in weight; the speed ought not to be over 110 and length of drop not over 8 inches for wet-crushing silver mills. For dry-crushing mills these figures, with the exception of speed, should be somewhat less. The greatest speed of any battery reported is 108, in the Eberhardt & Aurora mill, in White Pine, Nevada, the stamps weighing 750 pounds, and the battery being a dry-crusher with 8-inch drop. Gold-mill batteries are run at a less speed and with a greater drop, other considerations than mere crushing-power, such as amalgamation, altering the usual rules for wet batteries. The Rising Sun mill, in Placer county, California, for instance, has, with 800-pound stamps, a speed of 65 and a drop of 11 inches—the longest noted.

STEAM STAMPS.—A single steam 2-stamp battery is reported in Arizona. Not much progress has been made in the introduction in the far West of steam or of air-cushions to increase the force of the blow given by the stamp in falling. For the copper ores of the Lake Superior region the steam stamp has been found to be very effective, and it would seem that it should be equally so in the case of very tough gold and silver ores, though no sufficient tests have yet been made to afford a basis for comparison with the ordinary gravity stamps. The whole question is, of course, one of relative economy; if the first cost, wear, and repairs of the steam stamp overbalance its superior crushing capacity, it would not be practicable to adopt it.

THE ORDER IN WHICH THE STAMPS FALL.—The order in which the stamps fall has a great influence upon the crushing power of the battery. Four orders are in general use, namely: 1–4–2–5–3, 1–5–2–4–3, 1–3–5–2–4, and 1–3–5–4–2. Many other orders are used, but they are usually adopted through ignorance of the proper order. Millmen do not agree as to which one of the above-named four orders is the correct one; it is a safe rule to lay down, however, that no one of the five stamps after falling should be followed by either of its neighbors. This is the case with the second and fourth orders mentioned, and although they do not give the pulp a bad motion, the first and third orders are preferable, and of these two the first seems to be the best. This opinion is borne out by the usage in most places, many more batteries dropping in the order 1–4–2–5–3, although 1–5–2–4–3 is the order in the California mill batteries and many others.

There is generally one cam shaft for every ten stamps, the second row of stamps having their cams so set that no two stamps fall at the same instant.

In dry-crushing batteries it is often customary to give the outside stamps ½ inch more drop than the others in order to prevent the pulp from drifting too much to the corners.

ROTATION OF STAMPS.—The rotation of the stamp in rising is governed by the amount of grease upon the cam and the tappet. Where the cam and the tappet are dry, the stamp makes nearly a complete revolution, and where there is much grease it scarcely revolves at all. From one-twelfth to one-eighth of a revolution is about what the stamp ought to revolve in rising, as this will be sufficient to cause the shoes and dies to wear evenly and at the same time permit of the cam and the tappet being well lubricated. The rotation in falling is usually very slight.

MINERALOGICAL CHARACTER OF THE ORE.—The mineralogical character of the ore has considerable influence on the quantity crushed. As a rule, quartz ores are more readily pulverized than limestone, which, though softer, is less brittle, and oxidized or chloride ores easier than sulphurets. The least difficult ore to crush is the Silver Reef sandstone, and the hardest the flinty quartz of some of the Arizona mines.

AVERAGE NUMBER OF TONS CRUSHED.—In wet-crushing the number of tons pulverized varies from 1 to 5 in twenty-four hours, according to the character of the ore, and in dry-crushing from three-quarters of a ton to 2 tons to the stamp. At Silver Reef, however, no less than 10 tons of sandstone per stamp are passed through a wet battery. This rock requires to be disintegrated rather than crushed, the individual particles of quartz composing it being small enough to pass through the screens without further division.

AMALGAM CAUGHT IN THE BATTERY.—The data in the schedules in regard to the amount of amalgam caught in the batteries of gold mills are rather vague. From one-third to two-thirds of the total amount of amalgam saved is caught in the battery, and the usual amount is over one-half. There is ordinarily little silver saved, except that which is alloyed with the gold. Battery amalgamation in silver mills, which was formerly common, is now seldom practiced.

GOLD AND SILVER IN BATTERY AMALGAM.—The gold and silver in the battery amalgam of gold mills form from one-third to one-half of the weight of the amalgam; the more silver the gold contains and the finer it is the more quicksilver does it require for amalgamation. The gold bullion from this mixture is worth from $12 to $18 per ounce, showing that the proportion of silver in the alloy is very small. Scarcely any other metals are found in this bullion with the exception of a little lead or copper. The fact must not be lost sight of that in the amalgamation of gold that metal is not so perfectly alloyed with the quicksilver as is the case with silver, but is merely coated on the outside with a thin film of quicksilver, which causes the separate particles of gold to adhere together, thereby forming a pasty mass which has the appearance of a true amalgam. This is easily perceptible where the gold is coarse, as the particles can be distinctly felt by pressing the amalgam between the fingers. In the amalgamation of gold it is necessary to keep the amalgam at a certain consistency. If there is too much quicksilver, it will flow too easily and not be caught on the amalgamated copper or silver plates; and if there is not enough, the gold is imperfectly amalgamated and is carried off in the tailings.

WEAR AND CONSUMPTION.—Cams last from one year to three years, the length of time depending on many things, but principally upon the care with which the battery is handled. At the California mill, where especial pains is taken to keep account of the wear and tear, experience shows that the usual life of a cam is sixteen months. Cams usually break at the hub, but sometimes the extreme tips break off. It is said that in certain European works where the stamp-battery has been introduced it has been abandoned on account of the rapid wear of cams. Inappropriate handling only can explain such a result.

Other things being equal, the length of time which shoes and dies last depends upon their dimensions. A frequent replacement of those portions of the battery causes a serious interruption of work and a rapid accumulation of old iron, while the battery is less efficient with old shoes and dies than when provided with such as are new or but little worn. The usual practice is to employ shoes and dies which will last one or two months in wet-crushing and from two to four months in dry-crushing.

Shoes wear much faster than dies, because the latter are always covered with a layer of rock or pulp, the impact being between the shoe face and the ore. The exact relation probably depends upon the feeding, a thick layer of ore protecting the die more effectually than a thin one. From a half to two-thirds as much iron is worn from the dies as from the shoes in a given time. In many mills the dimensions of the shoes and dies are so selected that they will wear out in the same time, and this plan certainly seems commendable, since only a simple stoppage is required to reset the battery instead of two. Shoes and dies are commonly used till their weight is diminished to from 25 to 50 pounds. Occasionally, however, they break in the battery, and must then, of course, be removed, however little they may be worn.

The consumption of iron in the battery per ton of ore crushed depends upon a number of points, some of which are as follows: The material employed for the castings chips if too hard and wears rapidly if too soft. Steel generally lasts about three times as long as iron. The character of the ore is perhaps less strikingly influential on the consumption of iron than might have been supposed. The Comstock ores, for example, are usually in a fine state of division as they leave the mine; yet the shoes and dies of the California mill do not wear unusually long. The fact is, that comparatively little of the Comstock ore will pass the screens without further division, and the large particles are as hard to crush as any quartz. The Silver Reef ores, as has already been pointed out, are exceptional. The level of the die surface has much to do with the consumption of iron. In the gold mills of California and Colorado, where battery amalgamation is practiced, the freedom of discharge is intentionally diminished by setting the die surface several inches below the lower edge of the screen. The wear of iron is consequently greater in these mills than in wet-crushing silver-stamp batteries. The size of the openings in the screens, and no doubt also their number, has an influence on the wear of the iron corresponding to the work done in reducing the pulp to the necessary fineness. Dry-crushing batteries certainly consume less iron than wet-crushing batteries per ton of ore crushed, probably because in wet batteries the particles of dust are removed from the face of the iron as fast as formed, allowing the die to descend immediately upon the larger fragments. The modifying conditions are so numerous that accurate data as to the consumption of iron in the batteries cannot be given. The wear per ton of ore crushed in dry batteries, however, sometimes falls a little below one pound per ton, and may average a pound and a quarter, while in wet-crushing over two pounds is not uncommon, and a pound and three-quarters is not far from the mean.

Screens last from five to thirty days in wet-crushing and from seven to forty days in dry. The average for both wet- and dry-crushing mills is about fifteen days where punched screens are used in wet mills and brass wire in dry, for in wet-crushing the punched screen will last one-third longer than the brass-wire screen.

QUICKSILVER USED IN BATTERY.—In battery amalgamation the amount of quicksilver added depends on the quantity of gold in the ore. It is necessary to add quicksilver enough to amalgamate the gold and still keep it in that pasty condition which causes it to adhere to the copper or silver plates. From one to two ounces of quicksilver per ounce of gold in the ore is about the usual quantity used.

WATER USED IN THE BATTERY.—It is generally difficult to ascertain the quantity of water used in batteries, but some trustworthy data show a consumption of from 500 to 700 gallons per ton crushed. According to the circular of Prescott, Scott & Co., of San Francisco, each stamp requires 10 pounds per minute, which, at the rate of 3 tons crushed per stamp in twenty-four hours, would make 576 gallons per ton. Many mills, however, use much more water. Where water is scarce, it is usual to pump it back from the slime tanks or ponds to the battery.

SHAPE OF OLD SHOES AND DIES.—Shoes generally wear slightly concave in the center, the outer rim being rounded or chipped off. Dies usually wear a little convex in the center, gradually becoming somewhat concave toward the outer edge. These depressions commonly vary from half an inch to 1 inch.

METHOD OF HANDLING DRY-CRUSHED ORE.—In almost all dry-crushing mills where the ore is roasted the ore is removed from the screens by means of a screw-conveyor. Where no roasting is done the "pulp" (a) is removed in cars, which are placed to catch it as it falls from the battery, and is thus carted to the pans.

GRADE OF SLUICES.—The grade of sluices from the battery to the settling tanks is usually from three-quarters of an inch to 2 inches per foot; 1 inch to the foot is sufficient, except with very heavy ores, and more than 2 inches is probably never necessary. Where higher grades are employed, as they often are for short sluices, it is not because they are necessary, but because the disposition of the apparatus makes it convenient. The California mill at Virginia has a sluice 1,200 feet long with an average grade of five-eighths of an inch. It is probable that a somewhat steeper grade would have been adopted had circumstances permitted, but the fact that so low a grade answers the purposes of a highly-successful mill shows that high grades are indispensable only with very heavy ores. These data apply for the normal quantity of water fed to the batteries. It is worth while to remember, however, that the carrying power of water varies with the sixth power of the velocity, and the velocity varies as the square root of the fall. The transporting power of a current, therefore, varies as the third power of the fall. The slower the current in the settling tanks, too, other things being equal, the more complete the separation of the pulp from the water or the smaller the quantity of slimes leaving the mill. These facts show that a small consumption of water is desirable, and indicate, as the most economical practice, batteries with great discharging capacity fed with as little water as is compatible with efficient discharge, sluices of ample grade, and comparatively few settling tanks.

SETTLING TANKS.—Settling tanks vary in size, but the greater part of them are 6 feet long by 5 feet wide by 38 inches deep. The pulp generally fills them to a depth of about 24 inches, which represents about 3 tons of dry ore. A few tanks are made much larger, the maximum reported being 12 by 15 feet with a depth of 3 feet, and the minimum 4 by 4 feet with a depth of 32 inches.

PROPORTION OF SLIMES.—From 1 to 15 per cent. of the ore passes out of the mill in the form of slimes. Ores carrying clay produce far more slimes than those from which it is absent, and of course fine crushing and insufficient tank capacity increase the amount of ore which fails to settle in the tanks. Slimes from doré ore are usually poorer in silver and richer in gold than the ore itself. The treatment of the slimes, if they are saved in settling ponds, forms the subject of special manipulation; but there are many mills at which no attempt is made to collect and work the slimes.

PAN AMALGAMATION.

PLANT.—The number of pans required in a mill in proportion to the number of stamps is governed by the character of the battery (whether it is wet or dry), the quantity of ore crushed by the battery in a given time, the time required to amalgamate the ore, and the size of the pans to be used. In wet-crushing mills the proportion varies from two pans to every five stamps to two pans for every stamp. This latter is the case at the Leeds mill, at Silver Reef, in Utah. Generally from two to three pans, holding from 3,000 to 4,000 pounds, are required for every five stamps. In dry-crushing two pans to five stamps are always sufficient unless the pans are small or the ore is very easily crushed, and sometimes one large pan can be made to do the amalgamating for five stamps. In mills recently built one settler is allowed for every two pans. In the Star mill, at Cherry Creek, and at the Manhattan, in Austin, there is a settler for each pan. This is probably unnecessary, as roasted pulp is very easily settled.

(a) The word "pulp" is used by millmen to designate pulverized ore, whether wet or dry, roasted or raw.

Agitators are in use on the Comstock, at Bodie, and in some other placers, but they are not generally employed. The object in using them is not only to catch the amalgam which may sometimes escape the settler, but to save the coarse sand or concentrations which have been but imperfectly worked in the pans. The agitator is also a convenient receptacle for sweepings and metalliferous refuse. Where agitators are used the proportion is one agitator to every five or six settlers.

A clean-up pan is always needed in a mill, for in it the amalgam from scraping the pans, mullers, and old castings can be separated from the pulp and sand with which it is mixed. It is also very convenient for cleaning quicksilver.

But few mills have pans used especially for working tailings.

Hydraulic strainers are not often found away from the Comstock. They are a great saving in handling large quantities of amalgam.

Usually one retort and one melting furnace are sufficient for a 10-stamp mill.

For raising the quicksilver from the settler to the pan floor pumps, cup elevators, and hydraulic elevators are in use in the larger and better mills. In small and old-fashioned mills the quicksilver is returned to the higher level in buckets by hand; but the hydraulic elevator is superseding all others, because from the simplicity of its construction it is not liable to get out of order. It consists of a strong reservoir, into which, the quicksilver having been admitted, a column of water is turned on, forcing the quicksilver through a pipe to the floor above. If water pressure is not to be had, steam from the boiler, usually supplied at about 90 pounds pressure, will answer the same purpose.

The following tables show the more important apparatus of 128 amalgamating mills in which the pan process is in use. The greater number are silver mills, but some gold mills which have pans, settlers, etc., are included:

TABLE CXI.—AMALGAMATING PLANT OF LEADING MILLS.

ARIZONA.

County and district.	Number of mills reported.	Number of ore pans.	Number of settlers.	Number of agitators.	Number of clean-up pans.	Number of tailings pans.	Number of hydraulic strainers.	Number of quicksilver pumps.	Number of retorts.	Number of melting furnaces.
MARICOPA.										
Globe	2	6	8	......	2	......	......	......	2	2
MOHAVE.										
Cedar Valley	1	2	1	......	1	......	......	......	1	1
Hualapai	1	2	1	......	1	......	......	......	1	1
Owens	1	6	3	......	1	......	......	......	1	1
PIMA.										
Arivaca	2	4	2	......	1	......	......	......	1	1
Tombstone	8	24	12	1	8	......	......	2	6	4
PINAL.										
Globe	1	2	1	......	1	......	......	......	1	1
Pioneer	1	2	1	......	1	......	......	......	1	1
YAVAPAI.										
Peck	1	4	2	......	1	......	......	......	2	1
Pine Grove	1	2	1	......	......	......	......	......	1	1
Tiger	1	4	2	......	1	......	......	......	2	2
......	1	4	2	......	1	......	......	......	3	3

CALIFORNIA.

County and district.	Number of mills reported.	Number of ore pans.	Number of settlers.	Number of agitators.	Number of clean-up pans.	Number of tailings pans.	Number of hydraulic strainers.	Number of quicksilver pumps.	Number of retorts.	Number of melting furnaces.
MARIPOSA.										
Mariposa Estate	1	4	2	......	1	4	......	4	1	1
MONO.										
Bluid Spring	1	1	1	......	......	......	......	......	1	1
Bodie	6	84	43	9	8	......	......	7	11	8
SAN BERNARDINO.										
Clark	2	6	2	......	1	......	......	......	3	2
SHASTA.										
Pittsburgh	1	4	2	1	......	......	......	......	1	1
SISKIYOU.										
Sawyer's Bar	1	(a)	......	......	......	......	......	......	1	1
South fork Salmon	1	......	1	......	(b)	1	......	......	1	1

a Two barrels. *b* One barrel.

TABLE CXI.—AMALGAMATING PLANT OF LEADING MILLS—Continued.

County and district.	Number of mills reported.	Number of ore pans.	Number of settlers.	Number of agitators.	Number of clean-up pans.	Number of tailings pans.	Number of hydraulic strainers.	Number of quicksilver pumps.	Number of retorts.	Number of melting furnaces.
COLORADO.										
BOULDER.										
Grand Island	1	9	5		1				2	2
CLEAR CREEK.										
Griffith	1	4	2		1				2	1
CUSTER.										
Hardscrabble	1	8	4	2					1	1
DAKOTA.										
LAWRENCE.										
Bear Butte	1	4	2	1	1				1	1
Whitewood	1	4	2		1				1	1
GEORGIA.										
LUMPKIN.										
Twelfth	1	6							2	
IDAHO.										
ALTURAS.										
Bonaparte	1	12	*a* 7		1				1	
Middle Boisé	5	22	13	1	3				3	2
Rocky Bar, or Bear Creek	1		1							
BOISÉ.										
Banner	1	4	3		1				1	1
Elkhorn	1	1								
Granite	1	4	2							
LEMHI.										
Yankee Fork	2	9	5		1				4	1
OWYHEE.										
Carson	7	51	26	7	6				11	5
Wagontown	1	4	3	1	1				1	
MAINE.										
HANCOCK.										
Sullivan	1	2	1		1				1	1
MONTANA.										
BEAVER HEAD.										
Bald Mountain	1	2	1				2		1	1
DEER LODGE.										
Flint Creek	2	12	6	3	2				3	4
Summit Valley	8	36	18		6		4		11	10
JEFFERSON.										
Elkhorn	1	3	1		1				1	1
LEWIS AND CLARKE.										
Stemple	1	2	1						1	
NEVADA.										
ELKO.										
Columbia	1	4	2		1				2	1
Tuscarora	3	22	11		3			2	7	4
ESMERALDA.										
Black Mountain	1	2	1						2	1
Columbus	2	22	11		2		1	2	7	3
Esmeralda	1	4			1				2	1
Oneota	1	2	1		1				1	1
Pine Grove	1	2	1	3						

a Including 1 clean-up settler.

TABLE CXI.—AMALGAMATING PLANT OF LEADING MILLS—Continued.

County and district.	Number of mills reported.	Number of ore pans.	Number of settlers.	Number of agitators.	Number of clean-up pans.	Number of tailings pans.	Number of hydraulic strainers.	Number of quicksilver pumps.	Number of retorts.	Number of melting furnaces.
NEVADA—Continued.										
EUREKA.										
Cortez	1	3	1	....	1	....	....	....	1	2
HUMBOLDT.										
Mount Rose	2	9	4	....	2	....	....	....	3	3
Winnemucca	1	4	2	....	....	....	....	....	2	1
LANDER.										
Lewis	1	3	2	....	1	....	....	....	1	1
Reese River	1	8	8	....	1	....	....	1	3	2
LINCOLN.										
Eldorado	1	2	1	....	....	....	....	....	1	1
Ely	1	14	7	....	1	....	....	1	6	4
Pahranagat	1	8	4	....	1	....	....	....	1	1
NYE.										
Morey	1	4	3	1	1	....	....	....	1	1
Tybo	1	10	5	....	1	....	....	....	2	2
Union	1	12	6	....	1	....	....	1	2	2
STOREY, LYON, AND ORMSBY.										
The Comstock	17	210	*a* 130	31	26	*b* 73	*c* 2	2	52	*d* 3
WHITE PINE.										
Cherry Creek	1	8	8	....	1	....	....	1	2	2
Ward	1	10	5	....	1	....	....	1	3	2
White Pine	1	16	8	4	2	....	....	....	4	3
NEW MEXICO.										
DOÑA AÑA.										
Hillsborough	1	1	1	1	4	....	....	....	1	....
GRANT.										
Silver Flat	2	12	6	1	2	2	....	....	2	2
NORTH CAROLINA.										
MECKLENBURG.										
....	1	2	1	....	....	....	(*e*)	....	1	1
OREGON.										
GRANT.										
Granite	1	4	2	....	1	....	....	1	2	1
UTAH.										
BEAVER.										
Milford	1	6	3	....	....	....	....	....	1	....
JUAB.										
Tintic	2	11	6	....	1	....	....	....	2	2
SUMMIT.										
Uintah	1	24	12	1	2	....	....	1	4	2
TOOELE.										
Camp Floyd	1	8	4	....	1	....	....	....	2	2
Rush Valley	1	2	1	....	....	....	....	....	1	....
UTAH.										
Tintic	1	4	2	....	1	....	....	....	1	1
WASHINGTON.										
Harrisburgh, or Silver Reef	4	35	18	....	....	....	....	....	9	8
VIRGINIA.										
CULPEPER.										
....	1	5	1	....	....	....	....	....	1	1

a Including tailings and slime settlers.
b Including slime pans.
c California mill only.
d Not including melting furnaces in assay offices.
e One steam strainer.

TABLE CXII.—PAN AMALGAMATION PLANT: RECAPITULATION BY STATES AND TERRITORIES.

