remained in the tailings. While the gold obtained in our stamp-mill and amalgamating works is seldom purer than 897 fine, the gold obtained by this process is generally over 978 fine.

THE RUSSELL PROCESS IN ITS PRACTICAL APPLICATION AND ECONOMIC RESULTS.

COMPILED FROM MR. RUSSELL'S NOTES BY ELLSWORTH DAGGETT, SALT LAKE CITY, UTAH.

(Boston Meeting, February, 1888.)

INTRODUCTION.

In the first paper on the Russell process presented by Mr. Stetefeldt, in May, 1884 (*Transactions*, xiii.), the process was treated from a purely theoretical standpoint. In his second paper of October, 1886 (*Transactions*, xv.), the details of the plant, the chemicals used, and the manipulation in both mill and laboratory were treated. The aim of the present paper is to present economical results recently attained, new data affecting the preparation and manipulation of ore in the mill, and a more complete and systematic scheme of laboratory work.

This paper is compiled mainly from Mr. Russell's notes of his own experience. Many of these notes, particularly those relating to work at Cusihuiriachic,* Mexico, have been upon work directly under the personal observation of the compiler, who takes pleasure in thus presenting to the Institute, data and considerations of much interest and value not only to users of the Russell process, but to mill men generally.

The description of the chemicals used and of the preparation of the mill-solutions being in part the same now as when Mr. Stetefeldt's second paper was published, some use is here made of portions of that paper.

No theoretical matter is here given. For a treatise containing also the theory of the process the reader is referred to Mr. Stetefeldt's complete work on "Lixiviation of Silver Ores by Hyposulphite Solutions," now in preparation.

To facilitate the consultation of this paper, the following synopsis and index is prefixed. The figures in parenthesis indicate the page of this paper on which the subject is treated.

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* Called "for short," in this paper, as in common usage among mining men, "Cusi."

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I. Laboratory Work.

A.—*Leaching-Tests by Extra-Solution or Russell Process.*

Different ores require different treatment on the small scale, in the assay-office, as well as in the mill. Consequently, for the reasons explained further on, eight different leaching-tests are used in testing raw ores, and eight for roasted or chloridized ores. Of the eight methods, the one giving the highest per cent. extraction is then selected for all further leaching-tests on that particular ore.

In making the tests the following points should be noticed:

The metric system and assay-ton weights are supposed to be used. The weight of ore taken is $\frac{1}{3}$ or $\frac{1}{2}$ an assay-ton, and is placed in a beaker of about 300 c.c. capacity.

If the fire-assays are made by scorification, of course a smaller amount of ore should be taken, but no change need be made in the amount of chemicals or solution used. Also, if beakers of 300 c.c. are not at hand, smaller sizes and less volume of solution can be used, without altering the weight of chemicals. The given proportion between hyposulphite and blue-stone should be observed.

The preliminary washing with water in the case of roasted or chloridized ores is by decantation, the clear liquid being decanted upon the filter that is afterwards used to receive the ore.

The amount of hyposulphite of soda used is 20 grammes, unless otherwise stated.

The "blue-stone solution" referred to is a 20 per cent. solution made by dissolving 200 grammes blue-stone in hot water and diluting to 1.000 c.c. Unless otherwise stated, 25 c.c. of this solution is used.

In all cases the contents of the beakers should be occasionally stirred.

The washing with water at the end of the operation is to remove the last traces not only of dissolved silver, but also of hyposulphite of soda, which would interfere with the burning of the filter. After the final washing with water, the filter and ore are transferred to a flat dish and dried. When dry, the ore is separated from the filter and the filter burned—the ash being added to the ore, which is then ready to be assayed. If the fire-assays are made by scorification, the presence of the filter-ash is inconvenient. For such assays, the last traces of the ore are removed from the dry filter with a stiff brush, and the filter-paper is not assayed.

The following are the methods of making lixiviation-tests by the extra-solution (Russell process) in the assay-office. The first eight are for raw ores and tailings—the second eight for roasted or chloridized ores. For brevity the terms " Hypo" and " Blue " are here used for the salt hyposulphite of soda and the blue-stone solution respectively.

a. Leaching-Tests for Raw Ores and for Tailings.—1. Add 250 c. c. cold water and 20 grm. Hypo; let stand 12 to 16 hours cold; decant; add 25 c.c. Blue; dilute to 300 c.c. with cold water; let stand 12 to 16 hours cold; add 20 grm. Hypo; let stand 12 to 16 hours cold; filter and wash.

2. Add 250 c.c. cold water and 20 grm. Hypo; let stand 12 to 16 hours cold; add 20 grm. Hypo and 25 c.c. Blue; let stand 12 to 16 hours cold; filter and wash.

3. Add 100 c.c. cold water and 25 c.c. Blue; let stand 12 to 16 hours cold; dilute to 300 c.c. with cold water; add 20 grm. Hypo; let stand 12 to 16 hours cold; filter and wash on filter with hot Hypo solution.

4. Add 100 c.c. cold water and 25 c.c. Blue; let stand 1 hour cold; add 200 c.c. cold water and 20 grm. Hypo; heat to about 130° F.; filter and wash.

5. Add 250 c.c. cold water and 25 c.c. Blue; let stand 1 hour cold; add 20 grm. Hypo; heat on sand-bath to about 130° F.; decant; add 200 c. c. cold water and 25 c. c. Blue; let stand 1 hour cold; add 20 grm. Hypo; heat, filter and wash.

6. Add 200 c. c. cold water and 90 c. c. Blue; let stand one-half hour cold; add 90 grm. Hypo; heat to about 150° F. and wash.

7. Add 250 c.c. cold water and 25 c.c. Blue; heat to about 130° F.; decant; add 20 grm. Hypo and 25 c.c. Blue; dilute to 300 c. c. with cold water; heat to about 130 ° F.; filter and wash.

8. Add 200 c. c. hot water and 90 c. c. Blue; let stand one-half hour; add 90 grm. Hypo; heat to about 150° F.; filter and wash.

b. Leaching-Tests for Roasted or Chloridized Ores, Mattes and Tailings.—1. No washing; add 50 c.c. cold water and 25 c.c. Blue; let stand 2 hours cold; dilute to 300 c. c. with cold water; add 20 grm. Hypo; let stand 2 hours cold; filter and wash.

2. No washing; add 250 c. c. cold water and 25 c. c. Blue; let stand 12 to 18 hours cold; add 20 grm. Hypo; let stand 12 to 18 hours cold; filter and wash.

3. No washing; add 250 c. c. cold water and 25 c. c. Blue; let stand 2 hours cold; add 20 grm. Hypo; let stand 2 hours cold; filter and wash.

4. Wash with cold water; add 25 c.c. Blue; dilute to 300 c.c. with cold water; let stand 12 to 18 hours cold; add 20 grm. Hypo; let stand 12 to 18 hours cold; filter and wash on filter with hot Hypo solution.

5. No washing; add 250 c.c. cold water and 25 c.c. Blue; let stand 1 hour cold; add 20 grm. Hypo; heat to about 130° F.; decant; add 250 c.c. cold water and 25 c.c. Blue; let stand 1 hour cold; add 20 grm. Hypo; heat to about 130° F.; filter and wash.

6. Wash with hot water; add 200 c.c. hot water and 90 c.c. Blue; let stand one-half hour; add 90 grm. Hypo; heat to about 150° F.; filter and wash.

7. Wash with hot water; add 90 c.c. Blue; let stand one-half hour; add 90 grm. Hypo; dilute to 300 c.c.; heat to about 150° F.; filter and wash.

8. Wash with hot water; add 20 grm. Hypo; let stand (with residue of wash water, about 30 to 40 c.c.) 1 hour; add 25 c.c. Blue; dilute to 300 c.c. with cold water; heat to about 130° F.; filter and wash.

B.—*Leaching-Tests by Ordinary Solution.*

In the treatment of roasted ores and sometimes of raw ores, assay-office leaching-tests by the "ordinary" solution also, as well as by the "extra," should frequently be made. If these tests are made by the mill-solution, as is done in some mills, the results may be very misleading. For instance, if caustic soda or lime were present in the mill-solution, such a solution on many ores would give very low results, both in assay-office and mill. The low extraction would then be laid to a change in the character of the ore or to the furnace-work, whereas the true cause of the difficulty would be the impurity in the solution. By the use of a fresh hyposulphite solution in the assay-office test, the cause of the low results in the mill would become immediately apparent.

In the case of roasted or chloridized ores which have been so coarsely crushed in the mill as to require for assaying a re-crushing in the assay-office, a metal mortar should never be used, as may be seen from Table I. As there shown, the per cent. of extraction by the ordinary solution in the assay-office, on Cusi ore, is reduced 10 to 11 per cent. by the use of the iron mortar, while the results by the extra-solution are practically unaltered. The cause of this is probably the reduction of part of the silver to the metallic state, in which state it is much more soluble in the extra-solution than in the ordinary. But on the Lake Valley ores, the use of the iron mortar reduced not only the extraction by the ordinary solution to the extent of 25.3 per cent., but also the extraction by the extra-solution 18 per cent. No explanation can be given of the change

TABLE I.

EXPERIMENTS ON THE RESULTS OF GRINDING ROASTED ORE IN WEDGEWOOD OR AGATE, AND IN IRON MORTARS.

Name of Ore.	Mesh of Screen and Method of Crushing.	Kind of Sample.	Material of Mortar.	Mesh of Screen Used in Assay Office.	Value of Ore.	Value of Tailings from Ordinary Solution.	Value of Tailings from Extra-Solution.	Per cent. extracted by Ordinary.	Per cent. by Extra.
					Ounces per ton.				
Cusi.	20-Screen. Stamps.	Sample from Furnace	Wedgewood.	40	30.2	6.4	2.8	78.9	90.7
		Sample from Furnace	Iron.	40	30.2	9.6	2.8	68.3	90.7
		Sample of Tub Charges	Agate.	40	30.0	4.4	2.6	85.3	91.3
		Sample of Tub Charges	Iron.	40	30.0	7.4	2.8	75.4	90.3
Lake Valley.	16-Screen. Rolls.	Sample from Furnace	Agate.	40	16.0	2.9	2.0	81.3	87.5
		Sample from Furnace	Iron.	40	16.0	6.7	6.0	58.1	62.5
		Sample from Floor	Agate.	40	18.0	3.7	3.5	79.4	80.6
		" " "	Iron.	40	18.0	7.2	6.0	60.0	66.7
		Sample of Tub Charges	Agate.	40	18.0	3.0	3.0	83.3	83.3
		Sample of Tub Charges	Iron.	40	18.0	9.0	5.7	50.0	68.3

TABLE II.

EFFECT OF USING A COPPER VESSEL ON THE LEACHING-TEST BY THE ORDINARY SOLUTION IN THE ASSAY-OFFICE.

Roasted Ore.	Value in Ounces per Ton.		Ounces in Favor of Copper Vessel.	Per Cent. in Favor of Copper Vessel.
	Tailings from Solution heated in Copper Vessel.	Tailings from Solution heated in Wooden Vessel.		
73 80	20.52	24.84	4.32	5.8
74.76	8.88	10.32	1.44	2.0
77.46	10.92	15.24	4.32	5.6
60.00	13.80	20.16	6.36	10.4
72.00	15.30	19.56	4.26	5.9
75.60	9.84	11.52	1.68	2 2
76.32	9·84	12.90	3.06	4.1
63.96	14.94	16.82	1.88	3.0
69.30	15.96	18.60	2.64	3.8
72.78	8.40	9.12	1.02	5.4
72.72	8.28	10.98	2.11	3.7
71.70	12.42	15.46	3.0	4.7

taking place in this case. On some roasted ores the percentage of extraction by the ordinary solution in the assay-office is reduced 50 to 60 per cent. by the use of a metal mortar.

As observed by Mr. Frank Johnson, of the Yedras mill, in making leaching-tests by the ordinary solution, the solution should never be heated in a copper vessel. Table II. shows the difference, in the results by the ordinary leaching-tests in the assay-office, caused by heating the ordinary solution in a copper vessel, as compared with a glass or wooden one. This difference is caused by the dissolving of a very small portion of the copper vessel by the hot ordinary solution, thus making a very weak extra-solution.

C.—*The Best Assay-Office Leaching-Methods by the Extra-Solution for Various Roasted Ores, and the Results Obtained.*

Table III. gives the results obtained by assay-office leaching-tests by extra-solution on average samples of 18 roasted ores, of which 4 were roasted in Stetefeldt, six in Howell, three in reverberatory, and 5 in Brückner furnaces. The table gives also for each ore the two best methods of leaching-tests by extra-solution in the assay-office.

TABLE III.

METHODS USED, AND RESULTS OF LEACHING-TESTS BY EXTRA-SOLUTION, IN ASSAY-OFFICE, ON ORES ROASTED IN STETEFELDT, HOWELL, REVERBERATORY AND BRÜCKNER FURNACES.

Name of Mine.	How Crushed.	Mesh of Screen.	Per cent. of Salt Used.	Kind of Furnace.	Per cent. of Silver Extracted.		Leaching Methods Most Applicable.
Ontario, Park City	Stamps.	16	12	Stetefeldt.	97.1	95.5 Per ct.	6 or 7
Daly, " "	"	26	11	"	95.2		2 or 1
Lexington, Butte	"	30	15	"	93.2		8 or 5
Manhattan	"	40	6	"	96.7		1 or 3
Alice, Butte	"	26	15	Howell.	93.5	91.3 Per cent.	7 or 1
Sierra Grande, Lake Valley	Rolls.	16	7	"	84.6		5 or 7
Pyramid	Stamps.	30	....	"	93.6		7 or 6
San Antonio, Chihuahua	"	12	10	"	91.0		1 or 8
San Miguel, Chihuahua	"	12	8	"	93.6		1 or 3
Chloride, New Mexico	Rolls.	30	7	"	91.6		
Yedras, Sinaloa	Stamps.	26	7	Reverberatory.	90.5	90.7 Pr ct.	4 or 2
Sombrerete, Zacatecas	Rolls.	20	10	"	94.9		5 or 3
Carmen, "	"	20	0	"	86.7		3 or 4
Veta Grande, Chihuahua	Stamps.	16	6	Brückner.	87.8	88.6 Per cent.	3 or 1
Custer, Idaho	"	30	5	"	95.1		1 or 2
Bertrand, Nevada	Rolls.	12	6	"	83.9		5 or 8
Tybo, Nevada	Stamps.	30	....	"	93.1		5 or 1
North Mexican, Chihuahua	"	30	....	"	82.9		6 or 7

The reasons for employing eight different methods in the first testing of an ore are as follows:

1st. The assay-office extraction, which serves as a standard for mill-work, should always be the highest attainable by any use of hyposulphite and blue-stone.

2d. Collecting all the results by various methods on many samples of roasted ores, it is found that in the eight methods given are comprised all those which give the highest results on one or more samples of the ore investigated—these samples representing the ores treated in the principal mills of the United States and Mexico.

3d. If the best one of only two or three methods is used in the assay-office, the extraction in the mill may be greater than in the assay-office. Examples of this may be seen in Table IV., in the cases of Ontario, Lake Valley and Sombrerete ores. At Lake Valley (and also at the Rubio mill in Parral) the extraction in the mill was sometimes 6 to 10 per cent. above the extraction in the assay-office.

TABLE IV.

COMPARISON BETWEEN THE RESULTS BY RUSSELL PROCESS IN THE ASSAY-OFFICE AND THE RESULTS BY RUSSELL PROCESS IN THE MILL.

Name of Mine.	Kind of Furnace Used.	Mesh of Screen.	Per cent. of Salt.	No. of Assay OfficeExtra Solution Leaching Methods Tried.	Per cent. by Best one of those Methods in the Ass'y Office.	Mill Per cent.	Per cent Extracted in Ass'y Office more than in Mill.
Ontario..................	Stetefeldt.	30	0	2	85.8	85.7	+ 0.1
“	“	30	9½	2	94.7	96.2	— 1.5
“	“	20	12½	2	97.0	97.0	0.0
“	“	16	7½	2	89.0	89.5	— 0.5
“	“	16	8	2	91.5	92.0	— 0.5
“	“	16	12	2	96.3	97.1	— 0.8
“	“	16	16	2	95.0	95.7	— 0.7
“	“	16	18	2	95.9	95.1	+ 0.8
Lake Valley..........	Howell.	16	7	2	82.6	85.3	— 2.7
Sombrerete............	Reverb'y.	8 screen but with fine half out.	10	2	86.2	86.5	— 0.3
“	“	8	10	2	90.6	91.3	— 0.7
San Bartolo............	Howell.	16	9	8	88.5	87.3	+ 1.2
San Antonio..........	Howell.	26	10	8	89.2	88.5	+ 0.7
“	“	12	10	8	90.3	83.9	+ 6.4
San Miguel............	Howell.	26	8	8	91.9	90.4	+ 1.5
“	“	12	8	8	89.1	84.0	+ 5.1
Yedras....................	Brückner.	26	7	8	83.1	82.6	+ 0.5

4th. If the best one of only two or three methods is used in testing a new ore, the applicability of the process to that ore is not determined.

Usually there is not much difference in the results obtained by the eight methods; but sometimes, as in the case of the Yedras roasted ore, the difference is considerable. On that ore, a strong hot extra-solution in the assay-office, as in method 6 or 7, gave only 36 per cent. extraction. The mill-results on the same ore with a strong hot solution were 86 per cent. If only methods 6 or 7 had been employed in the preliminary testing of Yedras ore, the process might have been supposed inapplicable to that ore. On that ore the highest percentage in the assay-office was obtained by a cold, weak extra-solution, the result of which agreed with the results of a strong hot extra-solution in the mill.

It should be observed that the terms "strong" and "weak" do not mean the same in the mill as in the assay-office—the weakest extra used in the assay-office being stronger than the strongest in the mill.

In general, the method most applicable in the assay-office is not the most applicable in the mill, and frequently, as in the case of the Yedras ore, they are directly opposite. As pointed out further on, the proper methods for mill-work on roasted ores are very accurately indicated by the alkalinity or acidity of the first wash-water. For all raw ores, the manipulations are simpler and the methods very much more uniform.

D.—*Determination of the Percentage of Soluble Salts in Roasted Ore.*

In calculating the apparent percentage of silver extracted in the mill from roasted ores, it is necessary to know the percentage of salts which have been extracted from it by the first wash-water and the leaching-solutions.

If the ore contains no hydrate or sulphate of lime, or other salts of about the same solubility in water, the determination of the solubility is a simple operation: 10 or 20 grammes of the ore is placed on a weighed filter and leached with 500 to 600 c. c. of hot water. The residue and filter are then weighed. This method will give the amount of soluble salts extracted from the ore in the mill, usually within one-half and nearly always within three-quarters of 1 per cent., the amount of gold, silver, copper and lead salts which are extracted by the regular leaching-solutions rarely exceeding one-half of 1 per cent.

But if the roasted ore contains hydrate or sulphate of lime, the above method gives too high a solubility, owing to the solubility of these salts in a large amount of water. For instance, at Yedras, the solubility as determined by the above method was over three times as great as it actually was in the mill. In such cases the determination of the solubility is as follows: 10 or 20 grammes of the roasted ore is placed on a weighed filter and the same amount of the mill-tailings corresponding to that ore is placed on another weighed filter. Each is leached first with 300 to 500 c. c. of cold water and then with 2000 to 2500 c. c. of a strong (8 to 12 per cent.) hot hyposulphite solution. These volumes will be sufficient to dissolve all the salts which are in the least degree soluble in water or hyposulphite. The two residues and filters are then dried and weighed. The difference between the percentage of solubility of the ore and that of the tailings, gives, with sufficient exactness, the percentage of salts which has been extracted in the mill by both water and solutions. The slight error is due to the fact that the ore and tailings are not of the same nature—the latter having been concentrated previously to the weighing, by an amount equal to the solubility in the mill. But the determination is sufficiently exact for practical purposes, as an error of 1 per cent in the solubility makes very little difference in the apparent extraction, as shown in the following table, which also illustrates the mode of determining the apparent extraction. The two solubilities are assumed as being 9 and 10 per cent. respectively:

	Example I.	Example II.
Value of roasted ore, ...	25.0 oz.	25.0 oz.
Percentage of solubility in assay-office,	9.	10.
Calculated value of ore, ...	27.47 oz.	27.78 oz.
Value of mill-tailings, ...	2.5 oz.	2.5 oz.
Apparent mill-extraction, ...	90.9 per cent.	91.0 per cent.

The determination of the percentage of solubility should be made each day, or if the leaching-tanks have a capacity of 20 to 50 tons, it should be determined for each charge of ore.

E.—*Determination of the Strength of the Hyposulphite or Stock-Solution.*

As it is important that the proper relation should be observed between the amount of hyposulphite and that of blue-stone in the extra-solution, the strength of the hyposulphite or stock-solution should be determined whenever the extra-solution is to be made up for a large charge of ore. For small charges, say of ten tons each, a determination of the strength for each charge would be inconvenient; hence the strength is tested only once each day.

The best method of determining the strength of the hyposulphite solution is by starch and iodine. The reactions for this method are given in works on chemistry, but a description of the practical application of this method to the leaching process cannot safely be omitted here. If a small quantity of starch is present in a hyposulphite solution, and a solution of iodine is added, drop by drop, the sensitive blue color of iodide of starch will continue to be destroyed as fast as it is produced, as long as any hyposulphite exists. As soon as the hyposulphite has been converted into tethrathionate, the least excess of iodine will act upon the starch, and the blue color will be permanent. This method will give the total amount of hyposulphite, no matter in what form it exists, whether as sodium, calcium or silver hyposulphite, except as it exists in the extra-solution, or in the presence of caustic alkali. In the latter case, the method will indicate the presence of more hyposulphite than actually exists. Consequently, the hyposulphite solution used for the test should be neutralized (if alkaline), particularly in treating roasted lime ores, by the addition of a few drops of dilute sulphuric acid, before adding the iodine solution. No harm is done if the solution is slightly acid, provided the iodine and starch are immediately added. As just stated, the method is not applicable for the determination of the hyposulphite in the extra-solution, so that the sample of stock-solution must be taken from the solution which is to be used for the extra-solution before the blue-stone is added to it.

The standard iodine solution for a volume of one liter is made up as follows: The iodine is such as can be obtained at any drug store, no re-subliming being necessary. 5.11 grammes of the iodine is placed in a beaker of at least 200 or 300 c.c. capacity, which is set in a moderately warm place on the sand-bath at an angle of 20 or 30 degrees from the perpendicular. The iodine is then covered with about its own weight of iodide of potassium, and about 15 c.c. of water. In about twenty minutes, the iodine will be nearly or quite dissolved. About 200 c.c. of cold water is then added and the clear liquid decanted off. If any iodine still remains undissolved about three or four more grammes of iodide of potassium is added. As a little water has probably been left in the beaker from the decantation, no more need be used. The beaker is placed as before in an inclined position in a warm place. Usually this second addition of iodide of potassium will result in the complete solution of the iodine. If not, the second operation should be repeated. The whole solution is then diluted to 1000 c.c. with cold water and placed in a glass-stoppered bottle. 100 c.c. of this solution should correspond to one gramme of hyposulphite of soda ($Na_2S_2O_3 + 5aq$). But, as the solution may slightly decompose during the first few days after its preparation, it is best to use 5.5 grammes of iodine instead of 5.11 grammes in making up the standard solution. After a few days, the solution may then, if necessary, be diluted by the addition of a small amount of water, so that 100 c.c. shall correspond to one gramme of hyposulphite. This exact proportion is only necessary for the convenience of the man in charge of the leaching in the mill, by whom the solution is used.

The preparation of the starch solution is as follows: About 10 grammes of common starch is boiled or heated in about 500 to 1000 c.c. of water. Enough salt is then added to this solution to completely saturate it. The clear or turbid liquid is then decanted into a stoppered bottle, leaving the undissolved salt behind. The object of the addition of salt is to preserve the starch solution, which would otherwise decompose, and have to be made fresh every few days.

To standardize the iodine solution, about .1 to .2 of a gramme of a clear crystal of commercial hyposulphite of soda is weighed out in the fine assay-balances used for weighing gold and silver buttons. It is placed in a beaker with about 20 c.c. of water and 2 to 4 drops of starch solution. The iodine solution is then run in from a burette, until the blue color of iodide of starch is permanent.

The determination of the strength of the stock-solution is made in the mill by the man in charge of the leaching at the time the extra-solution is to be prepared. The reagents required are the iodine solution, the starch solution, and, if the ore is alkaline roasted, some very dilute sulphuric acid. The apparatus consists of a 10 c.c. pipette, a 100 c.c. burette graduated to $\frac{1}{5}$ or $\frac{1}{10}$ of a c.c., a glass beaker or tumbler and a glass stirring-rod. These are kept in a small closet near the leaching-tanks. As each charge of ore weighs nearly the same, each charge of extra-solution is always of the same volume. Consequently, a table can be prepared showing the number of pounds of hyposulphite present in the solution, to which the blue-stone is to be added, corresponding to any percentage of hyposulphite as shown by the test. In making up the extra-solution therefore, the man in charge measures out 10 c.c. of the stock-solution, adds a few drops of starch solution, runs in the iodine solution from the burette and reads off the percentage of hyposulphite. By referring to the table, he finds opposite this percentage, the amount of hyposulphite, say 390 pounds present in the volume of stock-solution which is to be used for the preparation of the extra. As the amount of blue-stone for a 50 ton charge at the rate of, say, 5 pounds per ton, would be 250 pounds, and as the amount of hyposulphite for the extra-solution must be twice as great as the weight of blue-stone, the amount of hyposulphite to be added with the blue-stone is 110 pounds.

F.—*Assaying of the Sulphides for Silver.*

The set of experiments shown in Table V. was made by Louis Janin, Jr., when in charge of the mill at Cusi. In experiments 1 and 2, of the 55 grammes of granulated lead, 15 were first placed in the scorifier and hollowed out to receive the sulphides, which should not touch the sides or bottom of the scorifier. The remaining 40 grammes were placed on the top of the sulphides.

In No. 4, the sulphides were first treated with nitric acid and the silver was precipitated from the filtrate with hydrochloric acid and filtered off—this filter, with its contents, then being added to the filter containing the residue undissolved by nitric acid. The two filters and their contents were then dried and assayed.

For assaying cupels, pulverize the cupel through a 30-mesh screen. Use 30 grammes fused borax, 30 grammes litharge and 30 grammes of carbonate of soda, and cover with salt. Use a slow fire until nearly fused. If the crucible tends to boil over, throw in 20 to 25 grammes of salt. For slags, all the slag from the assay is saved and pulverized through a 20-mesh screen, and mixed with flux used in experiment No. 4.

TABLE V.

EXPERIMENTS IN ASSAYING "SULPHIDES."