State or territory.	Number of mills reported.	Number of ore pans.	Number of settlers.	Number of agitators.	Number of clean-up pans.	Number of tailings pans.	Number of hydraulic strainers.	Number of quicksilver pumps.	Number of retorts.	Number of melting furnaces.
Total	128	853	472	68	111	80	9	26	221	127
Arizona	16	62	31	1	14	....	....	2	22	10
California	13	99	51	10	10	5	....	11	10	15
Colorado	3	21	11	2	2	....	....	....	5	4
Dakota	2	8	4	1	2	....	....	....	2	2
Georgia	1	6	....	....	....	....	....	....	2	....
Idaho	20	107	66	9	13	....	....	....	21	9
Maine	1	2	1	....	1	....	....	....	1	1
Montana	13	55	27	3	9	....	6	....	17	16
Nevada	42	379	230	30	48	73	3	11	105	41
New Mexico	3	13	7	2	6	2	....	....	3	2
North Carolina	1	2	1	....	....	....	....	....	1	1
Oregon	1	4	2	....	1	....	....	1	2	1
Utah	11	90	40	1	5	....	....	1	20	15
Virginia	1	5	1	....	....	....	....	....	1	1

PANS FOR WORKING ORE.—The pans used in amalgamation have a capacity of from 1,500 to 4,000 pounds and are of various construction. One only of them will be described, (*a*) the "combination", which, with its various modifications, is the one most employed in silver mills. When this pan is used for working free ore the sides and bottom (the latter perfectly flat) are cast in one piece. The sides flare slightly upward, the top being an inch or two larger in diameter than the bottom. A cast-iron cover is made to fit the top closely, and in this there is a hole, with a cover, through which the charge is introduced. The bottom of the pan is either provided with slots to receive the lugs of the dies, or with a false bottom of cast iron about an inch thick, provided with openings for the lugs which hold the dies in place. The bottom of the muller is flat, and also contains slots, in which the lugs of the shoes are fixed. There are three wings or flanges placed at the top of the pan at equal distances on the circumference. When the muller is in motion the pulp is thrown to the outside of the pan, and, rising to the top, strikes against the wings, by which it is projected to the center, where it again falls to the bottom. In this way a complete circulation is obtained, the pulp not only flowing around the pan, but receiving a motion from the outside to the inside and downward, and from the inside to the outside and upward, thereby inducing an intimate mixture between the pulp and the quicksilver. When it is necessary to grind the pulp the muller is lowered by means of a screw in the top of the sleeve until the bottoms of the shoes just touch the tops of the dies. Some of the other pans in use also give a fair motion to the pulp, but for convenience in handling and quality of work the combination pan up to the present time has best fulfilled the conditions for the amalgamation of silver ores.

Where this pan is used for working roasted ores its construction is somewhat different. The bottom is of cast iron, wooden staves are used for the sides, and the cover is made of wood. The die, instead of being in several pieces, is cast whole, completely filling the bottom of the pan; the shoes and muller are one casting, and as little metal is exposed as the necessary construction of the pan will permit. This is for the purpose of reducing as much as possible the action upon the iron of the acids and acid salts frequently present to a greater or less extent in roasted pulp. Pans for working raw ore are also sometimes made with wooden sides, this under certain conditions being a cheaper construction. In the combination pan the number of the shoes is from six to twelve, and they weigh from 500 to 800 pounds, and are from 2 to 3 inches thick. The dies are similar in number, weight, and thickness. In some mills, as at Austin, when the shoes are worn off the mullers they are replaced by wooden ones, which answer the same purpose as the iron, the object being to give motion to the pulp and not to grind.

As a novelty, it may be mentioned that in a certain mill in Arizona glass shoes and dies and a wooden muller are used. The muller has copper and iron plates fastened to it (said to be for the purpose of furnishing electricity), but the whole affair is reported to be a failure, and is cited merely to show what extraordinary pans are sometimes constructed.

METHOD OF HEATING PANS.—The usual method of heating pans is by injecting live steam directly into the pulp by means of an iron pipe which descends near the outside of the pan to within 6 or 8 inches of the bottom. This method is unquestionably the best, but it has its disadvantages, of which the diluting of the pulp and continual wearing away of the pipe are the greatest. This latter is particularly the case with roasted ores. In Virginia City, where the pulp is run thin, the slight dilution caused by steam is of little importance; but in places where it is necessary to keep the pulp thick it is inconvenient. It is less objectionable in working free than in working roasted ores, for there are many kinds of the latter which, although they may be charged into the pan so thick that

a For a history of the Washoe pan and its varieties, see *Exploration of the Fortieth Parallel*, vol. iii, Mining Industry, by J. D. Hague.

they check the motion of the muller, yet, in the course of half an hour or so, the different salts having dissolved, become so thin that, with the consequent rapid circulation, they slop over, and this without the addition of steam. The effect of running roasted ore as a thin pulp is to a great extent counteracted by the fact that roasted ore, being easy of amalgamation, does not require the quicksilver to be so finely divided and so intimately mixed with it as is the case with raw ores.

Another method of heating is by means of a steam-chest under the pan, and more rarely by means of a jacket, in which exhaust steam may be used. It is only with the freest ores that it is safe to heat pans in this way, and even with such ores the pans frequently leak into the steam-chest. The reason for this is plain; the action between the salts (whether in the shape of chemicals added to the ore or formed in roasting) and the iron of the pan will be strongest at the hottest point, namely, the bottom of the pan, and should there be a defective spot in the casting a hole would soon be eaten through. In Bodie this method is in use, but the conditions are favorable, as the ores are free.

NUMBER OF REVOLUTIONS OF PANS PER MINUTE.—Pan mullers generally make from sixty to ninety revolutions per minute. It is true that the faster the muller is run the more work is done in a given time, but questions of wear and tear of machinery have thus far usually limited the speed to ninety revolutions. The pans at the California mill are geared to run ninety-five revolutions.

WEIGHT OF CHARGE.—A pan charge weighs from 1,500 to 4,000 pounds, according to the size of the pans. A 4,000-pound pan is 5 feet 6 inches in diameter and 30 inches deep. In working rich ore the charges are generally smaller than in working low-grade pulp.

TEMPERATURE.—The temperature maintained in the pan has much influence upon the amalgamation of the ore. The effect of temperature on the amalgamation of the base metals, lead and copper, is not very well understood, and it can only be determined by careful and exhaustive experiments. It seems to be a well-established fact that in the amalgamation of chloride ores which carry a considerable percentage of carbonate of lead a fine bullion may be obtained without an appreciable loss of silver by keeping the temperature of the charge considerably below the boiling point (say at 120° F.). At exactly what temperature it will be necessary to keep a given kind of pulp experiment alone can show. Time, as before mentioned, has also an influence on the results. The reason for the earlier amalgamation of the silver at a temperature of 100° F. or thereabout seems to be that the conversion of the carbonate into the chloride of lead (the form in which it is necessary for the lead to be in order to be precipitated by the iron and amalgamated by the quicksilver) takes place much less readily at a temperature of 100° F. than it does at 200° F. Were the conditions entirely uniform, the metals would be amalgamated in the order of their electro-negative properties; that is to say, first silver, then copper, and last lead; but in practice this is not the case, as unquestionably lead is amalgamated at the same time as the silver. The reason is that the pan and the pulp do not furnish all the conditions of a true galvanic cell. For instance, it is possible to conceive of a particle of carbonate of lead, not in conjunction with a particle of chloride of silver, coming into contact with the iron of the pan in the solution of salt or salt and bluestone, as the case may be. Under such circumstances the particle of carbonate would be converted into chloride and metallic lead would be precipitated and amalgamated.

It is customary to keep roasted ore at a temperature of 200° F. At the California mill the pans are heated to 150° F.

TIME OF WORKING A CHARGE.—The time occupied for working a charge is from five to eight hours. It is largely governed by the value of the ore. With any given ore, experience shows how far it is profitable to prolong the amalgamating process, which proceeds more and more slowly as the silver is taken up. In many cases, where a comparatively small portion of the silver is extracted by pan amalgamation, this is not due to the imperfection of the process, but to the fact that, under the local economic conditions, it is not profitable to continue the process until a large percentage is saved.

CONTINUOUS AMALGAMATION.—A system of handling ore in wet-crushing mills, known as the "Boss" process, has been introduced at Bodie, which dispenses with the laborious work of shoveling the pulp from the settling tanks. The ore is, as usual, crushed in a battery, but with as little water as possible, and the pulp is thence conducted to the first of a series of pans in which the mullers are revolving. From this pan it flows to the next, and so on through the series, until from the last pan it passes through a series of settlers, and finally from the last settler out of the mill. The pans are heated by steam in false bottoms, so that the pulp is not thinned. It becomes thicker as it progresses. The chemicals are added in the first pan, and the quicksilver is added at a stage when the pulp is thick enough to hold it.

GRINDING.—When ores are submitted to chloridizing-roasting, grinding is scarcely necessary, because the silver chloride passes into solution, and to some extent this is also true of raw chloride ores; but with all other ores a finer comminution than can be attained in the battery is probably essential to effectual amalgamation. As a rule, the muller is kept down about half the time necessary to work the charge. During this period chemicals, such as bluestone and salt, if added, are supposed to have produced their full effect, and at the close of it the quicksilver is added. After the addition of the quicksilver the shoes are usually raised from the dies, to avoid the flouring of the quicksilver by excessive mechanical division in the presence of more or less adherent substances. At the California mill, it is true, grinding is continued throughout the entire treatment in the pans, and it is said with satisfactory results. This practice seems to carry great authority, because the usual method was formerly pursued under the same management. Possibly, however, the charge of the pans at this mill contains less adherent material than is ordinarily the case; it is certainly less base. Grinding, after the addition of quicksilver, is especially objectionable if lead is present.

It is asserted by some millmen of experience that grinding is unnecessary with silver ores unless they contain a large proportion of gold, and there is no doubt that finer bullion is obtained when the ore is not ground; but gold is more readily amalgamated than silver, and if grinding is desirable where gold is present it is difficult to avoid the conclusion that it is still more essential when the ore is purely argentiferous. Thus 76 per cent. of the assay value of the gold contained in about 300,000 tons of Hale & Norcross ore during the eight years ending January 31, 1874, was saved by amalgamation, while only 61 per cent. of the silver in the same ore was extracted. As is the case with almost all technical operations, local economic conditions determine the extent to which grinding is profitable; but experiments to decide such points are too often omitted, or conducted under the bias of preconceived opinions.

Quantity of quicksilver added.—To effect a separation between the pulp and the amalgam it is essential that a certain volume of quicksilver should be added in the pans, for otherwise the amalgam would not gather at the bottom of the settler. Not less than 100 pounds of quicksilver to the ton of ore appears to be sufficient for this purpose; and for the ordinary pan, constructed to treat 2 tons of ore at a time, the minimum charge of quicksilver is consequently, as a rule, 200 pounds, and this only when the ore is free and of low grade. When the ore is rich, a further addition of quicksilver is made. The proportion of quicksilver to silver in the fluid amalgam should never be less than a pound to the ounce, however rich the ore may be. In working free ore the quicksilver is added from two to four hours after charging, but with roasted ores it has become the custom to add the quicksilver at once upon charging; and experience has shown that when this plan is adopted amalgamation is more complete, while the loss of quicksilver is not increased.

Chemicals used in amalgamation.—The list of chemicals used in the amalgamation of silver ores is growing smaller year by year. At present, except in a few isolated cases, the only chemicals employed in working ores raw are salt and bluestone. The reactions which take place in pan amalgamation have been but little studied; indeed, chemists have not yet reached definite conclusions as to the essential reactions of the patio process, a subject to which a number of elaborate investigations have been devoted. There seems little question that the use of bluestone and salt accelerates the reduction of argentite. The sulpho-salts of silver are also attacked, but are either only partially decomposed or so slowly reduced that it is not as cheap to treat them raw with chemicals as to submit them to a preliminary chloridizing-roasting.

There is no exact rule in use governing the quantity of bluestone and salt added, though larger quantities of chemicals are usually employed in working high-grade ores than in treating poor ones. The baser the ore the more bluestone and salt are commonly considered requisite, but the quantity of each can only be increased to that amount which will result in a yield of silver corresponding to the cost of the extra weight of bluestone and salt employed. There can be no doubt that much money has been wasted in the unnecessary and ineffective use of chemicals. The proportion which salt ought to bear to bluestone is another point, which, although of the greatest importance economically, is not well understood. Most millmen use more salt than bluestone, but others more bluestone than salt. On the Comstock, where experiments have been conducted for many years, the proper quantity of bluestone and salt, and the proportion of the one to the other to be used with any grade of the ore of that district, has been ascertained approximately; but Comstock rules would not apply in general to other ores, though doubtless there are many of substantially the same character. At the California mill, with $30 ore, 2 pounds of sulphate of copper and 6 pounds of salt are used per charge of 2 tons; with $80 ore, from 10 to 12 pounds of bluestone and 20 pounds of salt. The chemicals are added on charging. Modifications in the Comstock practice depend on the baseness of the amalgam, the tailings assays, and the appearance of the quicksilver.

As a curiosity the following list of chemicals which are used in a mill in Arizona may be mentioned: One and a half hours after charging, 20 pounds of salt, 3 ounces of cyanide of potassium, 2 ounces of sulphuric acid, 1 ounce of nitric acid, 1 ounce of hydrochloric acid, and ¼ pound of carbonate of lime. The superintendent does not profess to know what reactions are produced. In some of the mills at Bodie hydrochloric acid and carbonate of soda are used. This is equivalent to an addition of salt. In districts such as White Pine, where silver occurs almost entirely in the form of chloride, little or no bluestone is required.

When grease has got into the pans caustic potash, soda, ashes, or cyanide of potassium is added. The latter is also used upon the quicksilver in the tubs to keep it clean. Caustic lime is sometimes used with roasted ores containing a great deal of cupric chloride. It forms chloride of lime, reducing the cupric to cuprous chloride.

Sodium amalgam is found to be useful in amalgamating ores which contain the binoxide of manganese, as it prevents the flouring of the quicksilver which that mineral occasions.

Two noteworthy cases of a waste of chemicals are reported, in one instance the use of copperas in pans with roasted ore, and in another the use of bluestone with roasted ore, which already contained dichloride of copper.

Chemicals are added on charging, with the exception of caustic potash, soda, etc., which should be added with the quicksilver, which they are designed to keep bright.

At Dayton, where large amounts of the Comstock tailings are saved and worked, it is customary to add sulphate of copper in acid solution in large quantities, sometimes as much as the equivalent of 30 pounds of bluestone, when it is found that the loss of quicksilver exceeds the usual quantity. The Comstock tailings contain a very large proportion of clay, and the loss of quicksilver is probably in great part a mechanical one. The clay is said to "ball up" and carry off globules of quicksilver. The acid sulphate probably acts on the clay very much as alum does, promoting its precipitation.

USE OF IRON IN THE PANS.—Iron is one of the important factors in the amalgamation of silver ore. It is used in various forms; in the first place as cast iron in the material of the pans. As wrought iron, in the form of rings, it is sometimes put in the pans, and it is occasionally added to the charge in the form of shavings, filings, etc.

In working free ore much less iron is required than in working roasted ore, and the ordinary wear of the battery and the pan is generally sufficient. One pound of iron will precipitate 3.85 pounds of metallic silver from the chloride, but in practice more than that amount of iron is consumed where there is copper either naturally present in the pulp or added in the form of bluestone. Carbonate of lead is also believed to increase the consumption of iron.

With roasted ore much more iron is required; and where there is a great deal of lead and copper present sometimes as much as 20 pounds of iron to the ton is consumed, not including the iron needed to precipitate the silver. The best form in which to use iron in the pans is as wrought-iron filings sifted as fine as possible, but such filings are expensive and difficult to obtain. The turnings from a lathe are the best substitute. These wrought-iron fragments protect the cast iron of the pan to a great extent from the corrosive action of the different salts in the roasted pulp, as they are more readily dissolved, thus relieving the pan itself from a large amount of loss.

At the Manhattan mill, in Austin, 10 pounds of iron turnings are used to the charge, and they are almost completely consumed. Four pounds of this iron would be sufficient to precipitate the silver; the other portion, as well as the iron from the pans, is probably employed in decomposing lead and copper salts.

LOSS OF QUICKSILVER.—The usual loss of quicksilver runs from half a pound to 3 pounds to the ton of ore treated. A reasonable limit of loss in present practice is 1½ pounds per ton. The waste is in part owing to the difficulty of completely separating the quicksilver from the sand in the settler, and this is particularly the case with ores which contain carbonate of lead or other heavy minerals. To counteract this mechanical difficulty agitators are sometimes used; riffles and blankets in the sluice-boxes from the settlers are also occasionally employed. In ordinary cases sufficient separation can be effected in the settler by the proper arrangement of the shoes on the muller and the regulation of the speed of rotation, but a certain amount of quicksilver is no doubt always carried off in minute globules.

A second cause of the loss of quicksilver is the formation of lead and copper amalgams in the treatment of ores containing these metals or in working silver ores with bluestone. These alloys are pasty substances, which are rapidly reduced to the finest powder by grinding; in other words, the quicksilver is floured, and the separation in the settler is very imperfect. The most radical cure for this condition of the quicksilver is the addition of sodium amalgam in the pans before the ore is drawn off. Sometimes the loss is not sufficient to warrant the use of this expensive alloy, and in such cases it is better to gather the quicksilver and amalgam together, as well as may be, by prolonged treatment in the settler.

In the patio process of amalgamation a very large amount of quicksilver is converted into calomel and lost. In the pan process it is highly probable that one or both chlorides of mercury form to some extent, but these compounds are, for the most part, reduced by the iron. Were this not the case, the loss of quicksilver would be far greater than that actually sustained.

The loss of quicksilver is slightly greater in cold than in warm weather, especially in gold mills. As it is a well-established fact that the process of amalgamation is facilitated by a high temperature it is not strange that cold should interfere with or retard it.

LOSS OF IRON IN THE PANS.—It is a very difficult matter to estimate the amount of iron consumed in the pans in working a given kind of ore. The aggregate amount of iron worn from shoes and dies, shavings, etc., can be easily determined, but it is difficult to estimate the loss from the walls of the pan itself and from the muller. Where free ore is ground the iron worn from shoes and dies is from 6 to 10 pounds to the ton. This iron is more than sufficient to fill the chemical requirements of amalgamation, and it is therefore probable that in such cases the iron of the pan and that of the muller are very little acted upon. With roasted ore, however, this is different, and, as it is not usual to grind it, a good part of the corrosive action comes upon the pan and the muller unless iron is added to the pulp in some form or other. This loss of iron can be estimated at from one-fifth of a pound where the ore is free, poor, and not ground, up to 25 pounds to the ton where the ore is base, rich, and roasted. At the Martin White mill, at Ward, Nevada, the loss used to be 20 pounds. When free ores are ground in the pan a considerable quantity of finely divided metallic iron separates from the amalgam in the clean-up pan.

EXPLOSIVE GAS IN PANS.—Explosive gas in the pans has only been reported as met with on the Comstock, and under the following circumstances: When the mill is shut down, and the pulp is allowed to remain standing in the covered pans, an explosive gas of some kind is formed, which ignites when a lighted candle is brought near the charging hole. Before this fact was well known several accidents happened which might have ended seriously.

A man standing on the pan-cover of a pan in the California mill brought his candle near the charging-hole to see if the muller was moving, as he had just put on the tightener; the gas becoming ignited, he was blown, together with the cover, into a settler. That explosive gas has been noticed only at Washoe may be due to the fact that the pan-covers fit much more closely there than is usual elsewhere, thus obstructing diffusion. It is difficult to understand how any explosive gas can form in the pans of the California mill. Neither free acid nor sodium amalgam is there employed as a chemical, and the liberation of free hydrogen or hydrogen sulphide is therefore improbable. Mr. H. C. Hahn (*a*) asserts that the amalgamation process is accompanied by the evolution of hydrocarbons, but this is difficult to comprehend in the absence of free acids. In short, the matter requires further investigation.

CLEAN-UP OF MILL AND PANS.—In most mills which run regularly there is a general clean-up at the end of each month. A very considerable quantity of hard amalgam adheres to the shoes and dies and fills the interstices of pans and settlers, which is removed when new shoes and dies are put in place. In custom mills a clean-up is sometimes made when each lot of ore has been worked off, if the lots are of considerable value.

SETTLERS.—The usual size for settlers is a diameter of 8 feet and a depth of 3 feet. Occasionally they are found 9 feet in diameter, and 10 foot settlers are reported in two mills. For ordinary work a diameter of 8 feet is sufficient. At the Manhattan mill, where a settler is used with each pan, the settlers are 6 feet in diameter.

There are two different kinds of mullers used in settlers. In the mills on the Comstock and some others the "spider" muller is common. This consists of four arms projecting horizontally from the center of the muller, and upon each one of these arms a shoe, either of wood or iron, is fastened, which serves to plow up and keep in motion the sand at the bottom of the settler. The other form of muller ends downward in a round disc of iron, which fills the bottom of the settler to within 3 inches or so of its circumference, upon the under side of which long wooden shoes, 3 inches thick, radiating to the outside, are fastened. A settler thus arranged is the best for base ores, as it permits the rubbing and assimilation of the floured particles of amalgam in the bottom of the settler. The muller usually revolves from twelve to fifteen times per minute, although it is sometimes geared to make as many as eighteen revolutions; this, however, is only where very heavy ore is worked, or where the ore has been crushed through a very coarse screen. Such a speed is always attended with more than the usual loss of quicksilver; the greater the diameter of the settler the less ought to be the number of revolutions of its muller. It is customary to keep a charge in the settler until it is nearly time to discharge a pan into it; that is to say, from two to four hours. It is a much easier matter to settle chloridized pulp than that which has not been roasted, for in the process of roasting the particles of quartz, etc., become porous and friable. They are consequently ground finer, and can be retained in suspension in the water with much less motion of the muller.

It requires from fifteen minutes to half an hour to discharge a settler through the series of holes provided for that purpose. The top plug is of course the first to be drawn, in order that the quicksilver may have as long as possible to settle and to avoid the production of rapid currents, which might carry off amalgam. A more rapid discharge would defeat these objects.

STRAINING SACKS.—Canvas is the material usually employed for straining sacks, though in some of the mills at Bodie and other places canton flannel is used for that purpose.

CLEAN-UP PANS.—In many mills it is customary to pass the amalgam from the strainers through a clean-up pan, as in addition to sulphurets iron from the shoes and the dies of the pan is mechanically mixed with it, and it is found more economical to extract these substances before the amalgam is retorted. This process could be introduced to advantage in most mills where it is not now in vogue. A common size for clean-up pans is 5 feet diameter and a depth of 16 inches. They are usually of the Knox pattern.