Number of Experiment.	Method of Assay Used.	Weight of Sulphides, A. T.	Weight of Fluxes Used.						Uncorrected Assay Value, oz. per ton.	Silver in Slag, Per cent.	Silver in Cupel, Per cent.	Total in Slag and Cupel, Per cent.	Corrected Assay Value, oz. per ton.
			Litharge, grms.	Sodium Carbonate, grms.	Borax, grms.	Potassium Cyanide, grms.	Charcoal, grms.	Granulated Lead, grms.					
1	Scorification............	0.1			5.			55.	11595	0.28	1.04	1.32	11750
2	Scorification............	0.05			5.			55.	11440	0.38	.48	0.86	11538
4	Crucible, with nail..	0.20	22.	12.5			0.7		11482	0.14	.31	0.45	11533
9	" no nail......	0.05	15.			25.			11399	0.13	.12	0.25	11427
3	" with nail..	0.1	22.	12.5			0.7		11272	0.27	1.09	1.36	11425
5	" no nail......	0.20	88.	11.					11375	0.10	.08	0.18	11395
8	" no nail......	0.05	77.	5.5			0.6		11252	0.09	.12	0.21	11275
6	" no nail......	0.20	25.			25.			11092	0.20	.19	0.39	11135

As shown in Table V., the method by scorification, using one-tenth of an assay-ton, gives the highest "uncorrected" as well as the highest "corrected" assay, but with the exception of No. 3, carries the most silver (1.32 per cent) into the slag and cupel. Nos. 5 and 8, which contain the most litharge (77 and 85 grammes) have the least silver in the slag and cupel (0.2 per cent.); Nos. 6 and 9, with cyanide of potassium, have a little more silver (0.32 per cent) in the slag and cupel. No. 3 is a common crucible-assay, made with a nail, but only 22 grammes of litharge. The slag and cupel contain the most silver (1.36 per cent.) of all the methods.

G.—*Assaying of the Sulphides for Gold.*

In assaying for gold the sulphides should always be first treated with concentrated nitric acid and only the residue assayed. Without the preliminary treatment with acid, the amount of gold obtained by assay will usually fall considerably short of the true amount. The following is the exact method:

Weigh out into a beaker (600 to 800 c. c. capacity) 1 to 3 assay-tons of sulphides. Cover with a watch-glass and add 200 c. c. of C. P. nitric acid, at the rate of 6 or 8 c. c. at a time. Set the beaker in a moderately warm place on the sand-bath. After action has nearly ceased, add a few more centimeters of acid. If action begins again, continue to add the acid until it ceases. Dilute with about 400 c. c. of pure hot water; let stand about an hour, and filter off the undissolved residue. Dry the residue and filter-paper and burn the latter. As the residue will contain considerable silver undissolved, no more need be added on account of the gold.

H.—*Definition of Terms Used in this Paper.*

The terms "ordinary," "old" or "simple" hyposulphite solution and process, refer to the sodium or calcium hyposulphite solution and to the Kiss and Patera process in which these solutions alone are used.

The term "Russell process" includes the use of cuprous hyposulphite, acid, or any alkaline carbonate, as soda-ash, one or more of them, either without or in connection with the use of the sodium or calcium hyposulphite solution as used in the Kiss and Patera process.

"Extra-solution" properly means the cuprous hyposulphite solution of the Russell process, and, to save repetition, the two terms are used interchangeably in this paper.

The "special extra-solution" is an extra-solution which has already been used on one charge of ore and is to be used on a second charge just previously to the use of a fresh extra-solution on that charge.

"Apparent extraction" is the extraction in percentage or in ounces by leaching in the mill, calculated from the value of the tailings and ore per ton, or, in other words, is the difference between the value of the tailings and that of the ore charged to the leaching tanks. The apparent extraction is calculated each day.

"Actual extraction" is the amount of silver actually obtained in the bullion or sulphides. When expressed in percentage, it is the amount of silver actually obtained compared with the silver in the ore actually charged into the leaching-tanks. The actual extraction is known only at the end of each leaching-run, usually of 100 to 1000 tons of ore.

Neither the apparent nor the actual extraction refers to any losses occurring previously to the charging of the ore into the leaching-tanks.

An "acid ore" as used in this paper, is a roasted ore, the first wash-water from which gives an acid reaction. Of this class are the Ontario (in 1883–84), Sombrerete, San Antonio and San Bartolo.

A "simple alkaline ore" is a roasted ore with an alkaline first wash-water, of which the Sierra Grande, Daly and Veta Grande ores are examples.

An "alkaline arsenical ore" is an ore with considerable arsenical compounds and an alkaline first wash-water. The Yedras ore is the only one of this class so far treated by the Russell process.

The "first wash-water" is used before the regular leaching-solutions and only on roasted ores, being employed to extract all salts formed in the roasting which are soluble in water.

The "second wash-water" is used after the regular leaching-solutions, on all ores, for the purpose of extracting the portion of the hyposulphite solution remaining in the ore, and also for the purpose of maintaining the volume of the stock-solution.

"Volume to saturate," for raw ores, means the volume of liquid which the charge of dry ore has absorbed per ton by the time the surface of the liquid stands level with the surface of the charge. For roasted ores, it means the volume of liquid which a charge of wet ore will absorb in the same way. By a "wet charge" is meant a charge from which water or solution has just been drained, but which still retains from 10 to 20 per cent. of moisture.

"Rate of leaching" is the number of inches depth of liquid, measured on the surface of the charge, which will pass through that charge per hour.

"Circulation" is the re-passing of the solution (usually the extra-solution) through the charge of ore, after it has already passed through one or more times.

By "tailings" is understood any material which has been treated by leaching or amalgamation, either raw or after roasting. In this paper, the term "roasting" or "roasted ore" will include chloridizing and chloridized ore. The term "Howell furnace" will include any self-discharging revolving furnace, either partially or wholly lined with brick. All expenses are given in U. S. currency, and the value of one ounce of silver is estimated at $1.

II. Mill Work.

A.—*Preparation of the Ore for the Process.*

a. The Crushing and the Effect of Various Sizes of Crushing on the Rate of Leaching, Roasting and Mill-Results.—All crushing of raw as well as of roasted ores must be dry. Also all tailings, which are to be treated by any leaching process, must be dried if they contain any wet slimes. For ores which are to be roasted, the reason for dry crushing is apparent. In the case of raw ores, a wet-crushing causes a separation of the coarse and fine particles, which is very objectionable. Table VI gives the size of crushing for the Russell process in various mills. The extraction from Silver Reef and Raymond and Ely raw ores would probably be increased by a finer crushing.

The leaching-rate of roasted ore is greater than that of raw ore, unless the soluble salts formed in roasting have first been dissolved from the roasted ore before it is charged into the leaching-tanks, in which case the leaching-rate is about the same as for raw ores. The fineness of crushing has, within certain limits, no appreciable effect on the rate of leaching, if the ore is well chloridized and is charged dry into the leaching tanks. For instance, when treating Ontario ore, a charge of the finest flue-dust from the farthest dust-chamber of the Stetefeldt furnace leaches just as rapidly as a charge of the coarsest material from the furnace-shaft. This is due to the perfectly roasted condition of the flue-dust, which causes the charge to maintain a soft and half-floating condition during the whole of the leaching, unless it is allowed to drain and settle for a considerable time. On the other hand, at Lake Valley, with a Howell furnace without an auxiliary fire, the flue-dust was so poorly roasted that even when mixed with the coarsest ore, it proved a great hindrance to rapid leaching.

TABLE VI.

LEACHING-RATE, SIZE OF CRUSHING, AND VALUE PER TON, OF MATERIAL TO WHICH THE RUSSELL PROCESS HAS BEEN APPLIED.

General *Description* of Material Treated.	Tailings *or* Ore.	Name and Location of Mine or Mill.	Rate Leaching. Inches per hour.	Value of Material. Oz. Silver per ton.	Size of Screen Used.
Previously treated by other processes one to three times.	Tailings.	Leeds, etc., Silver Reef, Utah		8.	Not Re-crushed. Orig. size not known.
	"	Bremen, Silver City, New Mexico......	½	11.5	
	"	Raymond & Ely & Meadow Valley, Pioche, Nev..		8.	
	"	Veta Grande, Parral, Mexico.............		12.	
Raw ore.	Ore.	Silver Reef, Utah................................		15.	10 Mesh.
	"	Raymond & Ely, Pioche Nevada		14.	4 "
Chloridized material with alkaline first wash-water.	Ore.	Sierra Grande, Lake Valley, New Mexico..	8	12.	16 "
	"	San Miguel, Cusihuiriachic, Mexico...	5	50.	12 "
	"	Yedras, Sinaloa, Mexico....................	7	62.	26 "
	"	Daly, Park City, Utah. (Experimental Plant)..		40.	20 "
	"	*Veta Grande, etc., Parral, Mexico*		32.	16 "
	"	Ontario, Park City, Utah, 1887...........		44.	26 "
	Tailings.	Sierra Grande, Lake Valley, New Mexico..	8	12.	16 "
Chloridized material with acid first wash-water.	Ore.	Ontario, Park City, Utah, 1883-4. (Experimental Plant)	10	88.	16 "
	"	Sombrerete, Zacatecas, Mexico..........	12	48.	8 "
	"	San Antonio, Cusihuiriachic, Mexico	10	63.	12 "
	"	Chloride, Chloride, New Mexico.........	5	26.	20 "
	"	San Miguel, Cusihuiriachic, Mexico	7	45.	12 "
	"	San Bartolo, " "	8	40.	12 "

NOTE.—All the above rates of leaching except for the Bremen tailings at Silver City are without syphons or vacuum.

For ores which are to be roasted in a Stetefeldt furnace, or which are to be thoroughly roasted in a Howell or reverberatory, there is practically no limit to the fineness of crushing allowable. But a roasting in a Brückner furnace may have a very bad effect on the rate of leaching, as shown further on.

For raw ores, if rolls are used, the limit of coarseness may be put at an 8-mesh screen. In the case of raw tailings, the limit of fineness allowable has only been reached in the treatment of the Bremen tailings at Silver City, New Mexico. The average fineness of that part of those tailings which could be successfully treated was such that 87.8 per cent. would pass through a screen of 150 holes to the linear inch, or 22,500 holes to the square inch. With this fineness of material, a vacuum of 14 inches of mercury produced a leaching-rate of about one-half inch per hour, measured on the surface of the charge. These tailings came from the upper end of the Bremen tailing-pits, which, being nearest to the mill which had produced the tailings, contained the coarsest material. The tailings gradually became finer, the leaching rate decreasing until it became one-seventh to one-tenth of an inch per hour. The treatment of the tailings was then abandoned. The ore from which these tailings had been produced was originally crushed through screens of 40 to 60 meshes to the linear inch. Subsequently the tailings had been once amalgamated and once concentrated. The extreme fineness of these tailings was due to the slate and lime composing the ore, which had decomposed and weathered during the three previous treatments, and during years of exposure to air and moisture in the tailing-pits.

Table VI gives the average rate of leaching in the mills for various ores crushed through different screens and treated by the Russell process.

TABLE VII.

RESULTS OF ASSAY-OFFICE LEACHING-TESTS. ONTARIO ORE ROASTED IN A STETEFELDT FURNACE.

Conditions Under which the Mill-Tests Were Made.	By which Process Treated in Assay Office.	Per cent. Extracted from Sample taken after Five Seconds in Furnace.	Per cent. Extracted from Sample taken after Lying on Cooling Floor.
16 Screen—eight per ct. Salt added between Battery and Furnace.....	Russell Process...........	81.6	89.4
	Ordinary "	64.3	85.0
30 Screen—seven and three-tenths per cent. Salt added between Battery and Furnace......................	Russell Process...........	86.4	91.4
	Ordinary "	62.7	80.6
16 Screen—twelve per cent. Salt added between Battery and Furnace..	Russell Process...........	90.0	96.2
	Ordinary "	76.0	90.8
20 Screen —twelve per cent. Salt added between Battery and Furnace..	Russell Process...........	91.9	97.0
	Ordinary "	84.6	92.5
		87.5	93.5
		71.9	87.2

Tables VII, VIII, IX, X, XI, XII, XIII, XXVI, XXVII, XXVIII and XXXI illustrate the changes taking place during roasting, and the effect of various sizes of crushing and percentages of salt on the results of the furnace-work, as shown by assay-office leaching-tests. Tables VII and VIII referring to the Stetefeldt furnace, IX to the Howell, X and XI to the reverberatory, and XII, XIII and XXVIII to the Brückner, while tables XXVII and XXXI sum up and compare the furnace-work by all the furnaces.

In regard to the size of crushing, Table VII shows that with 12 per cent. salt a 16- and a 20-mesh screen on Ontario give about the same results. These results are probably as high as could have been given by a 30-mesh. The results are given more in detail in Table VIII.

TABLE VIII.

CHANGES TAKING PLACE DURING ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS. ONTARIO ORE ROASTED IN A STETEFELDT FURNACE.

			Mill Run No. 1.	Mill Run No. 2.	Mill Run No. 3.	Mill Run No. 4.	Mill Run No. 5.	Mill Run No. 6.	Mill Run No. 7.	Mill Run No. 8.	Mill Run No. 9.
		No. of Furnace	2	2	1	2	2	2	2	2	2
		Mesh of Screen	30	30	30	30	20	16	16	16	16
		Per cent. of Salt Used	0	0	5½	9	12½	8	12	16	18
		Oz. Silver in Ton of Raw Ore	74.8	82.4	72.0	70.4	98.0	85.0	88.0	61.0	74.4
Samples Taken from Inside of Furnace Immediately After Falling. Time, Five Seconds.	"Shaft."	Oz. Silver per Ton	66.0	62.0	65.0	58.4	88.9	61.6	76.8	56.0	68.0
		Per cent. of Soluble Salts	6.5	3.5	8.5	12.0	11.5	7.8	16.0	17.2	19.0
		Per cent. by "Ord." in Assay Office	30.9	31.5	39.3	85.3	80.0	63.6	83.6	76.5	60.8
		Per cent. by "Extra" in Assay Office,	66.5	50.0	82.6	89.0	89.7	77.3	90.0	84.4	88.4
	Ret. Flue.	Oz. Silver per Ton	94.0	84.0	76.8	76.0	108.0	92.4	91.2	73.0	85.0
		Per cent. of Soluble Salts	2.0	1.5	8.0	11.0	11.5	7.6	16.8	18.0	19.2
		Per cent. by "Ord." in Assay Office	29.0	15.0	44.8	81.6	88.0	65.0	68.5	86.1	78.4
		Per cent. by "Extra" in Assay Office	61.6	45.5	82.8	91.6	94.1	86.0	90.0	88.9	91.1
Samples Taken from Cooling Floor After Lying Twelve Hours.	"Shaft."	Oz. Silver per Ton	70.0	60.0	66.0	58.4	89.2	60.0	76.8	58.3	63.2
		Per cent. of Soluble Salts	8.5	6.7	12.0	12.0	12.0	8.0	16.0	17.0	20.0
		Per cent. by "Ord." in Assay Office	74.2	83.4	60.1	94.4	92.0	82.8	90.6	89.0	88.0
		Per cent. by "Extra" in Assay Office	84.0	85.8	90.0	95.2	96.9	88.5	96.0	93.5	95.2
	Ret. Flue.	Oz. Silver per Ton	....	....	76.8	73.0	108.0	92.8	91.1	72.3	78.4
		Per cent. of Soluble Salts	....	....	11.0	11.0	12.2	8.2	17.0	18.6	21.0
		Per cent. by "Ord." in Assay Office	....	....	76.0	92.0	93.1	88.3	91.1	94.7	93.0
		Per cent. by "Extra" in Assay Office	....	....	86.6	94.1	97.1	90.3	96.5	95.6	97.0

For San Antonio ore, in a Howell furnace (Table XXVII) the change from 26- to 12-meshes produces no effect; while the San Miguel results are depressed 2.8 per cent. by the same change. It should be stated that, while roasting San Antonio ore, some San Bartolo ore was always present.

Sombrerete ore (Table XXVII) in a reverberatory yields 2.7 per cent. more with an 8-mesh screen than with a 30-mesh.

Table XII apparently indicates that a 10-mesh screen gives better results than a 26-mesh on Yedras ore, with a Brückner furnace. That such is not the case, is shown by the vault samples (Table XXVIII) for the same charge, which show only 67.5 per cent. extraction or 16.6 per cent. less than the furnace sample. The "vault" referred to is the chamber into which the roasted ore falls as it comes out of the furnace. The results on another charge, also (Table XIII) show that the Brückner furnace results are 4.5 per cent. less with a 10-mesh screen than with a 26-mesh.

TABLE IX.

CHANGES TAKING PLACE DURING ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS. CUSI ORE ROASTED IN A HOWELL FURNACE.

Conditions Under which the Mill-Tests Were Made.	By which Process Treated in Assay-Office.	Per cent. Extraction after 5 Minutes in the Furnace.	Per cent. Extraction after 15 Minutes in the Furnace.	Per cent. Extraction after 20 Minutes in the Furnace.	Per cent. Extraction after 25 Minutes in the Furnace.	Per cent. Extraction after 30 Minutes in the Furnace.	Per cent. Extraction after 35 Minutes in the Furnace.	Per cent. Extraction after 40 Minutes in the Furnace.	Per cent. Extraction after 45 Minutes in the Furnace.	Per cent. Extraction after Lying 3 Hours in Vault.
12-Screen—eight per ct. Salt	R. Process.	34.2	46.4	59.7	65.2	75.0	85.0	85.3	77.3	91.8
Two Rings—No Shelves........	Ordinary.	7.0	9.0	12.4	....	22.4	60.0	55.2	40.0	74.0
12-Screen—eight per ct. Salt	R. Process.	53.7	....	59.1	62.0	72.0	60.5	60.0	61.2	92.4
Two Rings—No Shelves........	Ordinary.	13.8	....	13.0	17.2	50.0	22.0	28.9	29.0	76.7
12-Screen—eight per ct. Salt	R. Process.	36.5	40.0	48.6	83.0	59.0	56.0	Out of Furnace	Out of Furnace	87.3
No Rings—3 Broad Shelves...	Ordinary.	0.0	9.0	9.2	56.0	36.0	20.0			79.9
12-Screen—eight per ct. Salt	R. Process.	33.0	54.2	58.0	85.7	70.0	68.0			89.2
One Ring—3 Narrow Shelves	Ordinary.	0.0	10.0	15.0	66.7	44.2	25.0			81.7
Average,	R. Process.	39.3	46.9	56.3	73.9	69.0	67.4	72.6	69.2	90.2
	Ordinary.	5.2	9.3	12.4	46.6	38.1	31.7	42.0	34.5	78.1

Table IV shows the effect of coarse- and fine-crushing on the leaching-results in the mill. For Ontario and Sombrerete, the relation between the assay-office and mill-results remains the same whether a coarse or fine screen be used, and, for Ontario ore, whether the ore was from the shaft or the dust-chambers. But in the case of San Antonio the discrepancy between the mill and the assay-office results was increased 5.7 per cent. by a change in crushing from a 26- to a 12-mesh screen, and in the case of San Miguel ore, the same change caused an increase in the discrepancy of 3.6 per cent. As a rule, however, a 16-mesh screen gives as good results as a 30-mesh, and the leaching-results in the mill will average within 1 per cent. of the results in the assay-office. Both the apparent exceptions given above may be due to defects in the roasting. At Lake Valley, owing to the imperfection of the screening apparatus, much ore had to be leached which would not pass a 10-mesh screen. But it was found that the material which would not pass such a screen was chloridized just as high and yielded as high a percentage in both assay-office and mill as that portion which would pass a 40-mesh. In the case of the Bremen tailings, as long as the leaching-rate remained at about one-half inch or more per hour, the difference between the mill and the assay-office extraction was only one-tenth to one-half ounce per ton.

TABLE X.

CHANGES TAKING PLACE DURING ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS. SOMBRERETE ORE ROASTED IN A REVERBERATORY FURNACE.

Conditions Under which the Mill-Tests Were Made.	By which Process Treated in Assay-Office.	Per cent. Extraction after 4 Hours in the Furnace.	Per cent. Extraction after 5 Hours in the Furnace.	Per cent. Extraction after 6 Hours in the Furnace.	Per cent. Extraction after 7 Hours in the Furnace.	Per cent. Extraction after 8 Hours in the Furnace.	Per cent. Extraction after 9 Hours in the Furnace.	Per cent. Extraction after 10 Hours in the Furnace.	Per cent. Extraction after 11 Hours in the Furnace.	Per cent. Extraction after 12 Hours in the Furnace.	Per cent. Extraction after 13-16 Hrs & Final Per cent.
30-Screen........................	R.Process	30.7	28.0	30.2	40.8	50.0	59.1	58.4	....	56.4	85.7
Ten Per cent Salt............	Ordinary	7.0	14.0	7.5	7.8	11.0	18.3	27.0	....	27.1	70.9
30-Screen........................	R Process	27.6	....	22.4	38.0	48.2	....	49.5	82.5	....	87.8
Ten Per cent. Salt..........	Ordinary	11.8	....	8.0	14.8	10.0	....	6.0	64.3	....	81.0
8-Screen........................	R.Process					41.2	63.4	71.1	84.3	86.7	94.3
Ten Per cent. Salt..........	Ordinary					10.0	19.9	13.8	59.6	44.2	83.7
Fifteen Per ct. Lead Ore	R.Process					38.0	....	42.7	38.0	88.0	92.4
8-Screen, Ten Pr. ct. Salt.	Ordinary					6.0	....	9.7	7.9	80.0	87.9
Average,	R.Process	29·1	28.0	26.3	39.4	44.3	61.2	55.4	68.3	77.0	90.0
	Ordinary	9.4	14.0	7.8	11.3	9.3	19.1	14.1	43.9	50.4	80.9

b. Effect of Various Percentages of Salt on the Roasting.—In regard to the percentage of salt for Ontario ore and a Stedtfeldt furnace, Table VII shows that either with a 16- or a 20-mesh screen 12 per cent. of salt gives the best results. The tables do not show the comparison of results caused by different percentages of salt with

TABLE XI.

CHANGES TAKING PLACE DURING ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS. YEDRAS ORE ROASTED IN A REVERBERATORY FURNACE.

Conditions Under which the Mill-Tests Were Made.	By which Process Treated in Assay Office.	Per cent. Extracted after 2-4 Hours in Furnace.	Per cent. Extracted after 5-7 Hours in Furnace.	Per cent. Extracted after 8-10 Hours in Furnace.	Per cent. Extracted after 11-13 Hours in Furnace.
26-Screen	Russell Process	72.5	78.2	87.0	90.0
Seven Per cent. Salt	Ordinary "	33.9	43.4	71.6	77.9
26-Screen	Russell Process	63.1	72.2	86.4	87.4
Seven Per cent. Salt	Ordinary "	25.7	37.7	66.5	78.7
26-Screen	Russell Process	69.2	67.9	77.5	78.5
Seven Per cent. Salt	Ordinary "	49.9	51.8	51.0	48.6
26-Screen	Russell Process	....	40.3	49.5	91.0
Seven Per cent. Salt	Ordinary "	....	23.4	46.6	66.3
Average,	Russell Process	68·3	64.6	75.1	86.7
	Ordinary "	36.5	39.1	58.9	67.9

TABLE XII.

CHANGES TAKING PLACE DURING ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS. YEDRAS ORE ROASTED IN A BRÜCKNER FURNACE.

Conditions Under which the Mill-Tests were Made.	By which Process Treated in Assay Office.	Per cent. Extracted after 2-4 Hours in Furnace.	Per cent. Extracted after 5 7 Hours in Furnace.	Per cent. Extracted after 8-10 Hours in Furnace.	Per cent. Extracted after 11 Hours in Furnace.	Per cent. Extracted after 14-15 Hours in Furnace.
26-Screen	Russell Process	76.4	63.2	61.5	58.7	70.9
Seven Per cent. Salt	Ordinary "	64.3	26.4	52.5	54.3	63.1
26-Screen	Russell Process	78.6	80.5	71.4	....	....
Seven Per cent. Salt	Ordinary "	49.6	48.2	51.4	....	....
10-Screen	Russell Process	....	79.5	80.0	84.1	....
Seven Per cent. Salt	Ordinary "	....	52.9	31.1	32.5	....
26-Screen	Russell Process	60.6	82.6	86.0	85.5	85.7
Three Per cent. Salt	Ordinary "	35.0	49.8	48.9	33.6	13.4
Average,	Russell Process	71.9	76.4	74.7	76.1	78.3
	Ordinary "	49.6	44.3	46.0	40.1	38.3

a Howell furnace, but at Cusi the extraction in the assay office was not usually appreciably altered by any percentage of salt above 12 per cent. although as high as 16 per cent. was tried. The Howell furnaces used at Cusi were of cast-iron, 23 to 27 feet long and 60 inches in diameter at the lower end; the lower end, for 6 to 8 feet, being lined with brick and the remainder of the furnace being either with or without shelves, and with or without one or two brick rings to retard the ore. Tables XII and XIII show the effect of various percentages of salt on the results of the furnace samples for Yedras ore roasted in a Brückner furnace. But in this case, also, Table XXVII shows a decrease of 7 per cent. in extraction from the vault-samples, as compared with the furnace-samples.

TABLE XIII.

CHANGES TAKING PLACE DURING ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS. YEDRAS ORE ROASTED IN A BRÜCKNER FURNACE.

	No. of Charge.	Description of Charges.	Value of Roasted Ore. Oz. Silver per Ton.	Per cent. Extracted by Ordinary in Assay Office.		Per cent. Extracted by Extra in Assay Office.		Per cent. Difference Between Ord. and Extra in Assay Office.
RESULTS OF OXIDIZING ROAST (BEFORE SALT).								
Sec. I.	1	26-Screen	65.8	49.0	44.1	74.3	73.8	25.3
	2	" "	58.0	44.2		76.1		31.9
	3	" "	57.8	40.3		70.5		30.2
	4	10 "	68.8	42.8		74.3		31.5
RESULTS ON ABOVE CHARGES IMMEDIATELY AFTER ADDING SALT.								
Sec. II.	1	After Adding Seven Per cent. Salt	61.2	50.1	54.9	79.2	77.9	29.1
	2	" " Five " " "	57.6	49.7		70.7		21.0
	3	" " Three " " "	....	66.8		82.3		20.5
	4	" " Seven " " "	67.4	52.9		79.5		26.6
FINAL FURNACE-RESULTS ON THE SAME CHARGES.								
Sec. III.	1	Total Time in Furnace—10 Hours	61.1	52.7	40.8	74.9	69.7	22.2
	2	" " " " — 9 "	54.9	38.5		59.0		20.5
	3	" " " " — 7 "	58.9	54.3		74.4		20.1
	4	" " " " —12 "	61.0	17.9		70.4		52.5

As to the results of an oxidizing roast, the first two columns of Table VIII show the effect of such a roast in a Stetefeldt furnace on the results of assay-office leaching-tests for Ontario ore. Table XLI gives the corresponding mill-results. The first section of Table XIII shows the effect of an oxidizing roast in a Brückner furnace on Yedras ore. The second section gives the results obtained from the same charges 15 minutes to 1 hour after the addition of salt, and section 3, the final furnace-results. The results from the corresponding vault-samples would probably be 7 per cent. less.

The comparative applicability of the various furnaces in the preparation of ore for the Russell process will be considered later.

c. Effect on the Leaching of Wetting-Down Roasted Ore while Red-Hot.—The effect of wetting down roasted ore on the cooling floor while red-hot may be very injurious, as shown in the following statements, the first furnished by Mr. Frank Johnson:

San Bartolo Ore, Cusi, Mexico.

	Value of Ore.	Extraction	
		by Ordinary in Assay Office.	by Extra-Solution in Assay Office.
	oz. per ton.	per cent.	per cent.
Before wetting down......	49.1	69.1	86.3
After wetting down........	49.1	20.3	80.3

Sierra Grande Ore, Lake Valley, New Mexico.