HOT STRAINING.—As before stated, the base metals which enter into silver amalgam are chiefly lead and copper. For many years attempts have been made to discover a process by which these metals could be eliminated from the amalgam before retorting. Where there are copper and lead minerals in the ore it seems impossible to prevent the presence of the metals in the amalgam. A process of hot straining, now almost entirely abandoned, was for some time in use for the separation of lead amalgam. By it two classes of bullion were obtained, one consisting of tolerably pure silver, and the other of lead, with perhaps 10 per cent. of silver. The copper, however, was retained in the silver amalgam, and partly from mechanical and partly from chemical causes the loss of quicksilver was considerable. But the greatest objection to the process was the ruinous effect that it had upon the health of the men. Quicksilver, perceptibly volatile at a temperature of 40° F., becomes much more so at 212° F., and to keep large quantities exposed to that temperature, where men can inhale its vapor, is extremely objectionable. A process by which lead and copper can be eliminated from the amalgam is much needed. None is known to be in use at present which satisfies the requirements.

In very cold weather a gentle heat is sometimes applied to the amalgam before straining. At the California mill it is customary to keep the straining water at a temperature of 100° F. This facilitates straining, and no evil effects on the health of the men have been noticed.

HYDRAULIC STRAINERS.—These at present are very infrequently used, but they effect a great saving of fuel in retorting. Ordinarily dry silver amalgam from the strainers, where it is free from base metals, has about seven

a Berg- und hüttenmännische Zeitung, 1870, p. 65.

parts of quicksilver to one part of bullion. When this amalgam is strained under pressure the quicksilver can easily be reduced to three and one-half parts. A pressure of from 100 to 125 pounds to the square inch is used for these strainers.

RETORTING.—The size of the ordinary retort in use is 12 inches in diameter and 4 feet in length inside. It will hold from 600 to 1,000 pounds, according to the quality of amalgam and whether the vent is in the center or at the upper edge of the end. It is usual now to cast retorts with the vent at the upper edge of the end, and if two vents are made opposite each other in the end the retort can be turned over. This kind of retort is fast superseding the old style, where the vent was in the center of the end. Retorts up to 20 inches in diameter are reported. The retorts at the California mill are 5 feet 6 inches long and 14 inches in diameter; the charge is 1,800 pounds. Retorts are sometimes turned either a half or a quarter revolution after a "belly" has formed on the bottom, but generally they do not last long after turning, and the practice is open to strong objections. The retorts are circular in section. Amalgam is not often charged in cups, but is frequently divided by discs of iron, in order to save the trouble of breaking up the bullion for the melting pot. The retorting of a charge occupies from five to nine hours, according to its size and character. From one-sixth to one-half a cord of wood is consumed per 1,000 pounds of amalgam retorted.

The charge is heated slowly to the boiling point of quicksilver and maintained at that temperature until most of the quicksilver has been expelled, when it is gradually heated to a light cherry red for an hour or so, to expel the remainder. A mixture of salt and ashes is used to swab out the retort and for luting on the head. When the retorting is properly done the loss of quicksilver by distillation is merely nominal. At the California mill about one-half a pound of quicksilver is lost per 1,000 pounds of amalgam retorted. Where the vent is at the upper edge, or the amalgam is not base, or the pipe is kept open by a rod, the vent does not choke. With dirty or base amalgam there is always danger that the vent may choke if it is in the center of the end of the retort.

An ordinary retort will stand from 100 to 300 charges if properly handled, but in practice the life of retorts is usually much shorter. Steel retorts have been tried, but it was found that they warped too much. The fact that steel burns out sooner than cast iron would be a sufficient objection to its use.

In some small gold mills, where the amount of bullion produced is not large, simple bulb retorts of cast iron are used in place of the cylindrical ones. The bulb retorts require no brick fitting, and can be used over an assay furnace or a blacksmith's forge, and are convenient for operations on a small scale.

RETORTED BULLION.—In addition to the gold and silver contents, the retorted bullion of silver mills usually contains lead, copper, and a slight amount of iron.

In retorting amalgam which contains a large quantity of copper a porous sponge, consisting principally of copper, but containing some silver and lead, is found on the top of the retorted bullion. Wherever the air reached this porous mass the metallic copper is converted into oxide. This substance is the bugbear of the melter, for the ordinary temperature of the furnace will not melt and reduce it, but converts it into a plastic mass, which no amount of flux seems to affect. This alloy contains from 20 to 30 per cent. of silver, and should never be thrown away. A high temperature will reduce and melt it, or it can be roasted, ground, and the copper extracted with sulphuric acid; the residue, consisting of silver and lead, can then be melted into bullion.

Certain alloys of copper, lead, and silver seem to retain a small percentage of quicksilver even after the bullion is melted; how much is not known, but it is sufficient to produce a strong reaction for quicksilver when tested, and its frequency renders the Gay-Lussac wet assay an untrustworthy method for such bullion.

MELTING FURNACES.—Melting furnaces are all very similar in construction. They are usually about 16 inches square and 18 inches deep, the top sloping toward the front. The materials are common brick, fire-brick, and cast iron for doors, etc., with wrought-iron ties.

CRUCIBLES.—A No. 50 or a No. 60 Dixon black-lead crucible is the kind mostly used for melting silver bullion. These crucibles last for from fifteen to twenty-five melts.

FLUXES.—Either slag from assaying, or borax, bicarbonate of soda, and saltpeter, are the fluxes usually employed in melting. Sometimes sulphur is used where there is much iron in the bullion.

AGITATORS.—Agitators are not very frequently used. Where they are employed it is more for the sake of concentrating the settler sands, sweepings, etc., than for catching amalgam. They are usually 7 feet in diameter and about 3 feet deep. The assay value of the sands recovered in them is usually higher than that of the tailings and less than that of the ore. Agitators are generally geared to make twelve revolutions per minute.

TAILINGS.—The tailings of free silver ores are almost always saved, those from roasted ore scarcely ever.

Tailings from the battery of a gold mill are handled in different ways, but the following is the usual method: After passing through the screens, the tailings, which contain considerable gold amalgam when battery amalgamation is practiced, flow over an amalgamated copper plate or silver-plated copper plate, on which most of the quicksilver alloy is caught. This plate is as wide as the battery screen and usually from 3 to 5 feet long, and is called the apron. From the apron the tailings flow into sluices in which are also amalgamated silver or copper plates, alternating with riffles or boxes, for catching quicksilver, and finally reach sluices in the bottoms of which blankets are laid. On these blankets the sulphurets, which are usually quite rich in gold, are caught. After leaving the blanket sluices the tailings sometimes run into a concentrator, in which further valuable contents are recovered from them. The Hendy concentrator, the one most employed in California, is usually placed after the plate sluices and before the blankets. Buddles are also often used, as well as the Frue vanner. The blankets used are generally 20 inches wide, and are manufactured for the purpose by woolen-mills in San Francisco. The arrangement of amalgamating and concentrating apparatus in gold mills varies greatly, however; at the Idaho mill, for example, there is no battery amalgamation, and the pulp from the battery is immediately concentrated on blankets. The concentrations are treated in pans, and the tailings from the blanket sluices pass over riffles and other amalgamating apparatus to buddles.

GRADE OF SLUICES.—The grade given to blanket sluices is usually from three-quarters to 1 inch per foot, depending on the specific gravity and the fineness of the ore and the proportion of sulphurets contained in it. Sometimes the grade is increased to as much as 3 inches to the foot. The apron used in gold mills has always more grade than the sluices, and in many mills there is an arrangement by which the end of the apron can be raised or lowered as different peculiarities of the ore may demand.

PERCENTAGE OF SULPHURETS.—The percentage of sulphurets in California gold ores is very varied. The average is about 2 per cent. These sulphurets, when concentrated, are worked in different ways. Where rich, the best method is to roast, chloridize with chlorine gas, and leach. One or two per cent. of salt added before roasting facilitates the formation of the terchloride of gold and reduces the quantity of chlorine gas required. That terchloride of gold can be made by roasting sulphurets containing gold with salt is a well-established fact, but although at one stage of the process most of the gold is in the form of terchloride, upon cooling this compound is partially decomposed, a portion only of the gold remaining combined with chlorine. In making the chloridation assay of roasted silver ores containing gold it has been noticed that a portion of the gold is always leached through the filter, showing the presence of terchloride. The cost of such chloridation is about $10 per ton. Where the sulphurets are poorer they are worked in pans or revolving barrels.

WEATHERING OF CONCENTRATIONS.—Concentrations are allowed to weather from three months to a year, salt sometimes being added to assist the decomposition when it is the custom to work them in pans, etc.

PERCENTAGE RECOVERED FROM CONCENTRATIONS.—The Plattner chloridation process and its modifications save 90 per cent. and upward of the gold contained in sulphurets, but the percentage obtained by the other methods is much less. It depends on the character of concentration, length of exposure to weathering, and the process by which the sulphurets are worked. The yield by other methods of chloridation often falls below 50 per cent., though usually it is higher.

Blanket sluices are of little use in silver mills except where the ore, although mainly free, contains rebellious sulphurets, as is the case on the Comstock and at some places in Idaho. At the California mill about 10 per cent. of concentrations are obtained per day. These are allowed to weather from one to six months, salt being added, and are worked in pans.

TREATMENT OF TAILINGS FROM SILVER ORES.—With that class of silver ores which is treated raw it is rarely possible to extract such a percentage as to render the tailings worthless, and therefore these are commonly saved and allowed to weather, salt sometimes being added to assist the decomposition. As a rule, however, there is enough salt remaining in the tailings after working the ore to produce in the course of a year or two the required effect. It must be remembered that, no matter how fine a particle of tailings may be, there can still be a particle of silver mineral in the center of it so completely enveloped by earthy material that chlorine cannot act upon it and quicksilver cannot touch it. When tailings are treated, only that portion of silver is amalgamated which is exposed to contact with the quicksilver. As the outsides of these tailings particles have already been once subjected to the influence of chlorides and quicksilver, the amount of silver which can be recovered by a second amalgamation, of course taking it for granted that the ore has been properly treated in the first instance, except where long-continued grinding is practiced or where the character of the gangue has been changed by weathering, is exceedingly small. The percentage extracted from tailings does not often exceed 50 per cent. of the assay value, and generally falls far below this point. With the tailings from roasted ore, which are of lower grade than those from raw working, it is still more difficult to obtain the silver.

SLIMES.—Slimes sometimes assay more than the ore itself, sometimes less. The percentage of slimes escaping from wet-crushing mills varies from 1 to 15 per cent., according to the character of the ore. Ores containing much iron oxide generally produce the most slime. The slimes are usually first caught in large tanks, and then in shallow ponds. Eventually very little escapes if proper settling room is provided.

SALIVATION.—Only a few cases of sickness through salivation have been reported. The bursting or imperfect luting of retorts and the cleaning up of pans while still hot are frequent causes.

SAMPLES OF ORE.—Samples of ore are sometimes taken from each car-load at the mine. This is the case at the California, where a handful is taken from each car. When the ore of a mine is treated by a custom mill ore samples are often taken by the mine owners as a check upon the working of the mill.

BATTERY SAMPLES.—Battery samples are always taken at the mill, and in wet-crushing generally in the following manner: Every half hour or hour a rectangular iron box, or dipper, is passed along under the lips of the mortar where the pulp falls into the sluices, and a portion of the slimes and sand is taken, care being observed to prevent concentration by the overflowing of the box. This sample is put in a bucket with others, and at the end of twenty-four hours the clear water is poured off and the collective samples for the day are dried and prepared for assay.

In dry-crushing mills it is usual to take the sample by passing a box along the lower part of the screen every hour or so. Sometimes the sample is taken from the conveyor or from the pulp in the bins.

TAILINGS SAMPLES.—Tailings samples are taken from one or all of the discharge plugs of the settler or are dipped from the top before each plug is drawn. A very good way is to dip a small gas-pipe into the settler close to the top, one end of the pipe being closed with the finger, and withdraw a portion of the pulp.

BULLION SAMPLES.—Silver bullion is sampled by taking a dip from the melted mass after stirring, or a little is poured into water while the crucible is still nearly full, and a little more when it is nearly empty. Either method is good. Gold bullion is sampled by chipping two diagonally opposite corners of the bar after it is cast.

ORE ASSAYS.—Ore assays are generally made in crucibles, though with very base ores scorifiers are sometimes used. They are seldom corrected for loss in assaying.

ROASTING FURNACES.

SYSTEMS OF FURNACES.—There are many different kinds of furnace in use for chloridizing ore. Among the more common types may be mentioned the Howell, Brückner, White, Stetefeldt, O'Harra, and ordinary reverberatory. The Howell, Brückner, and White furnaces are revolving cylinders. The axes of the Howell and the White furnaces are at an angle of a few degrees to the horizontal, the ore being fed at the higher end and discharged at the lower. In the White furnace the ore and the flame from the fire-box enter the furnace together at the higher end. In the Howell the ore is fed in at the higher end and the flame enters at the lower. This constitutes the material distinction between these two furnaces, and the difference is in favor of the Howell. The Brückner furnace is a cylinder of boiler iron, with ends usually made of the same material, though sometimes they are of cast iron. In the center of each one of these ends, or heads, there is a hollow trunnion, through which the flame enters or passes out of the cylinder. The axis is horizontal, but, like the Howell and the White, the furnace revolves on a series of wheels or rollers. The motion is conveyed to all these furnaces by a system of gearing, a belt of cogs passing around the outside of the cylinders, which gears with a small spur-wheel on a counter-shaft below the furnaces. By an improvement in the gearing of the Brückner the speed may be varied from zero to six revolutions per minute, according to the requirements of the roasting process. In the Brückner furnace the charge of about 3,000 pounds is introduced through a door in the cylinder, placed half way between the two ends, which is closed by means of a clamp. These furnaces are also constructed with two doors opposite each other. A characteristic feature of the Brückner cylinder is the diaphragm, which consists of iron plates about one inch thick fastened to tubes passing through the cylinder at right angles to its axis. The plane in which these tubes are placed makes an angle of about 30° with the axis of the cylinder, so that the line where the plates and cylinder join is an ellipse. This diaphragm extends to within 18 inches of each end, and when the furnace revolves it induces a circulation of the pulp from one end of the cylinder to the other. Although this diaphragm gives an almost perfect motion to the ore, it has several disadvantages, among which are the following: The cast iron of which the plates are composed is corroded by the action of sulphur and chlorine, and the lively motion conveyed to the pulp causes a great portion of it, sometimes as high as 30 per cent., to be carried into the dust chambers. A great part of this dust, too, leaves the furnace before it has had time to be roasted, and thus escapes decomposition altogether. In consequence of these disadvantages the diaphragm is now usually omitted. Although the furnace, as at present constructed, does not chloridize to quite as high a percentage as it would if provided with a diaphragm, it has been proved to be far better suited to the economical working of ores. When the charge has been properly roasted, it is discharged through the doors by opening them and revolving the cylinder. As these furnaces are at present built, the speed can be increased when it is necessary to discharge by introducing a gear-wheel of larger diameter than that which ordinarily gives motion to the furnace.

The Stetefeldt is a high-shaft furnace, into which the flame enters from two fireplaces on the sides at the bottom, the ore, mixed with salt, being fed into it at the top. The fumes and dust pass out through a flue at the top, descend, and, after passing through the flame of a smaller auxiliary fire, are conducted to the dust chambers, whence the fumes escape through the stack. The ore occupies only a second or two in passing through the flame, but it is retained in the heat at the bottom of the furnace three-quarters of an hour before drawing the charge. The auxiliary fire has also been applied in connection with the revolving cylinders, and with good results.

The reverberatory furnace, as applied to the chloridation of silver ores, has sometimes one hearth, and sometimes two or three hearths. Where several hearths are used, the ore is first put into the one nearest the flue, and after roasting a time is raked to the next, and finally to the last, near the fireplace. Thus, in a furnace with three hearths, there are always three charges in different stages of roasting.

The O'Harra furnace is a brick double-deck reverberatory furnace, 9 feet high, 8 feet wide, and 60 feet long, in which the ore is stirred by scrapers or hoes attached to an endless chain moved by the mill machinery. The two hearths are level, one above the other, and 47 inches apart (arches 10-inch spring), 60 feet long, and only closed at the ends by hinged sheet-iron doors. These doors close the furnace so that the draught is not injuriously affected, and also enable the scrapers to pass under them readily. There are four fireplaces, two on each side, so arranged that all may be used to heat the lower hearth, or two of them may heat the upper hearth. The draught is through a stack connecting with flues from the arches over the hearths. The endless chain passes over both hearths in opposite directions and over a pulley at each end of the furnace. To one of these pulleys power is transmitted by spur or friction gearing. To the chain at equal distances apart are attached two scrapers. Each of these carries 14 hoes—plates of wrought-iron about the size of a hoe—which are dragged along the hearths through the ore. These hoes are set at a slight angle with the chain, thus moving the ore somewhat forward as well as turning it over. The hoes of one scraper are set in a reversed position from those of the other; thus the ore is not pushed to the center or sides of the hearth, and the furrow made by one scraper is filled by the next. As the links of the chain and the scrapers are cooled by passing from one hearth to the other through the outer air, they are but little attacked and last for months.

It is not absolutely necessary to line any of these furnaces with fire-brick, but it is always best to use fire-proof material in the neighborhood of the flame. As a rule, fire-brick are much harder than ordinary brick, and for that reason are better suited to stand the wear and tear of a revolving furnace. There is not much difficulty in keeping the brick in place in the Howell and the White furnaces, their diameter being small, but with the Brückner it is otherwise. The large diameter of this furnace causes the arch to be insecure unless the brick are good and are put in very tightly. A very good method of increasing the security of the lining is to divide the circular arch into four parts by running flanges of boiler plate the length of the cylinder, thus making four independent arches. For this furnace it is cheaper in the long run to use the best fire-brick and as little clay as possible. Salt and ashes are said to make a very good mortar for laying the bricks in place, and they certainly form a mortar which does not contract on being exposed to heat. The heads of the Brückner furnace are usually lined with the best fire-clay, tamped around and over projections of iron, or large brick are molded in the form of segments of a circle and fastened to the heads by clamps and bolts.

STATISTICS.—Of the ninety-five roasting furnaces reported, eighty-nine were in use during the census year. More than one-half, fifty-seven, are of the cylinder type, while twenty-nine are reverberatories and nine are shaft furnaces.

TABLE CXIII.—ROASTING FURNACES IN AMALGAMATING MILLS.

ARIZONA.

County and district.	Mill or works.	Number of furnaces.	Number in use.	Capacity in twenty-four hours.	Character.
MOHAVE.				*Tons.*	
Hualapai	Mineral Park	1	1	6	Brückner cylinder.
PIMA.					
Arivaca	Arivaca	1	1	3½	Stewart reverberatory.
PINAL.					
Globe	Isabella	1	1	15	Howell cylinder.
Pioneer	Silver King	1	1	6	Brückner cylinder.
YAVAPAI.					
Peck	Peck	1	1	15 to 18	White cylinder.
Pine Grove	Bradshaw Basin	1	1	8	Do.
Tiger	Tiger	1	1	15 to 18	Do.
	Tip-top	1	1	15	Howell-White cylinder.

CALIFORNIA.

County and district.	Mill or works.	Number of furnaces.	Number in use.	Capacity in twenty-four hours.	Character.
AMADOR.					
Sutter Creek	Amador reduction works	1	1	4½	Reverberatory.
Do	Phœnix reduction works	2	2	8	Do.
Do	Sutter Creek	1	1	8	Do.
NEVADA.					
Nevada City	Pioneer reduction wrorks	1	1	4	Crosby cylinder with reverberating finishing hearth.
Do	Providence	1	1	5	Reverberatory.
PLUMAS.					
Quartz township	Plumas-Eureka	1	1	1½	Brückner cylinder.
SHASTA.					
Pittsburgh	Copper City	2	2	40	O'Harra automatic stirring hearth.
SISKIYOU.					
Sawyer's Bar	Klamath	1	1	2	Reverberatory.
South Fork Salmon	Black Bear	1	1	6	Do.

TABLE CXIII.—ROASTING FURNACES IN AMALGAMATING MILLS—Continued.

COLORADO.

County and district.	Mill or works.	Number of furnaces.	Number in use.	Capacity in twenty-four hours.	Character.
ARAPAHOE.				*Tons.*	
Argo	Boston and Colorado reduction works	8	8	3½ to 3½ each	Reverberatory (roasting matte).
BOULDER.					
Grand Island	Caribou	4	4	5½ each	Brückner cylinder.
CLEAR CREEK.					
Griffith	Clear Creek	6	6	3 each	Do.
Do	Farwell reduction works	5	5	2½ each	Do.
SAN JUAN.					
Eureka	Melville	1	1	7	Reverberatory.

DAKOTA.

County and district.	Mill or works.	Number of furnaces.	Number in use.	Capacity in twenty-four hours.	Character.
LAWRENCE.					
Bear Butte	Florence	2	2	6 each	Brückner cylinder.

IDAHO.

County and district.	Mill or works.	Number of furnaces.	Number in use.	Capacity in twenty-four hours.	Character.
ALTURAS.					
Middle Boisé	Buffalo	3	3	10	Hunt, Douglas & Stewart reverberatory.
Do	Davis tailings mill	1	1	1½	Reverberatory.
BOISÉ.					
Banner	Elmira	2	2	12½ each	Do.
LEMHI.					
Yankee Fork	General Custer	2	......	28	Hyde-Brückner cylinders.
OWYHEE.					
Wagontown	Tremont	1	......	10	Reverberatory.

MAINE.

County and district.	Mill or works.	Number of furnaces.	Number in use.	Capacity in twenty-four hours.	Character.
HANCOCK.					
Sullivan	Sullivan	1	1	10 to 15	Howell cylinder.

MONTANA.

County and district.	Mill or works.	Number of furnaces.	Number in use.	Capacity in twenty-four hours.	Character.
DEER LODGE.					
Flint Creek	Algonquin	1	1	20 to 30	Howell cylinder.
Summit Valley	Alice	2	2	12 each	Do.

NEVADA.