	Value of Ore.	Extraction	
		by Ordinary in Assay Office.	by Extra in Assay Office.
	oz. per ton.	per cent.	per cent.
Before wetting down......	16.0	81.5	87.5
After wetting down........	16.0	62.5	71.9

The above decrease in extraction by both the ordinary and extra-solution is probably due to the reduction of some of the silver compounds to the metallic state by the steam formed by the red-hot ore and water. No decrease occurs unless the ore is red-hot when wet down. Although the above tables give only the assay-office results, the effect on the mill-results was approximately the same. Of the above ores, the first was acid, the second alkaline, and both were roasted in Howell furnaces.

The following statement shows the effect on the Sombrerete mill-results of wetting down that ore while red-hot—the ore being acid and roasted in a reverberatory and leached in charges of 4½ tons each :

	Value of Ore.	Extraction		
		by Ordinary in Assay Office.	by Extra in Assay Office.	by Extra in Mill.
	oz. per ton.	per cent.	per cent.	per cent.
Average mill results on ore wet down while red hot....................	35.5	77.5	84.6	80.6
Average mill results on ore *not* wet down while red-hot....................	36.5	74.0	89.3	88.0

In the above statement only the difference between the extra in the mill and that in the assay office should be noted. The other differences are probably not due to the wetting-down.

That wetting-down while red-hot is not injurious on all ores is indicated by the following statement, although it refers only to assay-office results. The ore is "alkaline arsenical," and was roasted in reverberatory furnaces. The difference in value may be due to a change in the lime-compounds.

Yedras Ore, Sinaloa, Mexico.

	Value in Ounces.	Extraction	
		by Ordinary in Assay Office.	by Extra in Assay Office.
		per cent.	per cent.
Before wetting down......	63.1	50.9	73.6
After wetting down........	55.9	48.6	73.7

d. Weighing and Charging the Ore into the Leaching-Tanks.—For the success of the whole plant as well as of the various leaching-operations, the weight of ore charged into the leaching-tanks must be accurately known. Otherwise, the causes of losses and discrepancies are not likely to be discovered. By weighing the ore into the leaching-tanks, it is known at once whether the losses occur during the preparation of the ore for leaching, or during the actual extraction of the silver, and treatment of the product. For instance, at Cusi, in 1887, a loss of about 14 per cent. was caused principally by dust, but, perhaps, partly by volatilization of the silver in roasting. That this loss occurred in the preparation of the ore and not in the leaching, could not have been determined, without weighing the ore between these two operations. On the other hand, if the weight of ore charged to the leaching-tanks is determined by a merely approximate method, as by weighing only 1 cubic foot of a charge, much time and labor may be expended in searching for the cause of an apparent discrepancy, three-fourths of which may not exist. Aside from the question of the extent of the inaccuracy of this method, the fact that it is at all inaccurate is demoralizing, and discourages the making of comparative experiments on all subsequent operations. It is not exaggerating the importance of this point, to say, that, upon the care used in weighing the ore charged to the leaching-tanks often depends the success or failure of the whole plant.

The manner of charging now requires attention. As the proposed method of "trough-lixiviation" does not do away with leaching-tanks, but is merely a method of charging the ore into the tanks, a treatment of that subject will not be out of place here. In this method the ore is dumped into a trough (or first into an agitator) together with water or solution. The trough conducts the material to the leaching-tanks. The result of using this method is a separating in the tank of the coarse and fine particles—the coarse particles sinking to the bottom, while a hard layer of impenetrable slimes forms on the top, if one tank is used, and the order is merely reversed if various tanks are connected. When raw ores and tailings are thus treated, no amount of vacuum (up to even 20 inches of mercury) will cause a downward leaching; and if pressure below the ore is used, the liquid simply breaks a hole through the charge at the side of the tank. If water has been used in the trough or agitator, as would be necessary in the treatment of roasted ore, it is almost impossible to introduce the solution into the ore. If solution is used first, as for raw ores, this silver-bearing solution cannot be removed. It is true that, after the supernatant solution has been allowed to settle for a long time on the surface of the charge, that part of the solution can be decanted off, but this represents only a portion of the silver-bearing solution.

Although nearly all the chloride of silver which can be dissolved by assay-office tests with the ordinary solution, can be dissolved in the mill by the above method, so that, by taking a sample from the charge, and washing it with water in the assay office, to remove the dissolved silver, nearly all the chloride of silver can be removed, yet the method is entirely inapplicable in the mill. Another objection to the trough system is that the sulphides produced from the supernatant solutions are much reduced in value by the slimes and dirt from the ore.

The trough system was tried at Bullionville, Nev., in 1884, at Silver City, N. M., on tailings, and at Lake Valley, N. M., on roasted ore, in 1885. Although two of the above mills had been especially adapted to the method, it was abandoned for the reason above given. The sulphides produced by the trough system at Bullionville and Silver City assayed, respectively, one-third and one-eighth as much as sulphides produced afterwards by the ordinary method.

In the *Engineering and Mining Journal* of Nov. 26, 1887, the statement is made that for Cusi ore the minimum amount of ordinary or stock-solution required for trough-lixiviation was 12 parts of solution to 1 of ore, which is 384 feet per ton, although the strength of the solution was 1.6 per cent. By referring to page 52, it will be seen that the average amount of stock-solution used per ton of this ore, in the usual method of leaching, for a period of 9 months and with a solution of only 0.75 per cent., was only 80 cubic feet. Hence the trough-lixiviation, using a solution twice as strong, and requiring 4.8 times the volume, necessitated the use of 9.6 times as much hyposulphite per ton of ore. One of the evil results of this would be the production of a lower grade of sulphides and an excessive use of precipitant, as the amount of lead dissolved is directly proportional to the amount of hyposulphite used. By referring to the account of some Yedras experiments in the same journal of Jan. 21, it will be seen that the injurious effect of caustic lime was the same while using trough-lixiviation as while leaching by the usual method, even though the tailings from the former were subjected to a subsequent treatment in the assay office.

By referring to a statementmade elsewhere in this paper it will be seen that the rate of decomposition of hyposulphite solutions is directly proportional to the extent of surface exposed, so that the decomposition of the solution will be greater for trough-lixiviation than for the usual method.

In fact, aside from the numerous disadvantages, actual experience indicates that the sole advantage, if any, obtained by this method of lixiviation is the saving of 12 to 15 cents per ton expense of moving the ore from the cooling-floor to the ore-tanks.

In brief, the objections to the method are:

1st. In the case of raw ores and tailings and of most roasted ores, the difficulty of removing the dissolved silver mechanically retained in the ore.

2d. The production from the supernatant solution of sulphides of much lower grade than those produced from material charged dry into the tanks.

3d. The difficulty experienced in first extracting the soluble salts from roasted ore (washing).

4th. The great increase in the amount of stock-solution used per ton of ore and its greater deterioration.

Returning to the general subject of charging, we have to consider the proportion between depth of charge and depth of leaching-tank. As explained further on, in the treatment of raw ores, the extra-solution is made up in a separate tank, hence no allowance is made for the making up of an extra-solution in charging the tanks. Consequently, in charging raw ores, the tanks may be filled level full, as a charge, 5 feet in depth, will certainly sink 4 or 5 inches, this space being advisable in order that any solution running upon the top of the charge shall not frequently run over. But, in charging roasted ore, sufficient space above the charge is left for the making-up of the extra-solution, unless it is necessary to run the leaching-tanks to their utmost capacity, in which case the leaching-tanks are charged level full and the extra-solution is made up in a separate tank as for raw ones. Generally, for roasted ores the depth of the extra-solution is about 34 to 46 per cent. of the depth of the ore at the end of the first washing with water. These are the proportions for roasted ores which sink about 2 to 4 per cent. during the washing.

The rate of leaching is not diminished by an increase in the depth of charge. For instance, at Cusi was one large tank, into which the roasted ore was charged to the depth of nearly 6 feet, while in all the other tanks the depth of charge was only 22 to 24 inches. Yet the rate of leaching was greater for the 6-foot charge. Also, in the treatment of the raw Bremen tailings, the extreme fineness of which has already been mentioned, the rate of leaching for charges 50 inches in depth, was the same as for only 24 inches. The cause of this is that the slight increase in average compactness of the charge, due to increase in depth, is more than balanced by the increase in "head" of the liquid in the ore. Of course, it is understood that, in each case, the depth of the liquid above the surface of the charge is about the same.

That large and deep charges are the most economical, appears from the following facts: The time of leaching a 50-ton charge of ore is the same as for a 10-ton charge, if the depth of ore is the same, and is comparatively but little increased by an increase of two, or three times in the depth. Less water per ton is required for a deep charge, as a given volume of water will dissolve more salts in passing through a 6-foot than a 2-foot charge. The same is true of the leaching-solutions with regard to the dissolving of the precious metals. The result of using deeper charges is a decrease in the amount of chemicals required per ton, particularly when using the extra-solution. The labor also is much decreased, the difference between 22 nine-ton tanks (as at Cusi) and 6 or 7 fifty-ton tanks would be a decrease of about 50 per cent. in the labor connected with the leaching operations. The decrease in assay-office work would be about 60 per cent. The amount of stock-solution required is also much less, which, as pointed out later on, is an important item. In fact the difference in expenses, in favor of 50-ton as compared with 10-ton charges, is not less than 65 cents per ton. For dry regions the decrease in the amount of water required is an important item.

B.—*The Chemicals.*

a. Conditions regulating the amounts used. As shown in Table XIV, the weight per ton of ore of each chemical required varies considerably. The circumstances governing the variation are as follows:

1. *Hyposulphite of Soda.*—The amount required per ton varies:

First, *according to the condition of the ore, i.e.*, whether raw or roasted. The weakening of the hyposulphite solution is chiefly due to its dilution with water while it is following the first wash-water through the ore, as in the treatment of roasted ore, and while the last wash-water is following the solution in the treatment of both raw and roasted ore. As no first, but only the last, wash-water is required in the treatment of raw ore, while both are necessary in the case of roasted ore, it is evident that the weakening of the solution in this way is twice as great for roasted as for raw ore. Also, raw ore requires less hyposulphite for another reason. In treating roasted ore, the extra-solution, being always preceded by the first wash-water and nearly always by the stock-solution, must necessarily be used on ore containing a liquid different from itself, with which it is diluted and weakened in passing through the ore. In making it up for roasted ore an additional amount of hyposulphite must, therefore, be used. But in treating raw ore or tailings, the extra-solution is nearly always used first, and therefore on material which is perfectly dry, and no allowance has to be made for dilution.

Secondly, *directly as the volume of stock-solution.* The hyposulphite of soda is added to the solution at the rate of a certain number of pounds per ton of ore. As the principal object is to maintain the strength of the stock-solution at a certain percentage, say, 1½ per cent, and yet use the least possible quantity of hyposulphite per ton of ore, it is evident that the smaller the volume of stock-solution the less hyposulphite will be required. For instance, if, in works treating 100 tons per day, the amount of stock-solution were 3000 cubic feet, the amount of hyposulphite required to raise the strength of that solution by any given percentage would be twice as great as that required for a stock-solution of only 1500 cubic feet. Therefore, the amount of stock-solution should be as small as possible.

Thirdly, *directly as the strength to be maintained.* Evidently, the amount of hyposulphite per ton of ore required to maintain a certain strength of solution, will depend upon what that strength is to be. For instance, each pound of hyposulphite will decompose to about the same extent, whether it is in a strong or weak solution. Therefore, in a given time, twice as many pounds will decompose in a 2 per cent. as in a 1 per cent. solution.

Fourthly, *inversely as the richness of the ore in metallic compounds dissolved by the solution.* In the dissolving of the metals of the ore by the leaching solutions, little or no hyposulphite is lost or destroyed, as the S_2O_3 of the sodium hyposulphite attaches itself to these metals only temporarily, and again returns to combination with sodium as sodium hyposulphite, when the metals are precipitated with sodium sulphide. Also, in the manufacture of sodium sulphide, hyposulphite of soda is formed as a by-product, which enters the leaching-solution when the sodium sulphide is used as a precipitant. As the amount of sodium sulphide used in precipitating a given volume of solution, and consequently the amount of hyposulphite entering that volume, varies as the metal contained in it, and as the ordinary solution, in passing through a charge, seldom or never dissolves such an amount of metal as to become saturated, but approaches nearer to that point in proportion to the richness of the ore in silver, copper and (to a certain extent) lead compounds soluble in the solution, the increase in strength of that volume of solution will depend upon the richness of the ore. If only a weak solution is to be maintained, this addition to the strength of the hyposulphite solution may be sufficient to make up for the weakening from all causes.

Fifthly, *inversely as the capacity of the works.* Although the strength of the solution makes a difference in the volume of solution required to leach a ton of ore, it makes little difference in the amount of stock-solution required for works of a given capacity. It is found in practice that, with ore-tanks and precipitating-tanks of the proper dimensions, the volume of stock-solution required is less in proportion for works of 100 tons per day than for those of 50 or 25 tons. Therefore, it is evident that more hyposulphite per ton of ore will be required to keep up a given strength in works of small, than in those of large, capacity.

Sixthly, *according to the order in which the charges are treated.* The volume of stock-solution and, consequently, the amount of hyposulphite required to maintain its strength, depends upon whether the charges are treated in rotation and with regularity, or so irregularly that the number of charges requiring solution varies considerably at different times. Regularity in the treatment of charges is of importance for other reasons also.