County and district.	Mill or works.	Number of furnaces.	Number in use.	Capacity in twenty-four hours.	Character.
ELKO.					
Columbia	Columbia Con	1	1	13 to 18	Howell-White cylinder.
Tuscarora	Grand Prize	2	2	15 each	Do.
Do	Independence-Navajo	1	1	20	Do.
ESMERALDA.					
Columbus	Northern Belle	2	2	60 each	Stetefeldt shaft.
Oneota	Indian Queen	1	1	10	Howell cylinder.
EUREKA.					
Cortez	Wenban	1	1	5	Brückner cylinder.
HUMBOLDT.					
Mount Rose	Paradise Valley	1	1	15	Howell-White cylinder.
Winnemucca	Humboldt reduction works	2	1	10	Brückner cylinder, 10 tons capacity, in use; also Aiken shaft, not in use.
LANDER.					
Lewis	Starr & Grove	3	3	5 each	Brückner cylinder.
Reese River	Manhattan	1	1	22	Stetefeldt shaft.
LINCOLN.					
Eldorado	Southwestern	1	1	20	Howell cylinder.
NYE.					
Morey	Morey	1	1	20	White cylinder.
Tybo	Tybo	1	1	30	Do.
Union	Alexander	2	2	20 to 25 each	Howell-White cylinder.
WHITE PINE.					
Cherry Creek	Star	2	2	11 each	Howell cylinder.
Ward	Martin White	4	4	12 to 15 each	Do.

TABLE CXIII.—ROASTING FURNACES IN AMALGAMATING MILLS—Continued.

NORTH CAROLINA.

County and district.	Mill or works.	Number of furnaces.	Number in use.	Capacity in twenty-four hours.	Character.
CABARRUS.					
	American	1	1	6	Meares' revolving hearth.
MECKLENBURG.					
	New York and North Carolina reduction works.	1	1	20	Robinson.
ROWAN.					
	North Carolina	1	1	6	Reverberatory.
OREGON.					
GRANT.					
Granite	Monumental	1	1	20	Howell-White cylinder.
UTAH.					
JUAB.					
Tintic	Crismon-Mammoth	1	1	35	Howell-White cylinder.
Do	Ely	1	1	20 to 25	Aiken shaft.
SALT LAKE.					
West Mountain	New York and Utah reduction works	1	......	a 10	Brewster's rotary hearth.
SUMMIT.					
Uintah	Ontario	3	2	72	Two Stetefeldt shaft furnaces, 25 tons capacity each, in use; 1 Howell cylinder, capacity 22 tons, not in use.
UTAH.					
Tintic	Tintic	1	1	24	Stetefeldt shaft.

a Estimated.

DIMENSIONS OF FURNACES.—The dimensions of the Howell furnace are as follows: Length, from 16 to 27 feet; inside diameter, from 24 to 38 inches. The furnace is made in several sections, which are bolted together. Formerly the two sections near the fireplace were made larger than the others, and were only lined with brick. It has been found good policy, however, to make all the sections of the same diameter and line the cylinder with brick throughout.

The White furnaces are usually shorter than the Howell and of greater diameter, and are lined throughout their whole length. The bricks forming the linings of these furnaces are usually set on edge.

The Brückner cylinder is usually 12 feet long and 5 feet in diameter inside, the bricks being laid flat. Some furnaces, however, have been constructed for the Ætna company at Galena, Nevada, 16 feet long and 6 feet 6 inches in diameter. These furnaces are provided with four discharge holes.

The Stetefeldt furnace is built of different sizes. The shaft proper is usually between 30 and 40 feet high.

The usual size for a reverberatory furnace hearth is 12 feet square, the arch being made low, in order to keep the flame as near the pulp as possible, but they are sometimes much longer.

In the Howell furnace the discharge end is from 7 to 10 inches lower than the feed end. The cylinder is geared to make from three to six revolutions per minute, according to the diameter of the furnace, the angle of incline, and the character of the process. The less the diameter the greater should be the speed. These furnaces are now arranged so that the speed can be increased or decreased at will, so as to vary the length of time that the ore requires to pass through. By decreasing the speed the ore is retained longer in the furnace, thus facilitating the working of baser ores. The Brückner cylinder ought to be geared so as to make one revolution in two and a half or three minutes if 5 feet in diameter, or one revolution in four minutes if of the larger size, 6 feet 6 inches in diameter.

REPAIRS.—Neither of the cylinder furnaces require frequent repairs if the lining has been properly secured. The Stetefeldt shaft furnace, when built of good material, seldom gets out of order, and repairs may be limited to an occasional renewal of the brick in the fireplaces.

The repairs needed in a reverberatory furnace depend upon the fire material used in building it, upon the form of the arch (flat arches being more liable to fall in than round), and upon the care with which it is anchored. Generally these furnaces last for years, but slight patching here and there being required.

CAPACITY OF THE DIFFERENT FURNACES.—The Howell, the White, the Stetefeldt, and the reverberatories with several hearths or with a single very long one are continuous furnaces; the Brückner and the reverberatory with a single short hearth finish a given charge before a second is introduced. The capacity of the ordinary-sized Howell and White furnaces is from 10 to 15 tons, although over 20 tons are sometimes put through these furnaces in twenty-four hours, and at the Alexander mill, in Nevada, when working on ore containing most of the silver as chloride, a capacity of 50 tons was reached. The 12 by 5 feet Brückner cylinder will hold about 2 tons of ordinary ore, the amount put through in twenty-four hours depending on the time needed to roast, which is very different for different ores. A Brückner will ordinarily work 6 tons a day. The Stetefeldt furnace can work from 20 to 60 tons, according to its size, in a day, and the reverberatory furnaces from 1,000 to 2,000 pounds to the charge. The only O'Harra furnaces reported have a capacity of 20 tons each per twenty-four hours.

METHODS OF CONVEYING ORE TO FURNACES.—The usual way of carrying ore from the battery to the furnaces is by means of screw conveyors and elevator belts with sheet-iron cups attached. Sometimes cars are used for this purpose.

METHOD OF FEEDING ORE INTO FURNACES.—In the Howell and the White furnaces the ore, mixed with salt in the battery or after crushing, falls from the elevator into a chute, which carries it into the upper ends of the furnaces. For the Brückner the ore and salt are crushed in the battery together and conveyed to the hopper above the furnace, or the salt, having been ground fine in a mill or crushed by a separate battery, is added after the charge is partially roasted. At the Custer mill, in Idaho, the salt is pulverized separately, but is added with the charge of ore. The same methods are used with the reverberatory furnaces.

In the Stetefeldt the feed of the ore and salt is regulated by mechanical appliances, so that the supply of each entering the furnace can be adapted to the necessities of the ore.

MANIPULATIONS IN ROASTING.—In the ordinary reverberatory the work to be done consists in raking and hoeing the ore from the flue end of the furnace to the fire-bridge and back and forth, so as to expose every particle first to the oxidizing influence of the air and then to the chloridizing effect of the decomposing salt. The quality of the roasting depends almost entirely upon the care and diligence with which this manipulation is performed. After the roasting is completed the ore is raked from the furnace to the cooling floor and there sprinkled with water, either immediately or after an interval. In the Stetefeldt, every three-quarters of an hour the door at the bottom of the shaft stack is opened, and the ore which has collected at the bottom is drawn out upon the cooling floor and sprinkled.

In the Howell furnaces the ore falls from the lower end of the cylinder into an iron box set in a chamber between the fire-box and the furnaces and closed by iron doors. When the box is full the doors are opened, and it is swung by means of a crane out upon the cooling floor and dumped. It is then replaced, and the doors are closed. A simple brick oven may be substituted for this movable box.

When the charge in the Brückner is completely roasted the furnace is stopped, and, the discharge doors having been opened, it is again put in motion, and as the cylinder revolves the ore falls directly on to the cooling floor. It is almost completely discharged in ten revolutions, after which another charge is introduced.

Whichever furnace is employed the product is, or should be, the same. The color varies from a light reddish-yellow to a dark brown, its shade depending on the amount of oxide of iron. It should have a light, porous, and woolly appearance, and when horned out should show no sulphurets.

TEMPERATURE MAINTAINED.—In the Brückner and the reverberatory furnaces, after the desulphurization of the ore, which requires from two to eight hours, the temperature is raised and the chloridation period begins, which lasts from two to four hours. In the other furnaces, with one exception, a uniform temperature, about a cherry-red, is maintained at the fire-bridge, the ore being exposed to higher and higher temperatures as it passes through the furnace. In the original White furnace the ore is heated to the highest temperature at the point of entry.

TENOR OF ORE TO BE ROASTED.—Silver ores which require roasting before they can be amalgamated are of very varied composition. There are some ores which contain so little sulphur that only an incomplete chloridation is obtained unless sulphur, either in the form of brimstone, of iron pyrites, or of copperas, is added before the ore is introduced into the furnace, sulphur in some form being necessary for the decomposition of the salt and the liberation of the chlorine. There are other ores, on the contrary, which contain so much sulphur in the form of sulphides that a long oxidizing roasting is necessary before they can be prepared for chloridation. The typical roasting ore is quartz containing silver minerals and from 10 to 15 per cent. of iron pyrites, with a slight admixture of copper sulphides. Calc-spar, braunspar, and fluorspar, if present in any quantity in an ore, retard the chloridation, as they absorb a large part of the sulphuric acid. Arsenic and antimony minerals increase the loss of silver by volatilization. Zincblende requires a long oxidizing roasting to convert it into sulphate, and then a high temperature must be maintained before it will decompose the salt. Where there is a large amount of zinc in the ore the chloridation is an imperfect one. Lead and copper contaminate the amalgam and bullion. All these minerals involve the use of a large percentage of salt in roasting, but if present in only small quantities they do not perceptibly affect the chloridation.

In mixing ores the sulphur and the silver contents of the charge are kept at those percentages which have been determined in actual practice to be the most favorable under the circumstances to the chloridation of the silver. The effort made to obtain rational rules as to the most expedient relations of silver, sulphur, salt, etc., was a failure, most of the managers having determined their mixtures empirically and knowing nothing definitely of the percentage composition of their ores. The richer and baser the ore the more salt it requires.

DURATION OF THE ROASTING PROCESS.—In the Howell and the White furnaces the ore is exposed to the flame from seven to twenty minutes; in the Brückner and the reverberatory from five to twelve hours, and in the Stetefeldt it is about two seconds falling through the flame, and remains forty-five minutes at the bottom of the furnace.

Oxidizing-roasting.—When ores contain much arsenic and antimony much salt is saved by exposing them to a preliminary oxidizing-roasting. This may easily be done in reverberatory or Brückner furnaces, but in the Stetefeldt, the Howell, and the White furnaces it is not practicable. In roasting ores free from arsenic and antimony there appears to be no advantage in delaying the addition of salt.

Percentage of salt used.—When the salt is mixed with the raw ore there is not much difference in the amount required to roast a given ore in any of these furnaces, but the Stetefeldt and the Brückner are supposed to require a smaller percentage than the others. From 3 to 15 per cent. of salt, according to the character of the ore, are the usual limits, 8 per cent. being the average. Salt is usually the chief item of expense in roasting ores, and millmen frequently use more than the quality of the ore demands. They think it always best to be on the safe side, and by allowing a certain margin of excess often considerably increase the expense.

When salt is added.—Generally salt is added with the raw ore, though in working very base ores in the Brückner and the reverberatory furnaces it is customary to add it at the end of the oxidizing period.

Sulphur contents.—The percentage of sulphurets in roasting ores varies from 1 to 70 per cent. The closeness with which it is necessary to keep to a given sulphur content varies with the different furnaces. In the Stetefeldt there is but little latitude, from 3 to 5 per cent. sulphur being the limits. If this quantity will not liberate chlorine enough to chloridize the silver, the ore must be reduced in grade by the admixture of poorer qualities. In the White and the Howell there are larger margins, and in the Brückner and the reverberatory the very basest ores can be worked.

Percentage of silver chloridized.—The percentage of silver chloridized varies from 75 to 90 per cent. in these different furnaces, it seeming to depend more upon the character of the ore and the method of working than upon differences of construction and manipulation. The difference in this respect is perhaps a little in favor of the Brückner and the reverberatory, as these furnaces permit of an oxidizing-roasting before the salt is added.

Labor.—In all these furnaces, except the reverberatory, a man can roast 5 tons of ore per shift. One man can easily attend to two Brückner furnaces. In the reverberatory it is fair work for one man to roast a ton per shift.

Fuel.—Wood is the fuel used in all these furnaces except in the Stetefeldt, where some charcoal is occasionally burnt. This wood is of many different kinds, and its value as fuel is chiefly governed by its weight. The different furnaces require on the average the following weights of wood to roast a ton of ore, the weights being calculated on the basis of 2,200 pounds per cord as the average:

	Pounds.
Howell	300
White	300
Brückner	900
Reverberatory	600
Stetefeldt	200

The Stetefeldt furnace uses less fuel than the others, because it is built solidly of brick and retains the heat better; the Brückner uses the most, because it is a large cylinder, and it is necessary to reheat it each time it is discharged.

Power.—The power used to drive the mechanical furnaces is estimated at about two horse-power for the Brückner and one and a half horse-power for the White and the Howell furnaces.

Loss of weight of ore in roasting.—The loss of weight of ore in roasting ranges from 3 to 15 per cent. Very base ores, of course, lose much more weight than freer ores. Leaving the sodium sulphate out of consideration, the tendency of the roasting process is to reduce the weight of the ore. The roasted product, as a rule, contains considerable quantities of soluble metallic chlorides and sulphates.

Dust-chambers and flues.—The best arrangement of dust-chambers and flues is that of the Stetefeldt furnace. The system consists of a series of dust-chambers, sometimes as many as twelve, through which the dust and fumes pass, giving the dust an opportunity to settle, and of a long flue connecting with a stack, which is generally placed on a hill-side at some distance from the furnace.(*a*) The dust-chambers nearest the furnace are opened several times a day and the flue-dust is raked from doors at the bottom. The long flue is opened once in about six months, when the furnace is shut down. Two-thirds of the flue-dust is deposited in the first two or three chambers.

The flues and dust-chambers of the other furnaces are arranged in a somewhat similar manner, except that they are not usually so complete. The dust-chambers are often placed under the dry kiln, so that the heat from the furnace can be used to dry the ore, and the smoke-stack is often built in the mill itself, though it may better be placed as indicated in the preceding paragraph. The practice of placing the drying kilns over the dust-chambers is to be recommended only when draught is cheaper than fuel, which, however, is usually the case.

Amount of flue-dust caught.—The amount of flue-dust caught per ton of ore roasted depends on many circumstances, such as the character of the ore, its fineness, the style of furnace, and the draught. Those ores give the most flue-dust, which, before they are roasted, are light and porous and contain oxide of iron. The ores which give the least are hard quartz ores with sulphurets. Those ores which contain a large proportion of

a By thus taking advantage of the topography the expense of a high stack is avoided.

sulphurets do not, of course, need to be crushed as fine as those which contain little, the sulphurets being readily attacked by oxygen and chlorine when unprotected by a coating of minerals not affected by these gases. When there is no sulphur present it is necessary to add it as pyrite, brimstone, or, best of all, as copperas. The ore should then be fine, to facilitate the immediate and complete action of the chlorine.

If the reverberatory furnace is properly handled, it produces the least flue-dust, for the motion of the ore is less violent than in the others. With care in the reverberatory furnace the amount of flue-dust can be limited to 2 per cent. From the nature of the Stetefeldt furnace a considerable amount of flue-dust passes into the dust-chambers, but as practically all of it settles there, and as owing to the auxiliary fire it is chloridized to a higher percentage than the ore itself, this fact is of no importance. As much as 10 per cent. sometimes passes over. This is also true of the Howell furnace, except that the dust is usually not so well settled as in the Stetefeldt. At the Grand Prize, in Nevada, the flue-dust chloridized 5 per cent. higher than the ore itself, and at other mills this higher chloridation of the dust is very frequently the case. With the Brückner furnace the amount of flue-dust is usually not large, but it is not chloridized as high as the ore, and sometimes requires a second roasting.

The draught in all these furnaces can be regulated by two dampers, one between the furnace and the dust-chambers, the other in the smoke-stack.

Loss of flue-dust.—The actual loss of flue-dust is very slight with most ores where the furnace has plenty of dust-chambers and long flues, and it probably does not often exceed 1 per cent.; but where the ore is very finely divided, even under favorable circumstances this loss may reach 5 per cent. Extensive dust-chambers are usually desirable.

As regards the loss of silver and gold by volatilization, no reliable data are available. That there is such a loss cannot be questioned, but at present there is no accurate means of estimating it. Except with ores containing much arsenic, antimony, and zinc, it is probable that it never exceeds 1 per cent.

Difference in composition between flue-dust and roasted ore.—Flue-dust is different in composition from roasted ore. It is lighter, and its color is usually a shade darker. It contains ashes and charcoal, as well as more of the base-metal chlorides than the roasted ore. It often contains fine undecomposed sulphurets. The flue-dust from the Stetefeldt and the Howell furnaces is chloridized higher than the roasted ore. This is due to the action of the auxiliary fire through which the dust falls as it comes from the furnace. In some places there is an arrangement by which a small percentage of fine salt can be mixed with the dust as it passes through the flame to assist the chloridation. The flue-dust from the other furnaces is chloridized lower than the ore, and, although the auxiliary fire has been introduced in the construction of Brückner furnaces, it has not been found to have the desired effect. In the Brückner furnace the cause of this may be that the greater part of the flue-dust passes over into the chambers when the ore is first charged, and as it is not at all roasted the auxiliary fire is not capable of chloridizing it. The tail fire is more beneficial with continuous furnaces than with those which receive a charge at stated intervals.

Different kinds of furnaces as compared to each other.—The principal advantages and disadvantages of the furnaces described may be briefly stated as follows:

The Howell, of all the mechanical furnaces, is the one which is most easily handled. It is not an expensive furnace, and requires little power and few repairs. It has a continuous feed and discharge, which lightens the labor required. It is the furnace which, with the exception of the Stetefeldt, requires the least fuel to roast a ton of ore. Its flue-dust is chloridized to a higher per cent. than the roasted ore, and, with the possible exception of the Stetefeldt, it requires less manual labor to run it. Like all continuous furnaces, it requires the ore to be crushed fine and needs a large percentage of salt when the ore is light or at all base. These points, however, are not serious disadvantages. What prevents the furnace from being universally adopted is the fact that it is not available for the basest ores. The length of time during which a particle of ore is exposed to the action of the flame and the air is short, though in the latest furnaces the time has been considerably extended. The reason that this furnace is not adapted to chloridize the basest ores is that the period of oxidizing-roasting, which begins at the flue end of the furnace, is altogether too short to permit of any quantity of zincblende, galena, or other refractory minerals being oxidized, and they pass into the chloridizing portion before the metals are in a condition to combine with the chlorine gas. This defect of the furnace is of little importance as regards its availability in all cases where the ore is not of the very basest kind, and but very few of the ores of the Pacific coast are of such a character.

The White is a similar furnace to the Howell, and, except that the flame and the ore enter the cylinder together, there is no great difference in points of construction. As regards expense of building, power required to run it, repairs, continuous feed and discharge, and labor required in manipulation, what has been said of the Howell applies also to this furnace. The fact that ore and flame enter the cylinder together precludes the possibility of distinct oxidation and chloridation periods. It is necessary to crush the ore fine, and the flue-dust is not usually chloridized as perfectly as the ore, nor can the basest ores be worked in it.

The Brückner cylinder, like the reverberatory furnace, has the advantage that an ore which is in any way susceptible of chloridation no matter how base, can be effectually chloridized in it; but, unlike the latter furnace, the efficiency of the work does not depend upon the diligence of the roaster, whose business it is to stir the pulp. It chloridizes to a somewhat higher per cent. than the other mechanical furnaces, and requires less salt. These facts may be accounted for when it is remembered that the ore can be roasted sufficiently before the salt is added to form the sulphates necessary to the proper decomposition of the salt, so that when that is added nothing but an elevation of temperature is needed to complete the process. On the other hand, it is a furnace which in some respects is inconvenient to handle and requires more power to drive, as the charge is a heavy one (from 3,000 to 4,000 pounds). The bricks forming the lining occasionally fall out, and the furnace needs, as a rule, more repairs than the others. It also consumes more wood, for reasons already explained. It is, however, the best mechanical furnace known for working very base ores.

The Stetefeldt furnace has the following advantages: Less power is required than for any of the other furnaces, and but few repairs are needed; it has a continuous feed and discharge; it requires the least fuel and the least labor of any per ton of ore roasted; and its ore and flue-dust are chloridized to a very high percentage. Its principal drawbacks are its original cost, for it must be built of good brick and well anchored, and that the ore worked in it must have nearly a fixed percentage of sulphur and nearly a fixed value in silver to be well roasted. Very low and very high grade ores cannot, therefore, be worked to advantage. The loss of silver and of gold in this furnace is very slight, and where there is a large quantity of ore to be roasted per diem and the ore is uniform and of a suitable composition this furnace is much to be recommended.

The reverberatory furnace does excellent work on all kinds of ore, and admits of any necessary modifications in the process; but the fact that it involves so much manual labor, thereby making the process a very expensive one, prevents it from being employed in any but very exceptional cases. Its first cost, however, is much less than that of a cylinder or shaft furnace, and also, of course, considerably within that of an O'Harra mechanical reverberatory. As the price of labor is nearly the same at points near main lines of communication and at localities far removed from them, while the cost of freight on the iron and the fire-brick involved in the construction of most of the furnaces increases very rapidly with the distance from the railroads, the reverberatory furnaces may often be the most economical in remote districts.

TYPICAL GOLD AND SILVER MILLS.

As specimen works, illustrating the two extremes in the amalgamating processes, the Ontario mill and the two mills of the Homestake company are mentioned below in some detail. The former is a type of the dry-crushing, chloridizing-roasting, pan-amalgamating mill, working on base silver ores. Its process is complicated, involving the repeated rehandling of material; and though in this mill the costs are brought as low as is consistent with good results the large force, extensive plant, and quantities of supplies required make the operation an expensive one. The latter works represent the treatment of free-gold ores brought to the greatest simplicity. Between these two extreme cases are many gradations in type, ranging from the simple water-power gold mills of California and the free-milling silver reduction works of Nevada, Arizona, Idaho, Montana, etc., up to mills employing processes comparable with that of the Ontario, and often working at a still greater expense.

THE ONTARIO DRY-CRUSHING 40-STAMP SILVER MILL.

The mill of the Ontario Silver Mining Company at Park City, Uintah district, Summit county, Utah, is a fine example of works in which dry-crushing and chloridizing-roasting precede pan amalgamation. The mill was built in the autumn of 1876. Stamps were dropped January 26, 1877, and, aside from a delay of six weeks in the latter part of 1878, when the hoisting works of the company were destroyed by fire, and another of nine weeks in the spring of 1880, while furnaces were in construction, besides stoppages of a few days for cleaning up and for minor repairs, the mill had been running steadily until visited by the census expert for Utah. The building is a large and commodious but somewhat unsymmetrical structure, many modifications and additions having been made. The plant and methods are designed for close and economical work. The total cost to the close of 1880 was $321,000.

PLANT.—The plant includes the following machinery: Two Blake rock-breakers; one Lane & Bodley hydraulic lift; two rotary driers, 24 feet long and having a grade of 9 inches; eight Cochrane & Hendy self-feeders, improved by the local engineer; forty 850-pound stamps, drop 8 inches, speed ninety-four drops per minute; double-discharge dry-crushing batteries, having No. 30 and No. 26 brass-wire screens; twenty-four combination pans; twelve settlers; one agitator; two clean-up pans; one quicksilver elevator; four retorts and two melting furnaces; one battery of five 400-pound stamps, 7- to 7½-inch drop, speed ninety per minute, and screen No. 20, for crushing salt; two Stetefeldt roasting furnaces, with necessary flue-dust chambers; one Howell roasting furnace and dust chamber; one 250 horse-power horizontal engine and four horizontal boilers. In connection with the mill the company also owns boarding and lodging houses, barns, water-tanks, blacksmith and machine shop, etc.

The ore, the hauling of which is charged to the mill, is dumped from the wagons upon iron grizzlies 9 feet long, set at an inclination of 30° and having 2-inch spaces. That which passes over the grizzlies goes to the two Blake rock-breakers and then falls with the fine ore into the mill bins, which hold several thousand tons. Cars are filled from chutes and hoisted 8 or 10 feet by the hydraulic lift. A laborer empties them into hoppers over the rotary driers, to which an ingenious self-feeding apparatus of local construction is attached. One-tenth of a cord of wood per ton of ore treated is used in drying. The driers dump upon the floor through a side chute. The ore is then shoveled into wheelbarrows and taken to the battery self-feeders. Screw conveyors and belt elevators next carry it to hoppers over the Stetefeldt furnaces.

Salt is not crushed with the ore, but in a separate 5-stamp battery. This battery is fed by hand, and discharges through a chute into a wheelbarrow, from which it is dumped into a hopper. Immediately under this hopper, and connecting with it by a short tube, is a small horizontal screw conveyor driven by a cone pulley. This carries the required amount of salt to a tube leading to the main ore conveyor, immediately beneath. By means of the cone pulley the speed of the conveyor can be regulated, and thus the required percentage of salt be added. At the time of examination the proportion of salt used was 18 per cent. of the weight of the dry ore. The pulp and salt are intimately mixed in the conveyors before reaching the furnaces.

The battery guides consist of pieces of oak wood having the fiber parallel to the stems, mortised into the ordinary guides. These are found to wear much longer than guides cut in the usual manner. The new die surface is about level with the lower edge of the discharge. The end and middle stamps are given a trifle more fall than Nos. 2 and 4. The order of drop is 1—4—2—5—3.

Steel shoes and dies had been in use for two years with fair results. They wear about a year, at the end of which time the surfaces are extremely irregular, usually showing several depressions from three-quarters of an inch to 1½ inches in depth and having worn away about one-sixth. The shoes are then refaced and used another year. Experiments were made with a combination of steel shoes and cast-iron dies, but no definite results had been reached at the time the mill was reported on.

Screens usually last thirty days. Only three stems had been broken in four years, the part affected by the guide bearings wearing out first. One set of twenty stems lasted three and one-half years; the other set only two and one-half years.

FURNACES.—The Stetefeldt furnaces at this mill are 38 feet from screen to hearth. The walls are made of double courses of brick, between which is a 3-inch space filled with ashes. The water-jackets at the top of the furnaces have been removed, as they were found to be unnecessary. Each furnace treats from 22 to 25 tons per day, but the possible capacity is somewhat greater. The amount of wood consumed in roasting is one-fifth of a cord per ton. The arches and bridge walls of the fire-boxes usually require repairing two or three times per year, the work occupying three or four days at each time. Charges are drawn into cars every hour from the hoppers, but only the lower portion of the ore is removed, so that practically ore remains in the furnace-pit from four to six hours. It is then dumped in a heap on the cooling floor and allowed to remain from sixteen to twenty-two hours before being wet down. It has been found that this increases the amount of silver chloridized several per cent. beyond that obtained when it is wet down immediately. During the year February, 1879, to February, 1880, the average percentage chloridized was 86¼. At the time of the agent's visit, owing to improvements and experience, the average for ten days for both furnaces was 92.7 per cent.

The flue-dust assays much higher than the roasted pulp at the base of the shaft, but no average results were obtained. The flue-dust chambers are of brick, 10 feet wide, 16 and 18 feet high, and 30 and 44 feet long. These are divided by partition walls into smaller chambers 3½ feet wide. From these flues of considerable length lead to brick stacks upon the hill-side. The flue from the dust-chambers of the new Stetefeldt furnace is of brick, 300 feet long and 3½ by 5½ feet. The chimney, the base of which is perhaps 80 feet above the floor of the furnace room, is 87 feet high. It is about 5 feet square at the base and 3 feet square at the top.

A Howell furnace with flue-dust chambers and all the necessary appliances was built in the winter of 1879–'80. It is 24 feet long, with 52- and 62-inch cylinders. It ran only about twenty-five days, during which time a series of assays were made for the purpose of comparing the Stetefeldt with the Howell furnace, and the latter was then shut down. The comparative experiments were made by Mr. Stetefeldt.

AMALGAMATION.—After having been wet down the ore is taken in a car to the pan room. It passes over a pair of platform scales, where it is weighed, and two spoonsful are taken for the chloridation sample. The sampling spoons used are similar to a butter-tester in shape.

The pans, 5 feet 3 inches in diameter by 2 feet 10 inches deep, are charged with about 2,800 pounds (dry weight) of pulp, heated with live steam and run eight hours, the muller making sixty-five revolutions per minute. The temperature of the pans when first heated is 190° F.; at the end of two hours, 178°; in five hours, 170°; and shortly before discharging, 160°. Three hundred pounds of quicksilver are added per pan immediately after heating. Formerly it was the custom to add 1 and 1½ pounds of zinc per charge, and sometimes salt and iron turnings. This has been discontinued, as it lowered the fineness of the bullion from fifty to one hundred thousandths, and did not appreciably increase the percentage amalgamated or save the castings to any considerable extent.

The wings are not upon the sides of the pans, as has been ordinarly the practice, but are attached to a frame of 3- by 4-inch stuff running across the top and center of each pan. The wings are 1-inch boards, 2 feet long and 14 inches wide. The longer dimension is placed vertically and the shorter obliquely to the radii of the pan, 2 inches being left between the wing and the side of the pan and 7 inches between the wing and the sleeve of the muller.

The settlers are 8 feet by 3 feet 9 inches, make thirteen revolutions per minute, and retain a charge four hours. After a pan has been discharged into a settler, the latter is filled with water and run three and one-half hours, when it is emptied in half an hour by pulling two plugs.

The settler sands all pass through one agitator, 4 feet deep by 9 feet 4 inches in diameter, making twelve revolutions per minute, which had just been placed in position. No accurate data regarding the amount of quicksilver or amalgam caught could be obtained. Most of the tailings have been caught since September, 1878, and 20,000 tons, assaying $18, were supposed to be in the reservoirs at the time of examination. The quicksilver is strained from the amalgam through No. 0 duck bags. It falls into iron tubs, and thence runs to a tank. A rubber-belt conveyor, with small Russia iron buckets, takes it to a quicksilver reservoir above. From this there are two main lines of half-inch pipe, with branches to each pan, terminating in a hollow cylinder. This is supposed to hold 300 pounds. When it is full and the pan-charge ready, a plug in the bottom is withdrawn and the quicksilver runs into the pan.

SAMPLING.—The ore is weighed in the wagons as it comes from the mine. It is supposed to contain about 11 per cent. of moisture. The battery sample is taken from the screens at the top of the Stetefeldt every hour, these hourly samples constituting a 24-hour sample. After assaying, the percentage of salt used is allowed for in calculating the true battery-sample assay. Tailings samples are taken as usual. The chloridation sample previously mentioned as taken from the car gives the chloride assay by the common hyposulphite method; but a portion of this car sample is also leached, dried, and weighed, to determine its percentage of moisture and the quantity of soluble salts present. Assuming that the soluble salts are sodium sulphate and excess of sodium chloride, the assays made before and after roasting afford a means of comparison which is certainly better than nothing, and possibly sufficient to check the working of the furnace. As a means of comparing different roasting furnaces, such as the Stetefeldt and the Howell, however, it is manifestly insufficient, since soluble metallic salts are also formed in the roasting process. The percentage of soluble salts in the Ontario ore, after treatment in the Stetefeldt furnace, is from 18 to 28.

CLEANING UP.—The mill is thoroughly cleaned up once in three or four months, or when it shuts down for repairs; but a partial clean-up is made at the end of each month.

RETORTING.—The amalgam is taken on a car to the retort room. The retorts are circular, 14 inches in diameter, 5½ feet long, and have the vent at the upper part of the back end. They are charged with about 2,000 pounds, which fills them two-thirds full. Retorting is finished in from six to six and one-half hours from the time of sealing, a dull red heat being maintained during the last four hours. The mercurial vapor is condensed in a discharge pipe, 2 inches in diameter, which passes through a cylindrical water-jacket 4 feet long.

The old retorts lasted one and one-half years, running every other day. The newer ones are much thicker, and after having been in use two years appeared nearly as serviceable as ever.

MELTING.—The bullion, which is about one-seventh of the amalgam in weight, is melted in No. 50 Dixon crucibles, in fire-brick furnaces about 17 inches square. A little borax is added to flux the iron and sand, and the skimmings are saved and put through the battery once a year. The fineness of the bullion is 0.680 to 0.840, averaging 0.750, the base portion being chiefly copper.

LOSS OF QUICKSILVER.—Actual weighings have shown a loss of quicksilver of nearly 3½ pounds per ton of ore worked.

LOSS OF IRON.—It was estimated that during the year ending February 1, 1880, the amount of iron consumed in the mill was 21½ pounds per ton of ore treated. This included the iron of battery shoes and dies, pan-shoes and dies, mullers, sleeves, retorts, etc.

LABOR.—The regular mill force for twenty-four hours is as follows:

TABLE CXIV.—FORCE EMPLOYED AT THE ONTARIO MILL.

Class.	Number employed.	Length of shift hours.	Wages per shift.	Class.	Number employed.	Length of shift hours.	Wages per shift.
Foreman	1			Engineers	2	12	$4 00
Chief engineer	1			Firemen	2	12	3 50
Assayer	1			Salt feeders	2	12	4 00
Clerk	1			Watchmen	2	12	3 00
Night boss	1	12	$4 50	Carpenters	3	10	3 00 to 4 00
Ore weigher	1	10	4 00	Machinists	2	10	4 00
Rock-breaker	1	10	3 00	Machinists' helpers	2	10	3 00
Carmen and drying-furnace feeders	2	12	4 00	Retorter	1	10	4 00
Ore driers	12	8	3 00 and 3 50	Melter	1	10	3 50
Battery feeders	3	8	4 00	Storehouse keeper	1	10	3 75
Amalgamators	4	12	4 50	Blacksmiths	2	10	3 25 and 5 00
Carmen	2	12	4 00	Wood haulers and team	2	10	7 50
Furnace men	6	8	4 00	Assayer's helper	1	10	2 50
Cooling-floor men	12	8	4 00	Tailings-pit man	1	10	3 00

EXPENSES AND RESULTS.—During the census year the operations of the mill showed the following figures:

Amount paid on labor account	$127,404 81
Amount paid for mill supplies	$151 537 78
Amount paid for repairs and construction	$65,580 00
Gross ore worked in year ....tons	13,858
Net ore worked in year (estimated) ....do	11,481
Total number of days in operation	283
Total bullion product in year	$1,344,723 73
Total discount on bullion	$171,241 60

During the company's fiscal year, from February 1, 1879, to February 1, 1880, 15,372 gross or 12,342¾ net tons were treated. The average assay value of the ore was $130 94, and the average tailings assay was $17 49. The percentage extracted was 88. The bullion product for that period was $1,425,003. At the time of examination, however, battery samples for 10 days averaged $149 65, and tailings samples for the same time $11 12.

The cost of milling ore is $11 69, or $15 including hauling, superintendence, etc. Supplies cost: Coal, $8 25 and salt $8 per ton; wood, $4 per cord.

THE HOMESTAKE 80- AND 120-STAMP GOLD MILLS.

The two large mills of the Homestake Mining Company, at Lead City, in the Black hills of Dakota, are examples of the economy and simplicity to which the treatment of free-gold ores on an extensive scale has been brought. The 80-stamp mill, known as the "Homestake", was completed in July, 1878, at a cost of $164,500. The 120-stamp, or "Golden Star", mill cost $251,500, and was finished in September, 1879. Both mills are situated immediately at the mine. The process is wet-crushing, battery amalgamation, and amalgamation upon outside plates. The capacity of the larger mill is rated at 325 tons per twenty-four hours, and that of the 80-stamp mill at 225 tons.

PLANT.—The 80-stamp mill contains the following machinery: Four Blake improved No. 5 rock-breakers; sixteen Hendy self-feeders; eighty 750-pound stamps, in batteries of five each, drop 8½ inches, 84 per minute, falling in the order 1—4—2—5—3; one clean-up pan; one No. 5 Knowles steam pump, one 20½- by 40-inch engine, 190 horse-power; two 16-foot horizontal boilers.

The 120-stamp mill has six Blake improved No. 5 rock-breakers; twenty-four Hendy self-feeders; one hundred and twenty 750-pound stamps, in batteries of five each, drop 8½ inches, speed 82 per minute, falling in the same order as in the other mill; one clean-up pan; one Knowles No. 5 steam pump; one 26- by 60-inch engine, 300 horse-power; four 16-foot horizontal boilers.

The batteries of these mills are arranged back to back, with space for the feeders between, the discharge being in opposite directions.

LABOR.—The force employed in these mills is very small, the process being very simple and the operations as nearly automatic throughout as it is possible to make them. The regular mill staff is four. The pay-rolls are as follows:

TABLE CXV.—FORCE EMPLOYED AT THE HOMESTAKE 80-STAMP MILL.

Class.	Number employed.	Length of shift.	Wages per shift.
		Hours.	
Amalgamators	5	12	$4 00
Carpenter	1	12	5 00
Day laborers	9	12	3 00
Engineers	2	12	4 00
Feeders	2	12	3 50
Foremen	2	12	4 60
Machinist	1	12	4 00
Watchman	1	12	3 50

TABLE CXVI.—FORCE EMPLOYED AT THE GOLDEN STAR 120-STAMP MILL.

Class.	Number employed.	Length of shift.	Wages per shift.
		Hours.	
Amalgamators	5	12	$4 00
Day laborers	11	12	3 00
Engineers	2	12	4 00
Feeders	2	12	3 50
Foremen	2	12	4 60
Machinist	1	12	3 75
Millwright	1	12	4 00
Watchman	1	12	3 50

COST OF MILLING.—These mills had, up to August, 1880, crushed 275,282 tons, and the total gross product to that date was $1,924,769 52. The mine account during this period showed an average cost of $1 32 per ton, and incidental expenses amounted to 78 cents per ton. The expense of milling is shown in the following table:

TABLE CXVII.—COST OF TREATMENT AT THE HOMESTAKE MILLS.

Items.	80-STAMP MILL (153,372 tons treated).		120-STAMP MILL (121,910 tons treated).	
	Cost per ton.	Amount expended.	Cost per ton.	Amount expended.
Total	$1 30.18	$199,661 01	$0 77.70	$94,726 48
Labor	0 53.36	81,837 72	0 33.17	40,433 75
Supplies	0 10.36	15,887 88	0 02.41	2,938 75
Water	0 11.66	17,874 22	0 10.29	12,548 51
Wood	0 27.88	42,759 35	0 24.76	30,186 03
Machinery	0 16.20	24,847 88	0 03.26	3,977 20
Candles	0 00.77	1,185 75	0 00.13	152 70
Oil	0 02.42	3,721 00	0 01.09	1,323 75
Quicksilver	0 01.79	2,751 71	0 01.19	1,455 00
Teams	0 02.15	3,295 00	....	....
Coal	0 01.79	2,742 20	....	....
Lumber	0 01.54	2,361 16	0 01.33	1,631 30
Timber	0 00.26	307 14	0 00.07	84 40

Concerning the working of these mills Superintendent McMaster writes:

The gold contained in the ore is mostly coarse, and is easily saved in the batteries. The ore is in the highest degree free milling; even that from the lower levels with several per cent. of iron pyrites is readily amalgamated, the concentrated sulphurets from the tailings showing but $8 per ton. The average loss of gold while working ore containing from $8 to $10 per ton is but $2, as proved by fire assays of the tailings, indicating a yield of 75 to 80 per cent. During the year ending September 1, 1880, both mills had been kept continuously at work, stopping only to clean up on the first and fifteenth of each month. They are in good condition and repair, and the result of their work is in every way highly satisfactory. The amount of ore crushed per stamp in twenty-four hours has been increased by improved working to 3 tons. The quantity milled per month subsequent to the census year averaged 16,700 tons, while previously 2 tons per stamp, or 12,000 tons per month, was regarded as good work.

The average gross yield of the ore to June, 1879, was $9 69 per ton. Since then it was found advantageous to extract and mill all the rock between the walls of the vein. This lowered the grade of the ore somewhat, but the gross amount milled has been increased in greater proportion, while the cost of mining has been correspondingly reduced. The yield of ore from September, 1879, to February, 1880, varied from $4 25 to $5 60 per ton. Since that date it has been increased by the ore of higher grade extracted from the 100-foot level, and now averages $7 95 per ton.

Blankets and concentrating machinery are not employed, as on careful examination and experiment their use was found unnecessary, the pulp being easily amalgamated in the batteries and in passing over the plates. The most perfectly concentrated tailings will not assay per ton more than the rock from which they are derived. The loss of quicksilver in working is very small, being sixteen one-hundredths of an ounce per ton of ore milled, at present prices equivalent to half a cent per ton. The increased capacity of the mills has been aided by improved automatic machinery, reducing the cost of milling to 90 cents per ton in the 80-stamp mill and 71 cents per ton in the 120-stamp mill. The mills are run by steam. Firewood is delivered at $4 75 per cord. The cost of fuel is 28 cents to the ton of ore crushed. Water is purchased for the mills from the Black Hills Canal and Water Company at the rate of $2 per stamp per week. Forty miner's inches are required. The cost is 11 cents per ton of ore crushed. The water company obtain an abundant supply from the head of Whitewood creek, a distance of 10 miles. The average fineness of the bullion in the first year's working of ore from the open cuts on the surface was 0.825 gold, 0.165 silver, and the value per ounce $17 25; at the close of the census year the average fineness was 0.800 gold, 0.170 silver, and the value per ounce $16 75, most of the ore coming from the 100-foot level.

ARRASTRAS.

The arrastra is a simple and effective device for reducing ores, the legacy of the primitive but much underrated mining system of Mexico. Notwithstanding the slowness of its operation, it has not been entirely supplanted by the stamp- and pan-mill, and still has a large field of usefulness. The arrastra is the prototype of the amalgamating pan; indeed, it combines to a certain extent in a single apparatus the functions of stamps, pan, and settler. The leading advantages of the arrastra are: simplicity of construction and cheapness of first cost, thus placing it within the means of mine owners having but small capital and enabling it to be used during the period of development of mines, economy in working, and effectiveness in saving the precious metals of ores adapted to it when judiciously managed. Its disadvantages are the relatively small quantity of ore which can be treated in a given time and the fact that certain classes of ores are not suited to it, though this latter objection may also be said to apply to any particular type of stamp- and pan-mill.

Arrastras are still largely used in the United States, but of the several hundred scattered through the country a few only were selected for examination. Table CXVIII gives the number of arrastras, number and weight of drags in each, and character of motive power of thirty-two works, in which are employed one hundred and eleven arrastras.

TABLE CXVIII.—SPECIMEN ARRASTRA MILLS.

ALABAMA.

County and district.	Works.	Number of arrastras.	Number of drags.	Weight of drags.	Motive power.
CLEBURNE.				*Pounds.*	
No. 33	Houston & Pinson	1	6	100 to 300 each	8 horse-power engine; also runs battery.

ARIZONA.

County and district.	Works.	Number of arrastras.	Number of drags.	Weight of drags.	Motive power.
MOHAVE.					
Hualapai	Gross & Smith	2	2 each		4 horses.
YAVAPAI.					
Tiger	South Oro Belle	2	4 each	400 each	5 horse-power engine.
Walker	Thunderbolt	4	4 each	175 each	20 horse-power engine.
Weaver No. 2	Johnson	2	4 each	250 each	2 horses.
	Nash	3	6	200 each	1 horse for each arrastra.

CALIFORNIA.