Seventhly, *with roasted ore, as the ore is acid or alkaline.* The difference in weight of chemicals required by acid and alkaline ores is very marked. Below is given the amount of hypo used in each case per ton of ore:

Acid Ores.	Simple Alkaline and Arsenical Alkaline Ores.
Ontario..........................5 lbs. Hypo.	Veta Grande............4.0 lbs. Hypo.
San Antonio...............7 " "	Sierra Grande...........2.6 " "
Sombrerete................5.3 " "	San Miguel... ....3.7 " "
San Bartolo...............3.0 " "	Daly......................1.5 " "
Average..........................5.06 lbs.	Average..........................2.9 lbs.

The difference is due principally to the fact that both extra and ordinary solutions may be used much weaker for simple alkaline than for acid-roasted ores.

2. *Blue-Stone.*—The amount of blue-stone used per ton of ore dedepends

First, *upon the condition of the ore, i.e., raw or roasted.* As shown in Table XIV., the average amount of blue-stone for raw ores is 2.63 lbs., and for roasted ores 5.6 lbs., per ton.

Secondly, in the case of roasted ores, upon *the acid or alkaline character of the ores.* For the four acid ores, Ontario, San Antonio, Sombrerete and San Bartolo, the amount of blue-stone per ton is 6.8 lbs. On the other hand, for the four alkaline ores, Lake Valley, San Miguel, Veta Grande and Daly, the average is only 4.4 lbs. per ton of ore.

Thirdly, *upon the method of using the extra-solution.* If the extra is preceded by the special extra-solution, the amount of blue-stone required per ton is considerably diminished.

3. *Caustic Soda and Sulphur.*—The weight of sulphur used in making sodium sulphide is always two-thirds of the weight of caustic soda. The weight of caustic soda and, therefore, of sulphur, depends:

First, *upon the richness of the ore in metallic compounds soluble in solution, but not soluble in water.* It is evident that the amount of precipitant required per ton will, other things being equal, vary as the amount of silver to be precipitated or directly as the value of the ore in silver. Of the forms of copper occurring naturally, only one, so far as known, namely, carbonate of copper, is soluble in solution, and at the same time insoluble in water. As it is not likely to occur in roasted ore, it is only of importance in this connection when it occurs in an ore treated raw. Dichloride of copper occurs to a small extent in chloridized ores and is soluble in the hyposulphite solution, but practically insoluble in water. Sulphate of lead occurs in roasted ore and sometimes in raw ores, and is slightly soluble in hyposulphite solution. The amount, in excess of 0.5 per cent., present in the ore is immaterial, as only a few pounds in any event would be dissolved during the leaching.

Secondly, *upon the amount of blue-stone used, and the mode of using it.* Evidently, any unnecessary increase in the amount of blue-stone used per ton should be avoided; for, with the exception of that part of the copper taking the place of the silver dissolved by the cuprous hyposulphite, and the small part expended in counteracting the effect of caustic soda or lime, the copper is precipitated by the sodium sulphide, increasing the consumption of the precipitant, and lowering the grade of the sulphides. As the amount of bluestone used per ton of ore varies as the ore is raw or roasted, and (with roasted ore) as it is acid or alkaline, the amount of caustic soda is, indirectly, dependent upon the same conditions, the amount used being less for raw than for roasted ores, and less for alkaline than for acid ores.

4. *Soda Ash.*—The amount of soda ash used per ton of ore depends

First, *upon the presence or absence of sulphate of lead.* The amount of sulphate of lead in an ore (when over 0.5 per cent.) makes, for reasons already mentioned, but little difference in the consumption of soda ash.

Secondly, *upon the amount of sulphate of lime in stock-solution.* If the stock-solution is saturated with sulphate of lime, its dissolving power for sulphate of lead is decreased, approximately, one-half. The presence of such a percentage of sulphate of lime, would, however, prevent the economical use of soda ash for the precipitating of the remaining half, and is undesirable in other ways. In treating lime ores, soda ash is not generally used.

Thirdly, *upon the temperature of the solution.* The amount of soda ash required, varies as the temperature of the solution, since lead sulphate is more soluble in hot than in cold solutions.

5. *Sulphuric Acid.*—Acid is used for two purposes, the acidifying of the first wash-water for roasted ores, and the neutralization of caustic impurities in the solutions. Its use is confined almost entirely to the treatment of roasted ores. The weight of acid used per ton depends:

First, *upon the state of the ore, i.e., raw or roasted.* As raw ores have no first wash-water to be precipitated, and no caustic alkali is present in raw ores, it can enter the solution only by means of the sodium sulphide. This amount is so small that it is generally neutralized by the extra-solution, so that, when extra-solution is used, no acid is required in the treatment of raw ore.

Secondly, *upon the acidity or alkalinity of the first wash-water.* In treating roasted ores, the silver is usually precipitated from the first wash-water by means of acid and old iron. The amount of acid required per ton varies according to the condition of the first wash-water. If it be alkaline, more acid must be used than if it were neutral or acid. The state of the first wash-water varies with the nature of the ore. Very base ores often produce such an acid wash-water, that no further addition of acid is necessary. But "free" ores usually produce a neutral or alkaline wash-water. Often, even when the wash-water is quite alkaline, the amount of silver dissolved from the ore by it will be more than one-fourth of the silver in the ore, and sometimes (as at the Sierra Grande) mere dilution with water will fail to precipitate it, nor will precipitation take place on organic substances within a reasonable time, nor on iron or copper until the addition of sufficient acid.

Thirdly, *upon the amount of silver extracted by the first wash-water.* Often, as at the Ontario with a Stetefeldt furnace in 1883, and at Sombrerete with a reverberatory furnace, the amount of silver dissolved by the first wash-water is so small that it is not precipitated, so that no acid is required.

In other cases, as at the Sierra Grande, the amount may be 30 per cent. of the value of the ore, and 2 pounds of acid may be required.

Fourthly, *upon the presence or absence of caustic lime in the ore.* The presence of caustic lime in the roasted ore usually doubles the amount of acid required for precipitating the wash-water, making it 2 pounds for such ores, as against 1 pound for ores producing a neutral wash-water.

Fifthly, *upon the mode of using the extra-solution.* Hydrate of copper is practically insoluble in the ordinary solution, so that the extra-solution acts as well as an acid as far as the neutralization of caustic alkali is concerned. But as the blue-stone is usually added to only a small portion of the ordinary solution, its effects are confined to that portion. So, in the case of alkaline roasted ore, where the amount of caustic alkali is always greater than for raw ores, it might seem that the blue-stone used would not be sufficient to neutralize all the caustic alkali, and that the use of acid might become necessary to the extent of ½ to 1 pound per ton. This, of course, would be the case in the treatment of roasted ore containing caustic lime, if it were not for the fact that the extra-solution is usually employed on such ores in such a way as to counteract entirely the effect of the caustic alkali. If the ordinary solution is used first, before the extra, the caustic alkali acts injuriously upon the silver compounds dissolved by it. For this reason, the extra-solution is used first, and in such a volume that, although weak, it protects the silver from the action of the caustic alkali.

Sixthly, *upon the mode of precipitating the first wash-water.* Of course, the amount of acid used will depend upon the mode of precipitating the first wash-water, whether by that method just described, or by dilution, or by sodium sulphide.

b. Description and Cost:

1. *Sodium Hyposulphite* is packed in barrels of two sizes, one containing about 112 pounds, and the other 550 to 600 pounds. It does not decompose in contact with the atmosphere. If imported in large lots from Europe, it costs in New York 1.8 cents per pound. If bought from dealers, the price asked is 2¼ cents per pound in New York, 2.5 cents in St. Louis, and 3 cents in San Francisco.

2. *Caustic Soda.*—Only a high grade should be ordered, of 70 to 77 per cent., on account of freight-charges. It is imported principally from England put up in sheet-iron drums, holding about 630 to 700 pounds, usually 640. It should not be left exposed to the air more than a day or two after being broken up, especially in hot weather, as it gradually decomposes on the surface, owing to absorption of carbonic acid. The price in new York is 2.425 cents per pound for 60 per cent. Hence, a 74 per cent. caustic soda would cost three cents per pound.

3. *Sodium Carbonate.*—Although any carbonate of soda can be used, the bicarbonate is less suitable than soda ash, on account of cost, and for other reasons. It is most economical to buy the form of soda ash known as "pure alkali" of 58 per cent. made by the Solvay or ammonia process. It is made in this country at Syracuse, New York, and costs 1.4 cents per pound at the works. This alkali, in the form of a white powder, is so pure that it contains 98.7 per cent. of sodium carbonate, the remainder being principally sodium chloride and sulphate, and 0.25 per cent. moisture. On account of the entire absence of the sodium sulphide and caustic soda (which are always formed in the ordinary soda ash), the solution for the precipitation of lead does not require purification. Both caustic soda and soda ash may be purchased of Wing and Evans, N. Y. city.

4. *Copper Sulphate or Blue-Stone.*—Price per pound 4¼ cents in New York.

5. *Sulphur.*—It is immaterial whether the sulphur is crude or refined, lump or flower of sulphur. The lump-sulphur requires to be broken up only so that it will pass a one-inch mesh screen. The native sulphur, occurring in large quantities in Utah, Nevada and other places, is suitable. The price in New York of refined sulphur

in rolls is 2⅜ cents per pound; flower, 2⅝ cents; Virginia rock, 2⅛ cents.

6. *Sulphuric Acid* of 66° B. is best transported in iron tanks, which hold about 1700 pounds. If shipped in this way, the freight is much less than in glass carboys. The cost of a sheet-iron tank is $6.50. Price in New York per pound 1.25 cents; in St. Louis and San Francisco, 1.75 cents. Shipped in car-loads the freight, for instance, from San Francisco to New Mexico, is only 1.67 cents per pound.

c. Comparison Between Sodium Sulphide and Calcium Sulphide.—If the lead is precipitated by itself by means of soda ash, sodium sulphide, and not calcium sulphide, should be used as a precipitant for the silver. But in any case sodium sulphide is far better as a precipitant than the calcium salt, for the following reasons:

First, the precipitating power of a pound of sulphur combined with sodium is nearly three times as great as when combined with calcium. For in the preparation of sodium sulphide the monoq-sulphide (Na_2S) is formed, but in making calcium sulphide, sufficient sulphur must be used to form the pentasulphide $Ca\,S_5$, as the lower sulphides are practically insoluble in water and therefore cannot be used.

Secondly, the solubility of calcium hydrate as stated in works on chemistry, is such that at 100° C., 1300 parts of water dissolves only one part of the hydrate, while the same amount of water at the same temperature dissolves 2600 parts of caustic soda. The solubility of caustic soda being thus over 3000 times as great as that of lime, it is not strange that the time required in making calcium sulphide is many times that required for sodium sulphide.

Thirdly, the amount of heat required in the preparation of sodium sulphide is only 2 per cent. or 3 per cent. of that for calcium sulphide. This is due not only to the fact that the union between sodium and sulphur takes place almost instantly, but also because there is a great rise in temperature due to this chemical action which causes the temperature of the mass to rise far above the boiling point of water.

Fourthly, the sodium sulphide is over twenty times as strong as the strongest calcium sulphide solution which can be made, that is, a given volume of the former will precipitate twenty times as much as the same volume of the latter. Hence there is a less dilution of the stock solution during precipitation.

Fifthly, as shown not only by the formula but in practice, the product obtained while using calcium sulphide is contaminated with over three times as much free sulphur as that obtained while using sodium sulphide. The amount of sulphur in the sulphides at Cusi and Sombrerete (using sodium sulphide) is so small that the difference in weight and value between dried and roasted sulphides is only 6 per cent., thus allowing the substitution of a mere steam drying for the roasting, during which latter operation some silver is always lost by dust and volatilization.

Sixthly, if calcium sulphide is used, the product is contaminated with calcium sulphate also (the sulphuric acid being derived from sodium sulphate always present in the lixiviation solution), which decreases the value of both dried and roasted sulphides and increases the weight and expense for shipment and refining.

Seventhly, the extent and cost of plant required for making calcium sulphide is much greater. For instance, at Cusi while making calcium sulphide, four iron tanks were in constant use, three of which were always boiling.

For the preparation of sodium sulphide only one of these tanks was used, and for three hours only (using steam 1 hour), once in two and one-half to three days. The above comparison illustrates also the difference in time, labor and heat required.

Eighthly, the continual use of calcium sulphide gradually converts a sodium hyposulphite solution into one of calcium hyposulphite, which has been claimed as a better solvent for gold than the former, although no one has made the statement from his own experience, or has produced any experiments or statistics. On the other hand it has been repeatedly demonstrated on the small scale, and at Cusi on the large scale, that the solvent energy of the two solutions for gold is just the same.

Ninthly, the calcium hyposulphite solution decomposes much quicker than sodium hyposulphite, and, as shown in actual practice, a greater addition of hyposulphite to the solution is required to maintain it at 1 per cent. when calcium than when sodium hyposulphite solution is used. As shown in many laboratory experiments at the Ontario, the average rate of decomposition of calcium hyposulphite solution was from 5 to 13 times as great as for sodium hyposulphite. In these tests, solutions of 2, 5 and 10 per cent. were exposed in soup plates at a temperature of 65 to 70 Fah. for seven days.