County and district.	Works.	Number of arrastras.	Number of drags.	Weight of drags.	Motive power.
CALAVERAS.					
Angels	Leper, Keys & Co	6	3 each	300 to 500 each	Overshot wheel, 28 feet diameter, 50 inches water.
Do	Potter	2	3 each	200 to 500 each	Overshot wheel, 20 feet diameter, 25 inches water.
Independence	Champion	2	4 each	750 each	Overshot wheel, 40 feet diameter, 50 inches water.
FRESNO.					
Potter Ridge	Fresno Enterprise	3	4 each	800 to 1,000 each.	
PLUMAS.					
	Italian	35	Usually 4 each	500 to 2,000 each.	Hurdy-gurdy wheels.

COLORADO.

County and district.	Works.	Number of arrastras.	Number of drags.	Weight of drags.	Motive power.
BOULDER.					
Colorado	George Bacholder	2	2 each		Water-power.
OURAY.					
Iron Springs	Rock Point	4	6 each	200 each	Turbine.

IDAHO.

County and district.	Works.	Number of arrastras.	Number of drags.	Weight of drags.	Motive power.
BOISÉ.					
Granite	Iowa	1	8	250–600 and 800	40 horse-power engine; runs battery also.
LEMHI.					
Yankee Fork	Estes	1			Leffel turbine, 13½ inches diameter; 65 feet fall.
Do	Norton's	2	4 each	*a* 1,000–1,200 each	Overshot wheel, 15 feet diameter, 5 feet face; 150 inches water.
OWYHEE.					
Carson	Scales & Wagner's	2	4 each	250 to 300 each	In winter, 30 horse-power engine; other 8 months, 20 feet overshot wheel, 4 feet face.
Do	Trask's	2	4 each	300 each	Overshot wheel, 24 feet diameter, 2 feet face.

a When new; used until worn to 250 or 300 pounds.

MONTANA.

County and district.	Works.	Number of arrastras.	Number of drags.	Weight of drags.	Motive power.
DEER LODGE.					
Black Tail	Black Tail	2	4 each	2,000 to 3,000	Water-power.
	Glen Egbert	1	4 each	500 each	Overshot wheel, 14 feet diameter, 2½ feet face.
LEWIS AND CLARKE.					
Oro Fino Gulch	Schafer	2	One 4; one 8	500 each	Overshot wheel, 33 feet diameter.
MADISON.					
Silver Star	H. Siegchrist	2	4 each	800 each	Overshot wheel, 20 feet diameter, using 60 inches water.

NEVADA.

County and district.	Works.	Number of arrastras.	Number of drags.	Weight of drags.	Motive power.
ESMERALDA.					
	E. R. Willis	1	5	120 each	Undershot wheel.
STOREY.					
Gold Hill	B. B. Chandler	1	4	200 to 500 each	Overshot wheel, 24 feet diameter.

TABLE CXVIII.—SPECIMEN ARRASTRA MILLS—Continued.

NEW MEXICO.

County and district.	Works.	Number of arrastras.	Number of drags.	Weight of drags.	Motive power.
GRANT.				*Pounds.*	
Pinos Altos	Skillicorn's	4	4 each	80 each	20 horse-power engine.

NORTH CAROLINA.

County and district.	Works.	Number of arrastras.	Number of drags.	Weight of drags.	Motive power.
NASH.					
Griffith township	Mann	10	4 each	200 to 400 each	40 horse-power engine.

OREGON.

County and district.	Works.	Number of arrastras.	Number of drags.	Weight of drags.	Motive power.
BAKER.					
Chicken Creek	McCord	2	4 each	300 to 500 each	Overshot wheel, 24 feet diameter, 12 horse-power.
GRANT.					
Granite	Beagle	1	6	300 each	Hurdy-gurdy wheel, 10 horse-power.
JOSEPHINE.					
Yank	Sugar Pine	2	4 each	600 each	Water wheel, 20 horse-power.
YAKIMA.					
Peshaston	Cooper & Lockwood	3	6 each	200 each	Turbine, 20 feet diameter, 13 feet fall, 9 horse-power.
Do	Shafer	1	4	500	Horizontal hurdy-gurdy.
Do	Witer & Miller	3	5 each		Hurdy-gurdy, 8 horse-power; and breast wheel, 20 horse-power.

OWNERSHIP.—The greater number of arrastras are owned in connection with small but productive mines. Of those which are owned independent of mining property a few of the larger and best-equipped works are operated as custom mills, while the others are run on tailings. A large number of water-power arrastras are engaged in working over the tailings of the Comstock and other veins. Of the arrastras reported upon nearly all were constructed since 1876. As they are so often built merely as temporary make-shifts, it is natural that their average life should be short. Incorporated companies seldom own arrastra mills.

COST OF PLANT.—The crudest form of arrastra, to be operated by mule-power, can be built for $150. From this the cost of construction ranges upward to $1,000 for a substantial and large apparatus. Other plant is also involved in the better class of arrastra mills, such as the building, engine or wheel, ditches, flumes, pipes, retorts, and when pans, etc., are added, as is sometimes the case, the mill becomes an expensive affair. The cost, complete, of twenty-six mills, having ninety-two arrastras, is reported at $137,590, or an average of $5,292 for each works, and of $1,495 per arrastra, including all other plant. These mills, however, are of better construction than is usual, and their average cost is considerably above that of ordinary arrastras.

CONSTRUCTION.—The arrastra in its simplest form consists of a circular bed of rock from 6 to 10 feet in diameter, with walls of vertical planks, having an upright pivoted post in the center, from which extend two or four fixed horizontal arms. Stone drags, weighing usually from 200 to 1,000 pounds each, are attached by ropes or chains to the extremities of the arms, and are slowly drawn around by the rotation of the latter. The depth is usually between 18 and 30 inches. The pavement and drags are of the hardest rock conveniently obtainable, commonly granite or basalt. Sometimes flinty quartz is used, but a coarse-grained rock which will not become too highly polished is preferable. The pavement is laid with much care, and should be quite solid, so that while the amalgam may collect in the small interstices there should be as little loss as possible through leakage into the ground beneath. Hydraulic cement is sometimes used in setting the bed. The paving is usually 12 inches thick when new.

An improved apparatus, called the Americanized arrastra, has been introduced by Mr. A. B. Paul, of San Francisco. It is a double iron pan, the inner portion provided with a grinding muller similar to that used in common amalgamating-pans, while a set of drags travels in the outer circle, which is 14 inches wide and 5 inches below the pan muller. It is run by water- or steam-power, and is driven by bevel gear beneath the arrastra. A revolving amalgamated copper circle acts as do the aprons of gold batteries. The sizes made are 6 and $8\frac{1}{2}$ feet in diameter. The latter size requires 5 horse-power, weighs $4\frac{1}{2}$ tons, and has a capacity of from 7 to 10 tons in twenty-four hours, according to the hardness of the ore and the fineness to which it has been reduced before feeding.

DRAGS.—The number of drags used ranges from two to twelve per arrastra, the most common number being four. There is also a great variation in their weight, the lightest reported being only 80 pounds, and the heaviest one ton, the mean weight, when new, being somewhat above 300 pounds each. The usual speed is from ten to fourteen revolutions of the arms per minute for power arrastras, the range reported being from four to eighteen. The outer drags in a 10-foot arrastra move at the rate of about 400 feet per minute when the arms make fourteen revolutions, though the average is between 200 and 300 feet per minute. The drags are made of the same rock as that used for the pavement.

LABOR.—One man per shift can take care of two arrastras. Some water-power arrastras working on tailings are so arranged that the only attendance needed is in feeding and discharging them, so that practically the labor required is less than the constant work of one man. The owner often does all the work. Continuous arrastras receive no attention other than that demanded for repairs.

POWER.—The smaller arrastras are worked by a single mule or horse. When water-power is obtainable a small overshot, hurdy-gurdy, or turbine wheel is employed. The larger and more complete arrastra mills, which also include other apparatus, such as rock-breakers or stamps, are driven either by water-power or steam, and sometimes by both, using steam only in months when, because of drought or freezing, water is not available.

CAPACITY.—This varies greatly with the completeness of the plant and the character of the treatment, ranging from less than one ton to two tons per day for arrastras of the simplest pattern. In larger works, where rock-breakers and stamps relieve the arrastra of part of its duty, so that it acts simply as a grinding pan, the capacity is dependent upon the length of time necessary to complete amalgamation. A 12-foot arrastra driven by steam- or water-power, with heavy drags making fifteen revolutions per minute, may dispose of two charges, of 4,000 pounds each, of ore of medium hardness in twenty-four hours if very close working is not demanded, but this appears to be the limit of capacity. The annexed table shows the amount of ore treated during the census year by twenty-two specimen arrastra works, having fifty-three arrastras, few of which, however, were running very constantly. The average per mill was 413.75 tons, and for each arrastra 171.70 tons.

TABLE CXIX.—ORE TREATED BY TWENTY-TWO SPECIMEN ARRASTRA MILLS IN THE CENSUS YEAR.

State or territory.	Number of works.	Number of arrastras.	Tons treated.
Total	22	53	9,101¼
Arizona	4	9	195
California	4	13	3,205
Colorado	2	6	284
Idaho	2	4	2,022¼
Montana	3	5	2,250
New Mexico	1	4	500
Oregon	6	12	645

TENOR OF ORE TREATED.—Arrastras are usually employed in working free-gold ores, though they also produce doré bullion. Very little ore as low in grade as $10 per ton is reported to have been worked in these mills, the average value being somewhat higher than that of ore treated in pan mills.

PROCESS.—The ore, after having been broken by hand or by a rock-breaker or stamps, is fed into the arrastra, a sufficient quantity of water being added to keep the pulp at the proper consistency. It is not customary to add quicksilver to the charge until it has been ground from two to four hours, but the chemicals, when used, are generally added on charging, though sometimes they are put in with the quicksilver. The quantity of quicksilver used is governed by the contents of the ore. In working gold ores the object is to produce a pasty amalgam rather than a liquid one, while in treating silver ores larger quantities are used, sometimes amounting to more than a pound of quicksilver to the ounce of silver in the ore. As assaying is not often practiced in connection with arrastra mills, the millman judges the tenor of the ore being worked by panning out a sample of the pulp. No chemicals are commonly used with gold ores, though potassium cyanide in solution is sometimes employed to "liven" the quicksilver, and lye or wood ashes is used for the purpose of saponifying any grease which may have found its way into the arrastra. With silver ores bluestone (or copperas) and salt are used, the quantities of each being very indefinitely regulated, as is the case in pan amalgamation. The length of time occupied in working a single charge is usually ten to twelve hours. Toward the close the pulp is thinned by the addition of water, and the speed of the drags is diminished to give the amalgam an opportunity to settle. The arrastra is discharged by withdrawing successive plugs, arranged as in the common settler, or by opening a gate in the side. A clean-up is made after each run on custom ore, and usually once a fortnight if working steadily on the owner's ore.

The arrastra is sometimes used as an adjunct to the gold stamp-mill. At the Iowa mill, in Idaho, and in some works in California, the pulp flows continuously from the battery to a constantly working arrastra, and thence overflows through a gate near the top of the side upon amalgamated copper aprons and sluices. In this continuous process a flask of quicksilver is charged in the arrastra at the beginning of the run.

SCALES & WAGNER'S ARRASTRA MILL, OWYHEE COUNTY, IDAHO.

This mill is selected for description as a typical example of the better class of arrastra works. It is situated on the west bank of the Jordan river, 1 mile south of Silver City, at a distance of from half a mile to 2¾ miles from the mines of Carson district and 10 miles from Wagontown district by road. The mill was originally constructed in 1874 at a cost of $60,000, and was intended to be a 10-stamp wet-crushing silver mill. It was bought by the present owners in February, 1877, and considerably improved, though a part of the old machinery had been previously removed. The building is a substantial one, partly of stone with brick facings and partly of wood, and is conveniently arranged.

The mill is designed to treat the richer ores of the district. These are of two distinct classes—a free-gold ore, and a silver ore of which the principal mineral is probably argentite, also containing gold. The gangue is a moderately friable quartz.

The plant consists of a light 10-stamp battery, two arrastras, three half-ton Wheeler pans, two settlers, and a retort. Power is furnished during eight months of the year by a 20-foot overshot wheel with 4-foot face, and in winter by a 30 horse-power horizontal engine.

The stamps are used in place of a rock-breaker, to prepare the ore for the arrastras, and thus save wear and time in grinding. Dry-crushing is employed. The mortars are open in front, without screens, and the ore is delivered in variable fineness, ranging from dust to pieces as large as hazel-nuts. One battery of five stamps, running four hours a day, usually supplies sufficient ore to keep the arrastras working.

The arrastras are on a floor immediately in front of and below the battery. They are driven by a bevel-crown wheel from the counter-shaft, and power is communicated to the latter by a belt and pulley. The inside diameter is 10 feet. The paving and drags are a very hard, flinty quartz, obtained from a barren ledge about 1 mile south of the mill. Each arrastra has four drags, weighing from 250 to 300 pounds apiece. The speed is from twelve to fifteen revolutions per minute. A charge weighs 800 pounds dry, and the length of time occupied in working in the arrastra is ordinarily six hours, though with very rich rock the time is extended to eight hours. Four and a half hours after charging, when working gold or doré ores, the quicksilver is added, the proportion being determined by the assay of the ore. With silver ores the amalgamation is performed in the pans, the arrastras then being used merely for grinding. The average amount of quicksilver used is 125 pounds per charge, though with the highest grade of ore this is increased to 200 pounds. With gold ores a little lye is sometimes added. The treatment of silver ores in the pans is essentially the Washoe process, bluestone and salt being used in quantities ranging from 1 ounce to 2 pounds of the former and from 5 to 20 pounds of the latter, depending on the tenor of the ore. Both salt and bluestone are added on charging and in the dry state.

The arrastras are discharged by a plug-gate in the side, and the pulp flows directly into the pans through a wooden spout. Treatment in the pans and settlers is similar to that ordinarily followed in pan mills. A clean-up is made after working each custom lot, and these parcels of ore are sometimes very small, amounting occasionally only to a half a ton or so at a time of carefully-sorted $500 or $600 ore. Amalgam is strained by hand through a canvas bag and retorted in a cylindrical retort, 3 feet 6 inches long and 9 inches diameter, having the vent at the center of the rear end. The time occupied in retorting is usually seven hours.

Tailings are worked in the pans by the same process as in the first treatment, with the exception that larger quantities of bluestone are used, ranging from 4 to 5 pounds per charge.

The arrastra pulp is sampled every half hour as a check on the ore assays of customers, and tailings samples are taken from the settler discharge. The percentage of gold extracted by first treatment reaches 90 per cent., and of silver from 75 to 80 per cent.

The force employed in summer is as follows: Inside the mill four men, at $4 each, per twelve-hour shift; outside, two laborers, at $3 50 per ten-hour shift. In winter, when steam is used, two engineers, at $5 each per twelve-hour shift, are added to the force. During the census year the mill ran 330 days, and 22,440 hours' work were done. To treat a ton of ore required twelve and seven-tenths hours' labor (calculated as if done by one man). When in operation the mill runs full time.

As the mill works on high-grade ore, the process is necessarily an expensive one. The average cost of supplies during the census year, as will be seen from the accompanying analysis, was $3 95 per ton of ore treated. In this respect the operations of this mill are to be considered as representative not of the average practice, but of the best.

TABLE CXX.—SUPPLIES CONSUMED IN SCALES & WAGNER'S ARRASTRA MILL DURING THE CENSUS YEAR.

Items.	TOTAL CONSUMPTION.		CONSUMPTION PER TON TREATED.	
	Amount.	Cost.	Amount.	Cost.
Total	....	$7,018 75	....	$3 95
Red fir .... cords.	575	4,168 75	0.32	2 35
Quicksilver .... pounds.	3,000	1,440 00	1.69	81
Salt .... do...	14,000	420 00	7.89	24
Bluestone .... do...	3,000	750 00	1.69	42
Lard-oil .... gallons.	20	40 00	0.01	02
Chemicals and sundries	....	200 00	....	11

During the census year the mill treated 1,772¼ tons, of which less than 300 tons were tailings. The lowest yield of ore per ton was $38, the highest $600, and the average $115 85. The tailings treated yielded about $9 50 per ton. The total product was $205,331 75.

The custom charge at this mill is $15 per ton for lots of 100 tons and over, $16 for lots of from 50 to 100 tons, and higher rates for very small parcels.

Chapter VI.—LEAD SMELTING AT LEADVILLE, COLORADO.

By S. F. Emmons.

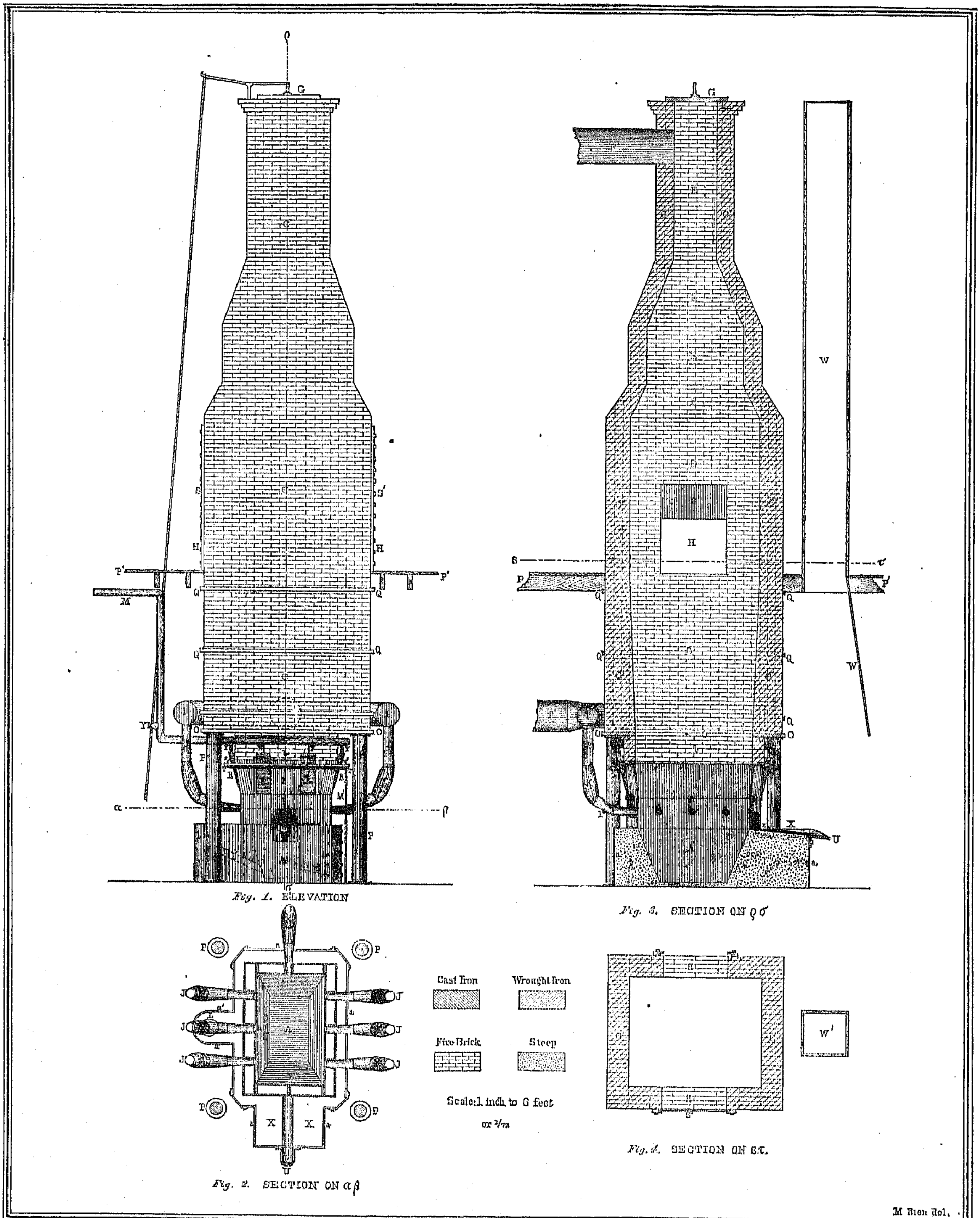
PLANT.

Introductory.—Although a very large amount of technical data on the various smelting works of the West was collected by the census experts, they were not found sufficiently complete to serve as the sole basis for a detailed description of the processes employed; nor do these works in general, as far as they are open to public inspection, present any features which are unusual or new to metallurgical science. At Leadville, however, where the numerous smelting establishments produce annually about $15,000,000 worth of argentiferous lead bullion, metallurgists have necessarily acquired an unusual amount of practical experience in the conduct of the operations of lead smelting and in the management of the business connected therewith. It has therefore been judged expedient to present a succinct account of the natural and economical conditions of smelting at this point, of the character of the plant, and of the processes employed. For this purpose recourse has been had to the MS. of a report by Mr. A. Guyard on the lead smelting of Leadville, which is to appear as an appendix to a monograph on the geology and mining industry of that district. An abstract of this report has been made by Mr. W. F. Hillebrand, and is supplemented by data obtained from census material and by himself and the writer personally, which appears in the following pages. In this the chemical investigations and calculations made by Mr. Guyard have been freely used, and the two illustrations which accompany it are taken from his plates; but all discussion as to the fitness or unfitness of methods employed, or of theoretical questions arising therefrom, has been avoided.

Topographical conditions.—An important condition in the disposition of smelting works, as well as of quartz-mills and other reduction works, is that the force of gravity may be used as an aid in handling the material to be treated, which is generally of a heavy and bulky nature. To such a disposition the surface character of the Leadville region is admirably suited by nature. The town itself is situated on a gently sloping mesa included between Evans gulch on the north and California gulch on the south, at the western base of the foot-hills of the Mosquito range, in which its ores occur. Along the high banks, which rise from the bottom of either of these gulches to the comparatively flat surface of the mesa, and at a sufficient elevation to allow room for the slag dumps below them, are located the various smelting works. They are thus situated so that from the mines an equally favorable grade leads either to the upper or the lower portion of the works, and the railroad which follows the surface of the mesa sends its branches on the level of or above the charging floor, and thus delivers its freight of fuel or of ore where it may descend through the various stages of the process until the final product, the bars of bullion, is obtained.

Disposition of the plant.—No less than sixteen smelting works have been built at Leadville in the few years that have elapsed since its mines were opened. Of these, however, a number have been closed, some temporarily, others permanently. The general plan in these works is that adopted elsewhere, and involves the occupation of two principal floors. The lower of these floors is at such a height above the adjoining valley bottom as to afford a convenient opportunity for dumping slag and other waste. On this floor the furnaces are built, and room is also commonly provided for the blower and the engine by which it is actuated. The furnaces are usually placed in a row within a single inclosure, but sometimes they have a wall intervening between them. The upper floor is on a level with the feeding-door of the furnaces, from 12 to 14 feet above the lower, and affords space for ore-bins, fluxes, mixing-beds, and the operations connected with charging the furnaces, such as crushing and sampling. When the slope of the ground is great, however, the storage bins for ore and fuel are sometimes placed at a still higher level, with passages for wagons between. One roof generally covers the whole establishment, with the exception of the offices, laboratory, and scales, which commonly occupy detached buildings.

Furnaces.—Shaft furnaces only are employed in Leadville. Of these two varieties were in use during the census year, the one presenting a circular horizontal cross-section, sometimes called the Piltz furnace, while the other is rectangular; but in 1882 the latter had entirely replaced the former. While the circular section presents advantages in the regularity of the descent of the charges, it is more expensive in construction, and the diameter of the hearth is limited by the strength of the blast; indeed, with any ordinary blowing-engine a round furnace can be successfully worked only when it is of very moderate dimensions. The rectangular or Raschette furnace, on the other hand, may be constructed with a width at the tuyeres corresponding to the strength of the blast-engines, and the production may be increased by increasing the length of the cross-section. The horizontal elongation of the furnace has its limits, indeed, as has been proved by the history of the rectangular Raschette furnaces in Europe, but the capacity may nevertheless be increased considerably above that of a circular furnace of similar construction without deleterious effects upon the working. In lead smelting, and especially in smelting argentiferous lead ores,



RECTANGULAR FURNACE

it is very undesirable to employ the high-pressure blasts required by a circular furnace of large diameter, since the higher temperatures which result promote the production of fumes which are only imperfectly recovered and are always difficult of treatment.

The rectangular and the round furnaces of Leadville are constructed on the same general plan so far as height, method of support, water-jackets, tuyeres, etc., are concerned, but the sizes adopted vary greatly, the limits of capacity being from 15 to 40 tons in twenty-four hours.

Plate I, Fig. 1, represents a rectangular furnace in elevation drawn to a scale of 6 feet to the inch, and Fig. 2 the same furnace in horizontal section at the tuyere level. Fig. 3 shows a vertical section of the same furnace on its longer diameter, and Fig. 4 a horizontal section at the charging-doors. The masonry shaft (*C*) rests upon a cast-iron plate (*O*), supported by pillars (*P*), in order that there may be no unnecessary weight on the lower portion of the furnace. The walls for some distance above and below the tuyeres are formed of sectional water-jackets (*B*), constructed of cast or wrought iron or of steel. In the furnace illustrated the water-jackets are twelve in number, firmly bolted together and provided with openings for the insertion of the tuyeres (*N*). A cold-water pipe (*M*) runs around the furnace above the water-jackets, and water is admitted to each of them by a faucet (*Y*). Outlets (*M'*) for the hot water and gutters (*T*) for its removal are also shown in the illustrations. The interval (*b*) between the water-jackets and the plate on which the shaft of the furnace rests is filled with fire-brick, which can be readily removed in case of necessity. The lower ends of the water-jacket rest upon the hearth (*X*). This consists of cast-iron plates (*a*) bolted together and lined with a thick coating of fire-brick or of "steep" (brasque), a mixture of fire-clay and coke-dust, either in equal parts, or in the proportion of two of the former to one of the latter. The usual form of the hearth is shown in the illustrations; this, however, is modified in detail, according to the working of the furnace and the judgment of the manager or smelter, changes in it involving no modification of the iron plates.

The hearth plates include also the lead well *L* and the so-called "siphon tap". A similar device was long ago applied to the small charcoal-iron blast-furnaces of Silesia, to permit of the manufacture of castings without tapping the furnaces; it was not successful, however, the iron chilling too rapidly for the object in view, owing to its high melting point. This arrangement was introduced into lead smelting in Eureka, Nevada, by Mr. Albert Arents with great success, and has since been widely adopted both in this country and abroad. The lead rises through the oblique tap-hole *L'* shown in Fig. 1 to the same level in the well which it occupied in the furnace, and can be baled into the molds at leisure, and without the disturbance of the furnace-working incident to the old method of tapping at long intervals.

At the end of the furnace just above the hearth an opening (*V*) is left in the water-jacket to facilitate the tapping of slag. This opening is filled with clay, in which a hole can be pierced when required, allowing the slag to pass through an inclined gutter (*U*), shown in the illustrations, into a slag-buggy. A hood (*W*) is generally placed over the tapping-hole to draw off the fumes emitted during the tapping. The number of tuyeres varies with the size of the furnace, depending mainly on the length of the cross-section. A tuyere is always placed at the end of the furnace opposite the slag-tap, and sometimes also above it; this last, however, is somewhat in the way, and is often omitted. A sliding valve (*l*) at the elbow of the nozzle admits of the inspection of the interior of the furnace. The tuyeres are connected with the main blast-pipe (*I*) by canvas hose (*K*), the flexibility of which permits their withdrawal from the furnace when necessary. This convenient device is, of course, applicable only when cold air is supplied to the furnace, as is almost invariably the case in lead smelting. Feed-openings (*H*) on the upper floor are closed by sliding doors (*S*). The furnace terminates upward in a short chimney (*E*) and may, in case of need, be run without the dust-chambers, with which it is connected by a flue (*F*) indicated in the illustration.

As an example of recent construction, the furnaces in the works of Eddy, James & Grant, at Denver, may be cited. There are eight of these furnaces of the same pattern, all built in the spring of 1882. The height to the charging-door is 18 feet; the dimensions 3 feet below the charging-door are 60 by 102 inches; at the top of the jackets, 48 by 92½ inches, and at the tuyere level, 36 by 80½ inches. Each furnace has ten tuyeres, four on each side and one at each end. The capacity of each is about 30 tons. The waste gases and fumes are drawn from all these furnaces at a point below the charging-doors into large dust-chambers connecting with a single stack—a very convenient arrangement so far as the comfort of the workmen is concerned; whether it is accompanied by any ill effect upon the working of the ore is regarded as uncertain, but each furnace is provided with an independent stack, to be used in the event of its proving desirable to return to the ordinary practice.

Plate II shows a furnace with a circular horizontal section on the same scale as the rectangular furnace illustrated on Plate I, and a comparison will show that the general principles governing the construction are the same in both. A main point of difference is in the anchoring, which in the square furnace is necessarily effected by bars (*Q*), while the same object is more conveniently attained, when the section is circular, by a shell of sheet iron (*J*) composed of plates about a quarter of an inch in thickness. The diameter of round furnaces at the tuyere level is from 33 to 48 inches, and the capacity varies with this dimension.

DUST-CHAMBERS.—The appliances for catching flue-dust in the Leadville smelting works are generally very imperfect, but the reproach does not apply to Leadville alone, for, however extensive the system employed elsewhere, it fails to accomplish its purpose completely. It is said that some English lead works have dust-chambers no less than 5 miles in length, and yet fail to recover all the dust carried from the furnaces; some of the Leadville works, however, make no attempt to collect the flue-dust, a practice unworthy of imitation. The ordinary provision consists of brick chambers on or below the charging-floor, either divided into sections by walls and curtains or not. One such chamber is 75 feet long, 25 feet wide, and 15 feet in height, and another of the same length is only 4 feet wide and 6 feet high. The dust-chambers are sometimes built of iron instead of brick, and the circuitous direction given to the current by the interposition of walls and curtains in brick chambers is then often obtained by the use of adjoining vertical cylinders, the air and fumes entering the bottom of one and the top of the next.

BLAST-ENGINES.—The blowing engines employed are most commonly of the Baker rotary pattern, though at one establishment the Root blower is in use. The pressure of the blast furnished by these blowers varies from half an inch to 1½ inches of mercury, or say from one-fourth to three-fourths of a pound per square inch, the most usual tension being 1 inch of mercury, or about half a pound per square inch. Where several blowers are employed in furnishing blasts to more than one furnace the pressure is equalized, and the probability of an interference with the work through the stoppage of a blower is decreased by connecting them all with the same main blast-pipe.

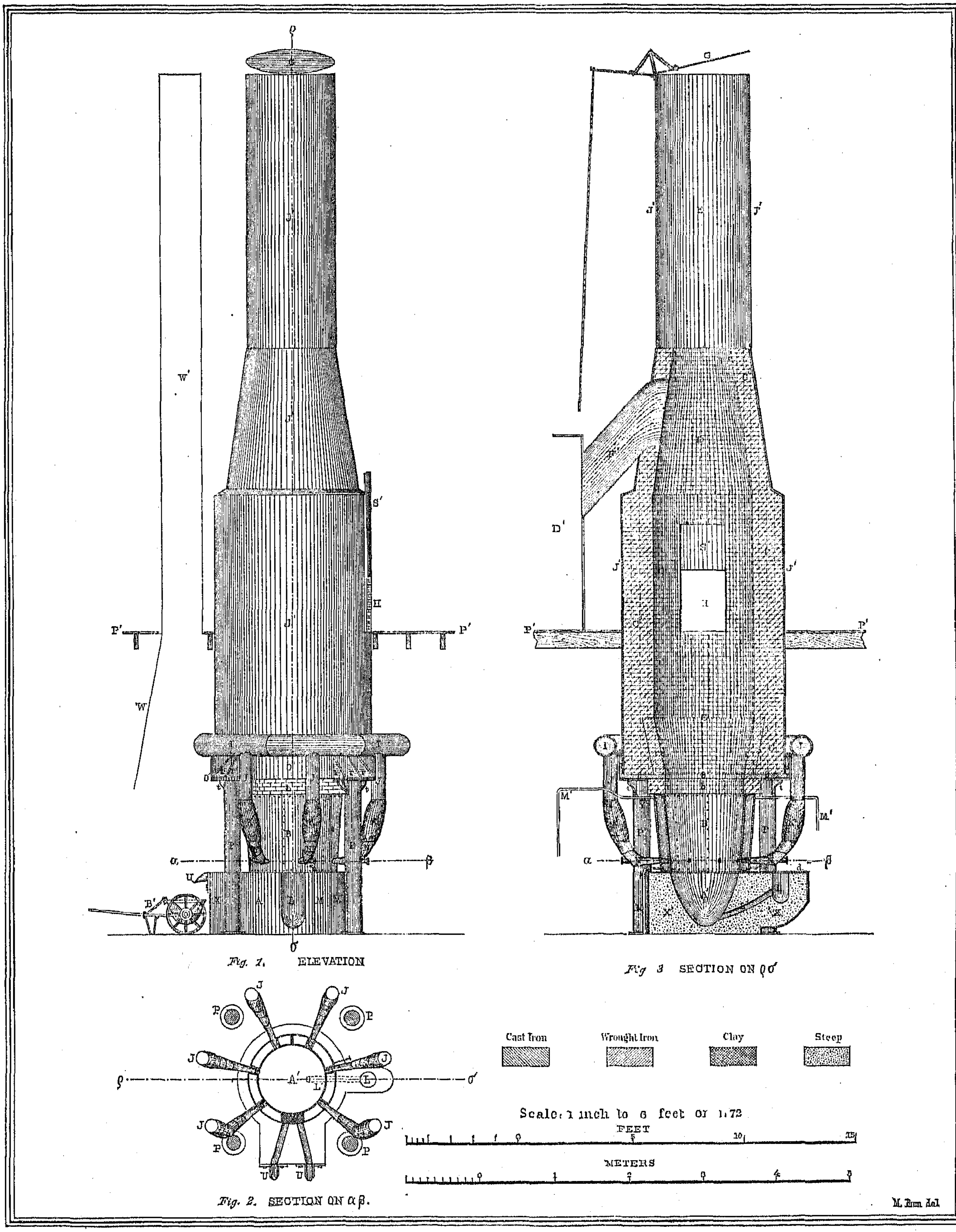
The iron work of the furnaces is sometimes made by Denver firms, but usually the entire plant is ordered from the East.

BARTLETT FILTER.—An experiment was made at one of the works with this arrangement for collecting flue-dust which gave some interesting results. The following is condensed from Mr. Guyard's description:

The stack of one of the square furnaces was connected with a Sturtevant fan by means of a sheet-iron flue, through which the fumes were drawn from the furnace and blown through a sheet-iron pipe 150 feet in length, which was connected, by means of two branch pipes, with two boxes of thin sheet iron. The dust was collected in the sheet-iron pipe as in an ordinary flue. Each branch pipe was provided with a damper, or valve, similar to those used in stovepipes, so that the fumes could be distributed to one or both of the boxes at pleasure. Each box consisted of a dust-chamber and a fireplace, the former being provided with sliding doors, placed at either extremity, and the fireplace with doors in front and sheet-iron pipes at the back, communicating with a stack. At the top of each of the dust-chambers were twenty-eight apertures, to each of which was fastened a cloth bag, 30 feet high, suspended to the beams of a light wooden structure, in which the apparatus was inclosed, and which was provided with very large openings for ventilation. When the apparatus is at work the fumes blown in distribute themselves in the dust-chambers and ascend the cloth bags, through which they are filtered. The gases come out perfectly colorless and free from any lead dust or even soot. The wind entering freely through the apertures of the building shakes the bags, and the dust with which they are charged falls back into the dust-chambers. When a sufficient quantity of this dust has been accumulated, the doors connecting with the fireplace are opened and a light wood fire is kindled. The soot soon catches fire and burns off, leaving the dust white. During a run of five days 3,030 pounds of calcined dust were caught in a Bartlett filter from one furnace. The experiment was not entirely satisfactory, owing to defects in the manner in which it was carried out; but the defect was one of arrangement, and by no means inherent in the filter. The furnace was worked without closing the feed-hole, as with an ordinary dust-chamber. The Sturtevant fan consequently drew in as much air as smoke, so that the chamber of the furnace had to be left half open, and about half the smoke escaped directly into the open air. The use of this arrangement was abandoned by the owners of the works partly on account of the expense involved and partly, as stated by them, on account of the large percentage of arsenic (15 to 20 per cent.) in the condensed matter and its low tenor in silver. As Mr. Guyard, in his analysis of this substance, found extremely little arsenic and much lead, chiefly combined with phosphoric acid, chlorine, and bromine, it is difficult to imagine on what ground the presence of arsenic in such quantity could be inferred. Mr. Guyard's analysis is supported by the fact that arsenic is present to but small extent throughout the district, while phosphoric acid exists in large quantities in many of the ores.

RAW MATERIAL.

ORES.—The ores of Leadville are remarkably pure argentiferous lead ores. They are locally divided into two general classes: the "sand carbonates", which are loose, sandy masses of carbonate of lead with chloride of silver, and the "hard carbonates", which are masses of porous siliceous material with a varying proportion of hydrated oxides of iron and manganese, carrying carbonates of lead and chlorides of silver, and sometimes containing a considerable proportion of unaltered argentiferous galena. As a rule, with the exception of mechanical mixtures of clay and varying proportions of iron and silica, they contain but few foreign ingredients. Intimately associated with the carbonates is generally a little pyromorphite or chloro-phosphate of lead, amounting in one exceptional case to 10 and in another to 30 per cent. of the whole. Sulphate of lead also occurs in small quantity, with small and variable amounts of oxidized compounds of copper, arsenic, antimony, and manganese. The latter is often abundant, and is associated with or replaces iron oxide. Ores which are rich in manganese are generally poor in silver. The galena is frequently covered by a coating of carbonate showing clearly the alteration of the sulphide, first to sulphate, and then to carbonate. In some few mines bismuth and vanadium ores have been found. But a small proportion of the ores smelted is furnished by districts outside of Leadville. Of this the greater part comes from Ten-Mile district, in Summit county, and especially from the Robinson mine, whose deposits carry much pyrite and zincblende. The silver in the oxidized ores is present in combination with chlorine, bromine, and iodine, either as chloride, chloro-bromide, or chloro-bromo-iodide, as the analyses on page 289 of specimens from several mines made in the laboratory of the United States geological survey at Denver show.



CIRCULAR FURNACE

	R. E. Lee mine.	Amie mine.	Big Pittsburgh mine.
Chloride of silver	21.589	15.755	99.965
Bromide of silver	77.986	84.091	None.
Iodide of silver	0.425	0.154	0.035

Large masses of chloride of silver, or horn-silver, have been found, and on several occasions tons of ore have been taken from the R. E. Lee mine assaying from 8,000 to 15,000 ounces of silver to the ton and almost entirely free from lead. According to Mr. Guyard, sulphide of silver is sometimes present in small quantity.

The Leadville ores in general contain little or no gold, its presence not being easily detected in the ore itself, but only being shown in the final product. The average daily output of the mines in 1880 is placed by Mr. Guyard at from 700 to 800 tons, and the total smelting capacity of the furnaces at 700 tons per diem.

FLUXES.—The fluxes used in Leadville are limestone and hematite. During the census year the limestone used was the blue dolomitic limestone (Lower Carboniferous), in which the Leadville ores occur, taken either from open quarries or from dead-work in some of the mines. In the latter case it often carried a small percentage of silver. Experiments showed that dolomite was a less favorable flux than pure carbonate of lime, and since the advent of the railroad limestone has been obtained from the beds of the Colorado Cretaceous formation at Cañon City, 117 miles distant, and more recently still from a bed in the Upper Coal Measures at Robinson, 16 miles distant by rail, where it costs $3 per ton f. o. b. Red hematite iron ore was at first exclusively used as a flux, being principally obtained from the Breece iron mine, where it occurs in large masses between the White and the Gray porphyry, and it is said to carry a small percentage of silver. More recently it has been the practice in many smelters to use the limonite which had collected on the dumps of the various mines, and which also carries a small percentage of silver. In many cases the ores themselves are so ferruginous that but little additional iron is required.

FUEL.—The fuels used are coke and charcoal. Previous to the advent of the railroad coke was scarce and dear, having to be brought 30 to 150 miles by ox or mule teams; hence charcoal was much more largely used than at present. This is furnished by the forests of spruce covering the neighboring mountain slopes. The charcoal produced from these woods varies greatly in quality, according to whether it has been burned in pits or in kilns. The pit-charcoal made in the neighborhood of Leadville is said to contain 2.5 per cent. of ash; a sample of kiln charcoal was found by Mr. Guyard to contain 1.62 per cent. of ash. One hundred and forty-two and one-half bushels of charcoal make 1 ton, the bushel weighing 14 pounds.

COKE.—The cokes used are brought from El Moro, in the southern part of the state, by the Denver and Rio Grande railroad, and from Como, in the South Park, by the Denver, South Park, and Pacific railroad. These cokes are made from coals of the Lignitic or Upper Cretaceous formation, and contain, according to determinations made at the smelting works of Messrs. Billings & Eilers, 22 per cent. of ash for the El Moro and 9½ per cent. of ash for the South Park coke. The composition of the ash of the El Moro coke is represented as being 84½ per cent. silica, 7.1 per cent. peroxide of iron, and 8.4 per cent. alumina, lime, etc. The ash of the South Park coke shows 29.1 per cent. silica, 47.8 per cent. peroxide of iron, and 23.1 per cent. alumina, lime, etc. About 40 pounds of coke make one bushel; hence, 50 bushels make one ton.

ORE BUYING.—Ore is purchased either directly from the mines themselves for cash or from sampling works, which either buy from the mines or act as their agents. Various considerations affect the price paid. From the assay value of the ore in silver a certain percentage is deducted for loss in smelting, which varies according to the nature of the ore, whether siliceous, ferruginous, or sulphureted, or according to a special arrangement made between the mine owners and the smelter owners. A further variable charge is made for cost of treatment, which is dependent on the nature of the ore and its tenor in lead. As a general rule, in regard to oxidized ores, the charge for treatment is lower the larger the percentage of lead they contain. When this tenor is between 5 and 30 per cent. the lead is paid for at from 15 to 45 cents a unit of 20 pounds; the higher the percentage of lead the higher the price paid per unit. When the ore contains less than a certain percentage of lead, which varies with the quality of the ore, the mine owner receives no remuneration for the lead contained in his ore, however rich it may be in silver. The following table gives a specimen of the rates charged for treatment of the ores of some of the best-known mines, the deduction made for loss of silver, and the price paid for each unit of lead above this certain percentage:

Name of mine.	Deduction for loss of silver in smelting.	Cost of treatment per ton of ore.	Price of lead per unit of 20 pounds.
	Per cent.		
Amie	10	$25	$0 25
Chrysolite	5	20	25
Dunkin	5	22	25
Carbonate	7½	20	25
Evening Star	7½	28	25
Morning Star	5	15	30
Iron	5	18	30
Tucson	5	21	25

These figures vary from month to month unless a time contract has been entered into, and are governed by the market quotations of silver and lead at New York and the prices of coke, charcoal, and fluxes at Leadville. Gold, when present in excess of one-tenth of an ounce to the ton, is paid for at the rate of $18 per ounce. The transportation of ore from the mines to the sampling or smelting works is paid for by the smelters at the rate of $1 to $1 85 per ton.

SAMPLING.—When the ore arrives at the sampling works, it is weighed in the wagon on scales generally occupying a detached building. It is then thrown into bins or piles in the open yard, every tenth shovelful as a rule being put into a wheelbarrow. The sample thus obtained is spread out on the sampling floor, and in the case of a sand ore is worked up directly to obtain a thorough mixture. Hard ores are first passed through Cornish rolls. When the ore is thoroughly mixed, it is repeatedly quartered till a sample convenient for drying has been obtained. After drying it is further crushed, mixed, and quartered, and a portion is then ground on the bucking plate by the bucker (*a*) until it passes through a sieve of 70 to 80 meshes to the linear inch. The sample is then divided into three portions, one of which is assayed at the smelter, and a second at the mine or by a public assayer who may be employed by the mine. If the results of the two assays agree closely, a mean is generally taken as the true value of the load; otherwise the third portion is sent for control to a third independent assayer. Sand ores require no crushing before charging into the furnace. For hard ores, slags, fluxes, etc., Blake, and occasionally Alden crushers, driven by steam-power, are employed.