Tenthly, not only is the use of calcium hyposulphite disadvantageous as just described, if a strong solution is to be maintained, and, as shown further back, more expensive in time, labor and fuel, but also the actual cost of chemicals is greater per ton of ore treated. For instance, at Cusi, the comparison between a run of 1900 tons of ore, using sodium sulphide, and 1000 tons, using calcium sulphide, showed that the expense of chemicals alone required for calcium sulphide was 34 per cent. greater per ton of ore treated than for sodium sulphide. If the cost of the labor and heat required were included, the difference would be still greater. At Sombrerete the comparison was made by Mr. Watson under the most unfavorable conditions for sodium sulphide, the cost of a pound of caustic soda at that place being over 30 times as great as caustic lime, yet the result of the test showed the superiority of the sodium sulphide.

d. The Objects of the Use of Sodium Carbonate or Soda Ash, as a Precipitant for Lead are:—First, the obtaining of a precious metal product entirely free from lead, thus decreasing the weight of product and cost of transportation, and, on the other hand, increasing the grade, and, to some extent, the rate paid per ounce of silver. At the Ontario, using a 1½ per cent. stock-solution, the grade of the product was increased over 100 per cent. by the use of soda ash, and the weight decreased 52 per cent., thus effecting a great decrease in express charges. As the silver in sulphides is generally paid for at the rate of 97 to 99 per cent. less $100, more or less, for treatment, and, as this $100 is 1 per cent. of $10,000, but 2 per cent. of $5,000 sulphides, there is a saving of 1 per cent. if the sulphides assay the former instead of the latter.

Secondly, the obtaining of the lead product pure from other metals, and in a more compact and marketable shape than as sulphide. The separation of lead by means of soda ash is so complete that not the slightest trace is left in the solution after precipitating, and the solubility of carbonate of silver, gold and copper is such that no trace of these metals is found in the lead precipitate, except silver to the extent of one two hundredth part of 1 per cent.; at the same time no other metals can be precipitated with the lead carbonate, as all the compounds of zinc, manganese, nickel, cobalt, iron, etc. which are in the least soluble in a hyposulphite solution are perfectly soluble in the first wash-water and are removed by it. The precipitate of lead carbonate is heavy and compact, occupying only about one-quarter of the bulk of the same amount of lead in the form of a sulphide. The precipitate is pure white, and can be sold as it is for about the cost of precipitation or converted into litharge by exposure to a low heat.

Thirdly, the securing of a great saving in the amount and cost of caustic soda and sulphur used per ton of ore, and the ability to use solutions of any temperature and strength, which would otherwise be impracticable on account of the greatly increased amount of sodium sulphide which would be required to precipitate the greater amount of lead dissolved. The solubility of sulphate of lead in the hyposulphite solution increases directly as the strength and temperature. For this reason, the leaching by the old process is usually done with weak cold solutions, but, if the lead is precipitated by soda ash, the amount of lead dissolved is immaterial, and the solution can be used at any strength and temperature. Besides, the cost of precipitating a pound of lead by soda ash is much less than by means of sodium sulphide. At the Ontario, using a stock solution of 1½ per cent. the net saving due to the use of soda ash in chemicals alone, without taking into account the value of the carbonate of lead, was 28 cents per ton of ore treated. With stronger solutions, the saving by the use of soda ash is much greater, with a 10 per cent. stock-solution being about $2.80 per ton of ore. In short, the use of soda ash in the case of most ores secures a three-fold saving, viz:—in express charges, in rate paid for silver, and in cost of chemicals required for precipitating.

C.—*General Order of Applying the Wash-Waters and Solutions.*

The order of applying the solutions is treated in detail further on. In general, the order for wash-waters and solutions is as follows:

a. For Raw Ore.—No first wash-water. A strong extra-solution is used first, either with or without circulation, followed by the ordinary, and finally by wash-water. In rare instances, the ordinary solution is used first, followed by a strong extra, circulated, then by ordinary, and lastly by wash-water.

b. For Roasted Ore with Acid First Wash-Water.—First wash-water; ordinary solution; strong extra-solution, circulated; ordinary solution; second wash-water. No exception to this rule has so far been encountered.

c. For Roasted Ore with Alkaline First Wash-Water (Simple Alkaline Ore).—First wash-water; much weak extra-solution, not circulated; ordinary solution; second wash-water. This has been the invariable order so far.

d. For Alkaline Roasted Ore with Arsenical Compounds (*Alkaline Arsenical Ore*).—First wash-water; ordinary solution; strong extra-solution, allowed to stand; ordinary solution; second wash-water.

D.—*The Wash-Waters.*

a. The First Wash-Water.—The chief difference between the lixiviation of raw and roasted ore is in the use of the first wash-water on roasted ore to remove the soluble salts formed in roasting, which would contaminate both solutions and products, if not removed before the leaching solutions are applied.

1. *Manner of Using the First Wash-Water.*—After the surface of the charge of ore has been levelled, water is turned into the tank, either upon the top of the charge or beneath it under the filter.

The first of these methods is used if little silver is extracted by the first wash-water, or if, as at Yedras, the extraction of a considerable amount of silver in this way is not objectionable. The outlet of the leaching-tank remains closed until the tank is about full of water. The outlet is then opened, and the leaching with water continued until nearly all the soluble salts are removed, as indicated by testing the liquid with sodium sulphide. The water running upon the ore is then shut off, and what remains in the tank allowed to run out. When the water has drained out of the ore, the leaching-solution is turned upon the top of the charge.

If the second method is used, which is the case if the amount of silver extracted by the first wash-water is to be diminished as much as possible, or if water is scarce and must be used economically, the wash-water is introduced under the filter and allowed to rise slowly through the charge of ore until the tank is filled. The water is then turned off, and the outlet of the leaching-tank opened. When the surface of the water in the tank reaches the surface of the charge, the water is turned upon the surface, which is kept covered until the washing is finished. The rest of the manipulation of the first wash-water is the same as that already described.

The introduction of the leaching-solution upon a charge from which the wash-water has been drained has two advantages over turning the leaching-solution upon a charge at the time the descending wash-water disappears under the surface of the ore. The first advantage is that, in practice, the stock-solution loses less in strength by this method. The second is, that the watching for the appearance of the first silver-bearing solution at the outlet of the leaching-tank is avoided, as well as the danger of losing silver by carelessness at this point. For, in the method just advocated, the changing of the outlet-hose of the leaching tank from the wash-water trough or launder, to that leading to the precipitating tanks, is made at any time after the charge has drained, the leaching-solution not being used on the charge until such change has been made. For the method of using the first wash-water at Yedras, see page 490.

TABLE XIV.

AVERAGE AMOUNT OF CHEMICALS AND VOLUME OF WATER USED PER TON ON RAW AND ROASTED ORES AND TAILINGS TO WHICH THE RUSSELL PROCESS HAS BEEN APPLIED.

General Description of Material Treated.	Tailings or Ore.	Name and Location of Mine or Mill.	Pounds of Blue-Stone Per Ton.	Pounds of Hypo Per Ton.	Pounds of Caustic Soda Per Ton.	Pounds of Sulphur Per Ton.	Pounds of Lime Per Ton.	Pounds of Acid Per Ton.	Cu. Ft. of First Wash-Water Per Ton.	Cu. Ft. of Second Wash-Water Per Ton.	Total Cu. Ft. of Water Per Ton.	Total Weight of Chemicals Per Ton.
Previously Treated by Other Processes 1 to 3 Times.	Tailings.	Leeds, etc., Silver Reef, Utah..	1.8	3.76	3.00	3.00	......	0.25	......	10.5	10.5	11.8
	"	Bremen, Silver City, N. Mex...	4.1	3.00	1.75	1.1	......	0.25	......	9.0	9.0	10.2
	"	R. & E. & M. V., Pioche, Nev..	2.0	1.50	1.50	1.5	......	0.00	......	9.5	9.5	6.5
	"	Veta Grande, Parral, Mexico.	4.0	2.75	......	4.0	5.0	0.25	......	11.5	11.5	16.0
Raw Ore.	Ore.	———— Silver Reef, Utah	1.8	3.76	3.00	3.0	......	0.25	......	10.5	10.5	11.8
	"	Raymond & Ely, Pioche, Nev.	2.0	1.50	1.50	1.5	......	0.25	......	9.5	9.5	6.8
Roasted Material with Alkaline First Wash-Water.	Ore.	Sierra Grande Lake Valley, New Mexico......................	2.6	2.6	1.4	0.9	......	1.80	39.5	2.5	42.0	9.3
	"	Daly, Park City, Utah............	4.5	1.5	......	......	......	......	......	......	45.0	......
	"	San Miguel, Chihuahua, Mex.	4.7	3.7	4.0	2.7	......	0.00	37.5	2.5	40.0	15.1
	"	Yedras, Sinaloa, Mexico.........	10.0	0.5	6.0	4.0	......	0.00	32.0	14.0	46.0	20.5
	"	Veta Grande, etc., Parral, Mexico................................	6.0	4.0	4.5	3.0	......	0.00	41.5	2.5	44.0	17.5
	"	Ontario, 1887, Park City, Utah	6.4	......	......	......	......	......	......	......	122.0	......
	Tailings.	Sierra Grande, Lake Valley, New Mexico.......................	2.6	2.6	1.4	0.9	......	1.80	31.5	2.5	34.0	9.3
Roasted Material with Acid First Wash-Water.	Ore.	Ontario, Park City, Utah, 83-84	7.0	5.0	5.1	3.4	......	1.00	138.5	2.5	141.0	21.5
	"	Sombrerete, Zacatecas, Mex...	7.0	5.3	4.9	3.3	......	0.00	47.0	2.5	49.5	20.5
	"	San Antonio, Chihuahua, Mexico................................	8.3	7.0	7.75	5.0	......	0.00	52.5	2.5	55.0	28.0
	"	Chloride, Chloride, N. Mex....	4.3	5.5	5.25	3.5	......	3.80	38.5	2.5	41.0	22.1
	"	San Miguel, Chihuahua, Mex.	5.0	3.7	5.0	3.4	......	0.00	37.5	2.5	40.0	17.1
	"	San Bartolo, Mex....................	5.0	3.0	5.0	3.3	......	0.00	37.5	2.5	40.0	16.3

2. *Volume of First Wash-Water for Various Ores.*—When the first wash-water is used as just described, the volume of the second wash-water for roasted ore is much diminished and amounts to only 1¾ to 2 cubic feet per ton. Table XIV gives the amount of water required for various ores, including the first and second wash-water and the total amount. Except in the case of the Ontario ore, the amount of the first wash-water varies from 12 to 53 cubic feet per ton of ore, the average being about 40. At the Sierra Grande, Sombrerete, and Chloride mills, the water is introduced from above. The saving in the amount of the first wash-water by introducing it below the charge is 25 to 40 per cent.

3. *Advantage of Charging Roasted Ore Cold into the Leaching-Tanks.*—As shown in Table XV, the amount of silver dissolved from roasted ore by the first wash-water varies with the temperature of the ore at the time the water is turned upon it. The term "hot dry ore" refers to ore which has a temperature of about 150° to 250° F.; "cold dry ore" to ore at a temperature of not more than 120° F. Of the silver dissolved and extracted from the ore by the first wash-water, at Cusi, only a portion actually left the leaching-tank. As shown in Table XVI, about 90 per cent. of all the silver extracted by the first wash-water is extracted during the first fifteen minutes' leaching, provided the water is introduced on top of the charge, as was the case in Tables XV and XVI.

As more or less water, comparatively free from salts, may exist under the filter at the time the first wash-water is used, being left there by the last wash-water of a previous charge, and as chloride of silver is precipitated from an aqueous solution by dilution with water, a large portion of the dissolved silver may be precipitated from the first wash-water, under the filter, and therefore not actually leave the leaching-tank until again dissolved by the hyposulphite solution subsequently used. For Cusi ore, even when 25 to 30 per cent. of the silver in the ore had been dissolved and extracted from the ore by the first wash-water, only 3 to 9 per cent. actually left the leaching-tank, while 2 to 6 per cent. left the tank when the ore was charged cold. Consequently, the difference between the results of hot and cold charging are not practically as great as would appear from Table XV. As far as the fineness of the wash-water precipitate and (except in some parts of Mexico) the cost of the precipitating is concerned, the amount of silver passing out of the leaching-tank in the first wash-water is of little importance. For instance, at Lake Valley all the silver dissolved by the first wash-water, amounting to 30 per cent. of the value of the ore, actually passed out of the leaching-tanks in the first wash-water; but

TABLE XV.

SILVER DISSOLVED FROM COLD AND HOT ROASTED ORES BY THE FIRST WASH-WATER.

No. of Charges Experimented on.	Value of Ore in oz. Silver Per Ton.	Per cent. of Salts Soluble in Water.	Calculated Value in oz. Silver Per Ton.	Value after Washing in Mill. Oz. Silver Per Ton.	Ounces Extracted by First Wash-Water.	Per cent. Extracted by First Wash-Water.
HOT DRY ORE.						
5	34.6	14.8	40.6	28.3	12.3	30.3
33	34.2	14.1	38.8	27.6	11.2	28.8
COLD DRY ORE.						
19	37.9	15.0	44.6	39.4	5.2	11.6

all of it was saved except 0.07 ounce per ton of ore. The only objection to allowing a large amount of the silver in the ore to pass out of the leaching-tank in the first wash-water is, that this is the only liquid, used in leaching roasted ores, which is thrown away or allowed to pass out of the mill; and, consequently, incomplete precipitation would cause a loss which could not afterwards be remedied; while no loss would result from incomplete precipitation of the regular leaching solutions, as any silver not precipitated at one time would be precipitated subsequently—the same solution remaining in constant use.

4. *Amount of Silver Extracted by the First Wash-Water from Various Roasted Ores.*—Table XVII gives the amount of silver extracted from the first wash-water. The actual loss by unprecipitated silver in the first wash-water, was only 0.07 ounce per ton for Sierra Grande ore, although 3 to 4 ounces were dissolved by it and passed out of the leaching-tank. The reason why more silver is dissolved by the first wash-water from some ores than from others is not known. There seems to be no connection between the amount dissolved and the percentage of salt, or the kind of furnace used. Even while using 17 per cent. of salt, at the Ontario, on ore roasted in a Stetefeldt furnace, only 0.25 per cent. of the silver was extracted by the first wash-water, while 30 per cent. was extracted from the Sierra

Grande ore roasted with 7 per cent. of salt in a Howell furnace, and only a trace of silver was extracted by the first wash-water from Sombrerete ore roasted with 10 per cent. of salt in a reverberatory.