SMELTING CHARGES.—The construction of ore-beds is carried on to a considerable extent at both smelting and sampling works. These beds average from 160 to 180 tons each in weight, and contain approximately equal parts (20 to 25 per cent.) of metallic lead, metallic iron, and silica, of which the proportion of lead is subject to the greatest fluctuation. The proportion of silver to lead is 1 ounce to about 6 or 8 pounds. Sulphureted ores are not roasted, but are thrown directly into the furnace, and are mixed in small quantities with the oxidized ores.

The charges vary so greatly in composition at the different smelters that it is hardly possible to give that of an average one. At first the aim seems to have been to produce a normal singulo-silicate slag, but a change has been gradually taking place to slags of a slightly more acid character, containing from 32 to 36 per cent. of silica. At one smelter the aim is said to have been to produce a slag in which the proportion of earthy base to metallic base should be as one to two, or to some even number. The following examples of different charges are taken from Mr. Guyard's report as specimens of their variable character:

	I.		II.		III.		IV.		V.	
	Pounds.		*Pounds.*		*Pounds.*		*Pounds.*		*Pounds.*	
ORE:										
Ore-bed mixture	100		200		123		500		.....	
Unmixed ore	50		300		183		200		645	
Lead scraps	......	150	10	510	......	306	......	700	......	645
FLUX:										
Dolomite	10		50		90		80		62	
Hematite	10		......		7		170		......	
Slag	30	50	150	200	60	157	80	330	80	142
FUEL:										
Charcoal	15		80		50		95		80	
Coke	20	35	60	140	50	100	65	160	60	140
Total weight of charge		235		850		563		1,190		927

The proportions of charcoal and coke in the fuel vary, according to supply and cost and from other considerations, within the limits of three parts of the one to four of the other.

The table on page 291 gives the calculations made by Mr. Guyard from data obtained for the census year in eight of the principal smelting works of Leadville: First, the average proportion of flux to 100 parts of ore; second, the proportion of fuel to 100 parts of ore; third, the proportion of fuel to 100 parts of smelting charge. In the fourth rubric is given the number of tons smelted per twenty-four hours in each of these works. From these data he calculates the relation of actual to nominal smelting capacity as varying from 26 to 80 per cent. Furnace III is regarded as fulfilling most nearly theoretically perfect conditions.

a The bucking plate in ordinary use in assay offices in the West is a cast-iron plate measuring 2 by 1½ feet, with flanges on the long side rising half an inch above the surface; the latter is planed down, but not polished. The bucker or rubber is a rectangular piece of cast iron 7 by 5 inches, and from 1 to 4½ inches thick. On the upper surface is a socket for a long wooden handle, and the lower surface is curved (a portion of a large cylindrical surface) so that, as the operator pushes it to and fro on the plate to pulverize the ore, a slight rocking motion may be given at the same time, which brings the particles under the bucker instead of pushing them before it.

	I.	II.	III.	IV.	V.	VI.	VII.	VIII.
Proportion of flux to 100 parts ore	25.00	39.0	42.0	24.0	28.8	46.6	33.5	31.70
Proportion of fuel to 100 parts ore	45.33	33.0	33.8	18.5	25.0	40.2	25.0	25.54
Proportion of fuel to 100 parts charges	36.00	23.5	23.7	15.0	19.3	31.5	18.5	18.60
Tons of ore smelted per twenty-four hours	28.00	104.0	51.0	11.5	23.0	15.9	69.5	32.50

The proportions obtained by Mr. Guyard as an average for the entire camp during the census year are:

32.83 parts fuel to 100 parts ore.

23.65 parts fuel to 100 parts charges.

He calculates that 88 per cent. of the lead in the ore is extracted as bullion by direct smelting, the remainder going into the slag and escaping up the stack; also that 1½ parts of fuel are required for each unit by weight of bullion produced.

In the following table Mr. Guyard has calculated with regard to the same furnaces shown in the preceding table: First, percentage of lead extracted in smelting in the form of bullion; second, the percentage of silver extracted; third, the average charges for smelting per ton of ore at each establishment; fourth, the cost to the smelter of treating each ton of ore; fifth, the average assay of slags in ounces of silver per ton; sixth, the average assay of flue-dust in ounces of silver per ton:

	I.	II.	III.	IV.	V.	VI.	VII.	VIII.
Percentage of lead extracted in smelting	85 to 88	86 to 91	88.0	85 to 95	85 to 90	90 to 93	87.0	90.0
Percentage of silver extracted in smelting	100	95 to 97	97.0	88 to 95	95.0	97.0	98.5	97.5
Charges for smelting per ton of ore	$15 to $30	$15 to $30	$15 to $30	$12 to $25	$16 to $30	$15 to $30	$15 to $30	$15 to $30
Cost for smelting per ton of ore	$12 to $18	$18 to $23	$10 to $15	$13 to $16	$15 to $18	$13	$13 68	$15
Average assay of slags in ounces of silver	2	4	0.5	1.5	1.5	1.5	1.5	4.0
Average assay of flue-dust in ounces of silver	36	37	36.0	35.0	35.0	36.0	36.0	37.0

The above table shows only the conditions which obtained during the year ending June 1, 1880. These have been varied since that time by a general reduction of the smelting charges, owing to competition and to the cheapening of supplies, and also by the reduced tenor of the ore in silver. The proportion of sulphureted over oxidized ores will also probably increase as time goes on.

GENERAL SMELTING OPERATIONS.

BLOWING-IN OF FURNACES.—The furnace is first dried by means of a slow charcoal or wood fire, whose temperature increases gradually for several days. When the drying is completed, the fire is allowed to burn out and the furnace left to cool. The crucible is then lined with steep or brasque; tamping, a simple lining of fire-clay, is sometimes put upon the dam, siphon, and siphon-tap. The furnace is then filled to the feed-hole with charcoal, the tuyere-holes, tympstone, and stack-damper being left open to create a draught. The charcoal gradually becomes incandescent to the very throat. When this zone has reached a low red heat the blowing-in begins. The tuyere holes of the water-jacket, with the exception of from two to four of those nearest the front, are sealed with plugs of clay, and the wind-bags of the corresponding tuyeres are tied up with strings. Tuyeres are inserted in the holes left open; the tympstone is set in, and the blast then turned on at full pressure. A long flame issues from the siphon-tap, and the fire is kept steadily up until the lead-well becomes red hot. The remaining tuyere-holes are then opened and all the tuyeres are set in. The blast is regulated to the normal pressure, and the furnace is now ready for the filling of the crucible.

FILLING OF THE CRUCIBLE.—Bars of bullion kept in reserve for this purpose, in amount varying from 4 to 12 tons, according to capacity, are thrown in at the feed-door with more fuel, the proportion being three bars of bullion (300 pounds) to eight shovels of charcoal, or about 14 per cent. of fuel. From 100 to 150 bushels of charcoal, according to the capacity of the furnace, are consumed during the blowing-in. When molten lead makes its appearance in the lead-well, a few pieces of live charcoal are placed on it to prevent it from cooling, and the furnace is ready for charging.

CHARGING OF THE FURNACE.—Old slags are first of all thrown into the furnace as a test of its temperature, which is not ready until the slag is perfectly fluid. The head smelter or his assistant opens the tap-hole in the tympstone from time to time to observe their degree of fluidity, and the regular charging begins only when they run quite freely. The charges are disposed on the inside of the furnace next to the walls, a depression being left in the center for the fuel. This is the mode of charging generally adopted, but there are variations in the manner of mixing the materials forming the smelting charges. At some of the smelting works fuel is first thrown in, then old slags, above these the fluxes, and then the ore; at others fuel is mixed with the old slags and fluxes with the ores. The mode of proceeding generally adopted, however, is to mix slags, fluxes, and ore together and keep the fuel separate. At the most successful establishment the method of mixing fuels and old slags on the one hand and fluxes and ore on the other prevails. In either case the distribution of the materials in the furnace is the same; *i. e.*, fuel is thrown in the center and the charge is distributed around it.

TAPPING OF SLAGS.—The tapping of slag commences as soon as the furnace is in regular operation, and occurs generally every fifteen or twenty minutes, although a few works have adopted the method of continuous flow from the tap-hole. The tap-hole is closed by a lump of clay at the end of an iron tapping-rod. The slag is caught directly in a slag-buggy or conical cast-iron pot mounted on wheels, and is either allowed to solidify entirely in the pot and then thrown out and broken up, or it is wheeled to the edge of the slag heap and tipped over so that it runs in a half liquid state down the sides of the dump. A single smelter adopts the plan of allowing the slag to solidify partially in the pot, and of then making holes through the hardened crust and tipping the pot over so that the still molten material runs out, leaving a shell over 2 inches thick. This shell is easily broken up for re-smelting, it having been found that it is a little richer in silver than the center. Slag samples for assay are taken two or three times a day from the stream in the slag gutter, and their specific gravity and contents in lead and silver are determined in the assay office. Any speiss or matte that may be formed is run into the slag-pot with the slag, and is either thrown out with the latter upon the dump, or, after cooling, is detached from the slag and preserved separately. The proportion of speiss and matte at Leadville is generally very small.

LADLING OF BULLION.—As often as necessary the bullion is dipped out of the lead-wells with wrought-iron ladles and poured into cast-iron molds. At a single smelter a different method obtains. The bullion is tapped periodically from an opening in the clay lining of the lead-well into an iron pot mounted on a small stove, in which a light fire is kept burning. From this pot it is ladled into the molds. The advantages of this method are that the surface of the lead in the well is kept covered. The lead is therefore hotter, and the passage into the crucible is more readily cleaned. Moreover, the lead being drawn from below, the surface is free from skimmings, and the bars are smoother and cleaner.

The bars of bullion are then sampled, weighed, and marked. The sampling is done by taking with a scoop-chisel a piece from the top and a piece from the bottom of each bar. The samples from 200 bars, or 10 tons, which constitute a car-load, are sent to the assay office, where they are melted together and cast into a small bar, from which pieces are taken for an assay.

CONTROL OF SMELTING OPERATIONS.—From time to time the siphon-tap has to be cleared by the insertion of a curved iron rod, about 2 inches thick, previously heated to redness at the curved end.

The tuyeres must be watched from the sliding valve, and when dark rings of chilled slag are observed around them they are removed by iron rods inserted through the tuyere, and the temperature is raised by the addition of more fuel or by a reduction in the proportion of charge.

The water-jackets require constant watching, in order that the temperature of the water issuing from them may be kept as nearly as possible at from 50° to 60° C.

The blast also requires constant attention and regulating, the pressure being increased or diminished as the condition of the furnace, determined by observation from the tuyeres, may demand.

If semi-fluid slags or raw ore form accretions, which do not disappear by an increase of the temperature, the blast must be shut off, the tympstone removed, and the hearth cleaned by means of bars and sledges; after which a little fuel is thrown into the hearth, the tympstone is replaced, and the blast is turned on again. At one period ores rich in lead were scarce at Leadville, and the charges generally contained much less than the normal 20 to 25 per cent. of lead. The running of the furnace became, in consequence, a much more difficult matter, and the formation of obstructions of various kinds was of frequent occurrence.

When accretions form on the walls of the shaft, it is necessary to "bar it out" once in twenty-four hours, or once per shift, as the case may be. To accomplish this charging is interrupted until the contents of the furnace descend to the level of the accretion. The blast is then turned off, a long chisel-pointed bar is introduced into the feeding-hole of the furnace, and, being inserted between the accretion and the furnace wall, is struck with sledge-hammers until the accretion is detached, when the blast is turned on again and the charging resumed.

The Leadville furnaces are generally run with a dark top; *i. e.*, the zone at the throat is so dark that no flames issue from it, and only a black smoke is seen ascending the chimney. This appearance is an indication that the furnace is running properly.

SMELTING OF FLUE- AND CHAMBER-DUST.—Flue- and chamber-dust are mixed in general with lime, and the mixture may or may not be molded into bricks. It is then spread over the ore beds, so that a little of the flue-dust enters into the composition of the smelting charges.

BLOWING-OUT OF FURNACE.—This takes place when the furnace needs repairing, or when an accident, interfering with the regular working of the furnace, has occurred. It is done by suspending the charges and continuing the blast until the whole contents of the furnace are molten. The charge soon burns with a bright top, and the furnace emits torrents of heavy white fumes. When the whole charge has reached the level of the tuyeres the furnace is emptied of its fluid contents, first from the tap-hole, then the breast is removed, and the bullion is taken out of the crucible.

LENGTH OF RUN.—The smelting campaigns are seldom less than three weeks, and often reach six, eight, and even thirteen months.

FURNACE PRODUCTS.

BULLION.—The bullion of Leadville is generally very pure, its constituents other than lead and silver, though numerous, being present in very small quantity. The character of these impurities is shown in the following analyses made by Mr. Guyard, I being bullion from the La Plata smelter, II being a mixture of equal parts of bullion from nine different smelters:

	I.	II.
Lead (by difference)	99.0708210	98.402379
Silver	0.6112445	0.793417
Gold	0.0000888	0.000801
Copper	0.0470100	0.071450
Tin	A faint trace.	0.000807
Bismuth	A faint trace.	0.011791
Arsenic	0.0301365	0.219528
Antimony	0.2138940	0.347881
Iron	0.0063000	0.012600
Zinc	0.0016052	0.000232
Cadmium	A faint trace.	A faint trace.
Sulphur	None.	0.048934
	100	100
Ounces of silver to the ton	178.275	231.408
Ounces of gold to the ton	0.026	0.260

The presence of tin in the bullion seems rather singular, inasmuch as it has not been detected in any of the ores or fluxes of Leadville. It has been suggested that it owed its origin to the great number of preserved-fruit cans scattered about the place, some of which may have found their way into the furnace. It is indeed said that these cans were at one time used at one smelter, probably as a precipitant for the lead in the galena. As tin has also been found in other products at different times, it seems hardly probable that this source can be adopted as that from which it is in all cases derived.

The average assay of bullion shipped from Leadville during the early part of the census year was nearly 300 ounces to the ton, but during the month of December, 1880, it had fallen off to less than 200 ounces. Mr. Guyard estimates the average loss of silver in smelting at 4.115 per cent., and of lead at 11.68 per cent., part of which, however, is recovered from the chamber- and flue-dust. The bars of bullion weigh on an average 100 pounds each, 200 bars, or 10 tons, constituting a car-load. They are shipped to eastern refineries, and when sold direct the latter pay the cost of transportation, which varies from $27 to $35 per ton. The price of lead in bullion is subject to great fluctuation, and has varied between $30 and $78 per ton at Leadville, the average price being $60. Payments are made for bullion at New York quotations, deducting for the cost of refining 3 cents per ounce of silver, or sometimes $14 to $15 per ton of bullion. In other cases the charges are 3 ounces of silver and 5 per cent. of lead per ton.

SLAG.—The slags produced at Leadville are in some instances normal singulo-silicates, but in general rather more acid, the object in producing the latter being to insure a smoother run of the furnace, to require less constant watching and to avoid the formation of sows and accretions. They flow freely, and generally possess, when cold, a compact, fine-grained structure, though frequently well crystallized in parts. They are for the most part strongly magnetic; and Mr. Guyard has shown that this property is not due to any magnetic silicate of iron, but to magnetic oxide of iron, he having isolated these substances in a greater or less quantity and in a state of perfect purity from all slags investigated by him. As a means of judging whether the slag is normal in its composition and contains any excess of lead determinations are made daily at a few of the smelters by means of the Jolly spring-balance. Slags from a normal run carry from 2 to 4 per cent. of lead and from 2 to 4 ounces of silver to the ton, though by attention and careful charging these figures are sometimes greatly lowered. On the other hand, they frequently run much higher, both in lead and in silver, owing to the faulty composition of the charges or to careless regulation of the smelting process. Slags from the earlier campaigns of some furnaces have been found to contain as much as 15 to 20 per cent. of lead, and silver in proportion; in view of which it is hardly a matter of wonder that failures were frequent. Slags accidentally rich in silver, and whole slag heaps from some of the works first started are re-smelted with the ores.

MATTES.—Since the sulphide ores occurring in the region are not roasted, but thrown directly into the furnace, mixed in small proportion with the oxidized ores, a certain amount of matte is necessarily formed, which consists mainly of sulphides of iron and lead, with, as ascertained by Mr. Guyard, a large percentage of magnetic oxide of iron. These mattes carry from 40 to 90 ounces of silver to the ton, and are roasted in heaps preparatory to being re smelted.

SPEISS.—Speiss, which is found only in small quantity, is an iron sulph-arsenide, and seems to concentrate in itself all the molybdenum as well as most of the nickel, traces of which exist in the ores. It is further characterized by its very small percentage of antimony, and, according to Mr. Guyard, by the total absence of cobalt. It is a question, however, whether an examination of a large number of samples would justify the conclusion drawn by Mr. Guyard that a complete separation of nickel and cobalt is effected in the smelting process. He found cobalt without nickel in the skimmings from the lead-wells and nickel without cobalt in the speiss.

Its silver contents vary from 2 to 4 ounces per ton. It is not roasted or subjected to any further treatment.

ACCIDENTAL FORMATIONS IN THE FURNACE.—Among these iron sows are the most important, and have frequently been the source of much trouble, sometimes necessitating, as before mentioned, the blowing-out of a furnace in order to effect their removal. Besides ordinary hearth obstructions, different shaft accretions are found in varying quantity, which it is unnecessary here to discuss further, save to quote an instance, mentioned by Mr. Guyard, of a small, round furnace entirely lined from the water-jackets to within 6 inches of the feed-hole with one of these accretions a foot in thickness.

FLUE- AND CHAMBER-DUST.—Leadville being situated at an elevation of 10,000 feet above the sea, the volume of air blown into a furnace with a given blast-pressure is far greater than with the same pressure at sea-level; consequently the draught of the furnace is correspondingly increased, and the quantity of dust and fumes escaping by the stack is very large. This would seem to necessitate the employment of a very perfect system of condensing flues and chambers. In point of fact, however, as has been seen, these arrangements are, with one or two exceptions, very poor; consequently a large proportion of the dust and fumes is lost in the air. Their composition is extremely complicated, and is characterized by the presence, in considerable quantities, of chlorides, bromides, iodides, and phosphates. They carry from 25 to 60 per cent. of lead, the latter figure applying to the fumes condensed in the Bartlett filter before described, and from 30 to 40 ounces of silver to the ton, although the Bartlett filter fumes held but 4.3 ounces. The composition of the latter is otherwise remarkable in that they contain over 11 per cent. of phosphate of lead, 9 per cent. of chloro-bromo-iodide of lead, and 18 per cent. of sulphide of lead, and from the further fact, according to Mr. Guyard, that iron, zinc, and manganese exist in them entirely in the state of sulphides. It must be borne in mind, however, that these fumes were condensed at a distance of 200 feet from the furnace. Mr. Guyard has calculated the weight of calcined dust collected from one furnace of 30 to 40 tons capacity during twenty-four hours at 1,400 pounds, and estimates that, where the filter is not employed, the loss of lead equals half a ton and of silver 4½ ounces per day per furnace. On this assumption more lead is lost in the air than is collected in the dust-chambers. As already shown, the chamber- and flue-dust is mixed with lime and thrown over the ore-beds to be re-smelted. In one case, however, a furnace was specially constructed for the purpose of roasting the flue-dust, though with what object in view it is impossible to say, unless on the erroneous supposition that much arsenic is present. In point of fact, arsenic is found in it in only small quantities, and this roasting deprives the dust of the carbon which would otherwise suffice for the reduction of all the lead contained in it, besides occasioning the loss of some silver. In another smelter the flue-dust is prepared for re-smelting by melting down in a reverberatory furnace, at first with, and now without, the addition of slag. It is then run out, and after cooling it is broken up and mixed with the charges.

COST OF LABOR AND MATERIAL.

Below are given a few data relative to the prices paid for fluxes, fuel, and the average wages of employés during the census year at Leadville, which serve to give a fair idea of the economic conditions of smelting at that time:

Dolomite, per ton	$3 00 to $4 00
Hematite, per ton	8 00 to 11 50
Charcoal, per bushel	10 to 18
Coke, per ton	25 00 to 60 00
Pine wood, per cord of 2,000 to 3,000 pounds	4 50 to 5 00
Foremen, per shift of eight to twelve hours	3 00 to 6 00
Head smelters, per shift of eight to twelve hours	3 00 to 4 25
Slag wheelers, per shift of eight to twelve hours	2 50 to 4 00
Feeders, per shift of eight to twelve hours	3 00 to 4 00
Helpers, per shift of twelve hours	2 50 to 3 00
Day laborers, per shift of ten to twelve hours	2 50
Engineers, per shift of eight to twelve hours	3 50
Fuelmen, per shift of eight to twelve hours	3 00

CONCLUSION.—In conclusion, it may be said that lead smelting, as carried on in this region, while not entirely beyond criticism, has been brought to a relatively high degree of perfection, and is extremely creditable to American metallurgists. One of the most useful practical lessons that has been taught by the comparative success of the various smelting works is that this has been proportional to the more thorough training in scientific metallurgy of its managers, the completeness and accuracy with which they have gauged the operations of their furnaces by chemical tests, and the intelligence with which the results of these tests have been applied to the practical conduct of their business.

Could this lesson overcome the idea so common among us that the adaptability and "cuteness" of the American, which is in so many points acknowledged to be superior to that of other races, enable him to master the science of smelting as readily as he does any branch of trade, it might prevent an increase of the already very considerable number of abandoned smelters which dot our western hills and valleys, and save a portion of the capital which is annually wasted through gross ignorance in the various operations connected with mining.