TABLE XVI.

DISTRIBUTION OF THE DISSOLVED SILVER IN THE FIRST WASH-WATER FROM CUSI ORE.

Number of Charges Experimented on.	Value of Charges in oz. Silver Per Ton.	Depth of Charges in Inches.	Per cent. of Salts Soluble in Water.	Calculated Ore Value in oz. Silver Per Ton.	Value after Washing in Mill.	Oz. Extracted by First Wash-Water.	Per cent. Extracted by First Wash-Water.	Silver Contained in 1000 c.c. at end of ¼ Hour.	Silver Contained in 1000 c.c. at end of One Hour.	Silver Contained in 1000 c.c. at end of Two Hours.	Silver Contained in 1000 c.c. at end of Three Hours.	Silver Contained in 1000 c.c. at end of Four Hours.	Silver Contained in 1000 c.c. at end of Five Hours.	Silver Per 1000 c.c. at end of First Thirty Inches of Water.	Total Time of Washing.
1	29.2	24	14	33.9	25.2	8.7		.180	.0064	.0032	.0028	.0008	None.		19 Hours.
1	29.0	24	14	33.7	26.5	7.2		.108	.0034	.0020	.0009	Trace	"		15 "
1	37.6	24	15	44.2	23.6	20.6		.270	.0060	.0018	.0008	.0008	.0004		16 "
1	37.6	24	16	44.8	29.0	15.8		.266	.0400	.0008	Trace	Trace	None.		18 "
1	39.6	24	15	46.6	37.4	9.2		.150	.0116	Trace	"	None.	"		15 "
1											.0008				
1											.0014				
1											.0028				
1											.0009				
38														.0038	

NOTE.—The Silver in the Wash-Water is expressed in grammes per 1000 c.c.

5. *Distribution of the Dissolved Silver in the First Wash-Water.*—Table XVI shows the amount of silver in grammes per 1000 c.c. of the first wash-water—all the wash-water being applied on top of the charge, and not under the filter. (The use of the latter method not only reduces the amount of silver dissolved, but causes it to be more evenly distributed through the wash-water.) The table indicates that at least 90 per cent. of the dissolved silver passes out of the charge during the first 15 minutes of washing.

Statistics of the action of the first wash-water on Ontario ore in 1883–1884 are given below. The wash-water was introduced first from below the charge until the tank was full, after which all the leaching was downward. No silver was precipitated on top of the ore, as sometimes happens. All the wash-water coming from the ore was divided into three parts, and each part was precipitated by itself. Nearly 77 per cent. of the silver dissolved by the first wash-water was found in the second part. If the leaching had been downward from the beginning, probably 75 to 88 per cent. of the dissolved silver would have been found in the first third of the wash-water.

TABLE XVII.

PROPORTION OF SILVER EXTRACTED FROM ROASTED ORES BY THE FIRST WASH-WATER: ITS VALUE, AND THE COST OF PRECIPITATING IT.

Name of Mine.	Wash-Water Introduced from Above or Below.	Mode of Precipitating.	Per cent. of Ore-Value Obtained from Wash-Water.	Value of the Precipitate in Oz. Per Ton.	Cost of Precipitating Per Ton of Ore.
1883.					
Ontario	From Below.	Not Precipitated.	0.25	Not Saved.	Not Precipitated.
Lake Valley	From Above.	Acid and Old Iron	29.	9000 oz.	9 cts.
Sombrerete	" "	Not Precipitated.	Trace.	Not Saved.	Not Precipitated.
San Bartolo	From Below.	Sodium Sulphide.	3.3	1600 oz.	12 cts.
San Antonio	" "	" "	6.4	1400 oz.	15 cts.
San Miguel	" "	" "	7.8	2400 oz.	15 cts.
Yedras	" "	" "	2.0	13934 oz.	6 cts.
Chloride		Acid and Old Iron			16 cts.

The wash-water being acid, 72 per cent. of the dissolved silver was obtained with the cement copper, by simply hanging old iron in the wash-water precipitating-tanks. The addition of ½ to 1 pound of acid to the wash-water might have been necessary, if the wash-water had contained much silver. But in this case it was not necessary, as the amount of silver dissolved by the first wash-water was only a fraction of an ounce per ton of ore, and all of it was regained except about one-twentieth of an ounce per ton.

The difference between the Ontario and Lake Valley ores, as to the percentage of silver dissolved by the the first wash-water, has already been remarked. If the actual ounces per ton of ore extracted in this way are compared, the difference is not quite as great, but is still about 6 or 7 times as great for Lake Valley, as for Ontario.

The precipitate from the first wash-water (Ontario) by means of old iron, contained about 73 per cent. copper, the amount of copper obtained per ton of ore being about 3¼ lbs., although the raw ore contained only about ⅓ of 1 per cent. or 7 lbs. per ton. The amount of silver and copper in the first wash-water at the Ontario in 1883–1884, was as follows:

Copper in the precipitate by iron, 73 per cent.; weight of copper obtained per ton of ore, 3.25 lbs.; grammes silver contained in 1st 40 cubic feet of wash-water, 6.0; grammes silver contained in 2d 40 cubic feet of wash-water, 31.0; grammes silver contained in 3d

40 cubic feet of wash-water, 2.6; proportion of ore-value leached out in 1st wash-water, 0.85 per cent.; proportion of wash-water silver obtained with the copper precipitated on iron, 72 per cent.

6. *Precipitation of the Silver from the First Wash-Water.*—Besides the method of precipitating by means of acid and old iron, the silver may also be precipitated from the first wash-water by means of sodium sulphide, or sometimes by dilution. The method by sodium sulphide is probably the better for Mexico of the first two, and is quicker, as the precipitation of the silver by iron requires about 12 hours, and heating of the wash-water in the precipitating-tanks. The average value of the various precipitates from different ores by the two methods would probably not differ much. But the method by acid and iron seems to be the most thorough, except when the wash-water contains such a very small amount of base metals, that (as at Yedras) all the base metals can be economically precipitated along with the silver. This very rarely happens. In most cases, when sodium sulphide is used, only enough is added to precipitate all the silver and part of the copper, without precipitating much of the remaining metals; but sometimes a small amount of silver remains unprecipitated. A point in favor of the method by acid and iron is, the extremely small bulk of the precipitate compared with that produced by sodium sulphide. This method consists in suspending old iron in the wash-water tanks and heating the water to about 175° F. (more or less), after acidifying it with 1 to 2 pounds of acid per ton of ore. The method of "dilution" consists in simply diluting the wash-water with sufficient pure water to precipitate the chloride of silver. It is not a safe method, however, for some ores, and it requires considerable water.

7. *Value of the Precipitates from the First Wash-Water.*—Table XVII gives the value of the precipitates, by both methods, from the wash-water of various roasted ores. The value of the precipitate of either method depends mostly upon the amount of copper present in the first wash-water, as it is practically impossible to avoid precipitating copper by either method. As a rule, the precipitate from ores with an alkaline wash-water is much the higher in value per ton.

b. The Second Wash-Water and Mode of Using it.—The volume of the second wash-water per ton for raw ores is always the same as the volume required to saturate the wet charges. It is used to restore the volume of the stock-solution, which would otherwise be diminished, for each charge treated, by an amount equal to the stock-solution remaining in the ore when the charge is drained. This amount is about 5½ cubic feet per ton of ore.

For raw ores there are two methods of applying the second wash-water. In the first the ordinary solution is left standing level with the surface of the charge, the outlet of the leaching-tank being stopped. A measured number of inches of water, equal to the amount required to saturate the wet charge, is then run upon the charge. The outlet of the leaching-tank is then opened and the leaching continues until the water is at the surface of the charge. The water remaining in the charge is then allowed to run to waste, but not into the precipitating tanks for the leaching-solutions. By the second method employed, if water is scarce and must be economically used, the leaching-solution is allowed to drain out of the charge, and is then followed by water to the extent of about 5½ cubic feet per ton of ore, all of which is allowed to pass into the stock-solution. The volumes of second wash-water given in Table XIV for raw ores and tailings are those required when the first method is used. The average amount required for the six raw ores and tailings by the first method is 10.0 cubic feet per ton of ore. The use of the second method would reduce this to about 5½ cubic feet per ton.

The mode of using the second wash-water for roasted ores is the same as for raw ores, if the descending first wash-water was followed by solution at the time the surface of the first wash-water reached the surface of the ore. But if the first wash-water has been drained from the charge before the solution is used, then likewise the solution must be drained from the charge before the second wash-water is used. In this case the second wash-water is then applied by suddenly flooding the surface of the charge with about one-half inch of water, which descends through the charge and is allowed to pass into the stock-solution.

c. The Total Volume of Water Required for both Raw and Roasted Ores.—Table XIV gives the total volume of water required per ton of ore for both raw and roasted ores. For all the roasted ores, including Ontario, the average is 50 cubic feet. The average volume for all the raw ores and tailings is 10½ cubic feet, if the first, and 5½ if the second method is used.

E.—*The Leaching-Solutions.*

a. Volume, Strength, Temperature, Preparation, and Description of the Ordinary or Stock-Solution, the Amount of Chemicals Required and the Rate of Decomposition.—The advantages of keeping the volume of stock-solution as small as possible are, firstly, less heating to preserve a given temperature, if the solutions are to be used warm; secondly, less decomposition per day or per ton, and consequently less consumption of chemicals. The conditions upon which the volume of the stock-solution depend are: first, the state of the ore, *i. e.*, raw or roasted, raw ore usually requiring less solution than roasted ore; secondly, the capacity of the works per day, the volume of the stock-solution being less in proportion for works of 100 tons capacity than for those of 50 or 25 tons, as already shown;* thirdly, the size of the charges, the volume of stock-solution being less in proportion for 50-ton than for 10-ton charges; fourthly, the regularity of the leaching, irregularity of using the solution necessitating a greater volume.

The strength of stock-solution used at various works for roasted ores is as follows (expressed in percentages): Ontario, 1.4; Daly, 1.4; San Antonio, 0.8; Sombrerete, 1.0; Yedras, 1.8; Lake Valley, 0.8; Chloride, 1.1; San Bartolo, 0.7.

In starting works the stock-solution is generally made up with 1.5 per cent. of hyposulphite, the amount required for 3500 cubic feet being 3281 lbs.; for 2000 feet, 1875 lbs.; and for 1500 cubic feet, 1406 lbs.

The following statement gives the volume of the ordinary or stock-solution used by the Russell Process per ton of ore.

	Volumes of Stock-Solution Cubic Feet.
Silver Reef tailings,	100
Bremen† "	9
Veta Grande "	70
Silver Reef raw ore	137
Sierra Grande roasted ore,	80
Yedras " "	100
Ontario, 1887–8 " "	196
Daly " "	180
Veta Grande " "	110
Ontario, 83–84, " "	190
Sombrerete " "	90
San Antonio " "	100
Chloride " "	102
San Bartolo " "	80
San Miguel " "	70

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* For leaching works, as now constructed, the volume of stock-solutions may be put as follows: For 100 tons' capacity, 3500 cubic feet; for 50 tons, 2000 cubic feet; for 25 tons, 1500 cubic feet.

† The small volume of solution used on the Bremen tailings is due largely to its strength (3 per cent.) in hyposulphite. Owing to the extreme slowness of the

The strength of these stock solutions in hyposulphite and the volumes of the first and last wash-water have already been given. The volumes and strengths of the extras are given elsewhere. This table, therefore, completes the description of the liquids used in the Russell process.

For simple alkaline ores, the solutions have all, so far, been used cold, and for raw ores and acid-roasted ores, at a temperature of 110° to 150° F. In the treatment of acid-roasted ores, even if the rest of the stock-solution has been used cold, that portion immediately following the extra-solution, and amounting to 18 to 24 cubic feet per ton, should be warm. In such cases, therefore, the use of an additional storage-tank for hot solution is necessary.

The ordinary or stock-solution is made by dissolving a proper quantity of sodium hyposulphite in a measured volume of water in the storage tanks. Generally the stock solution is made up of a strength of 1½ per cent. hyposulphite (94 lbs. for each 100 cubic feet of water).

After a little practice a stock-solution of one half the strength may often be used. The causes of the deterioration of the stock-solution have already been given. In making up the extra-solution, for each pound of blue-stone used, one pound of hyposulphite of soda is temporarily destroyed, but is regained when the metals are precipitated by sodium sulphide. After the most advantageous concentration of the solution has been determined by experience, it should be maintained by constant addition of sodium hyposulphite: this addition being made to the extra-solution for reasons already given. The strength of the solution in hyposulphite after it has been in use cannot be approximately estimated by its specific gravity for reasons that will soon become apparent. Hence a chemical test is the only one upon which reliance can be placed. The Iodine method has already been given and it is the quickest and most reliable. Another method is to ascertain how much silver chloride will be dissolved by a certain number of cubic centimeters of the solution, but this is somewhat unreliable, except in experienced hands, as two hyposulphites of silver may be formed, one of which forms imme-

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leaching, a strong solution was used to shorten the time as much as possible. In nearly all the cases, except that just mentioned, less solution could probably have been used if it had been desirable to do so, but such a course would have made necessary a preliminary assay of the tailings before removal, which would have caused delay. In nearly all cases the tailings are removed before the assay of the tailings is known, a course which is nearly always allowable if the leaching is systematically conducted.

diately, while the other forms only slowly and contains twice as much silver as the other.

The original stock-solution gradually accumulates impurities, such as sodium chloride and sulphate and others, to slight extent, the source of which is evident. The first of these is perfectly harmless, the second diminishes the solvent energy of the solution to a very slight extent, the effect of which is simply to correspondingly increase (by 2 or 3 per cent.) the volume of stock-solution required per ton of ore. Calcium salts are introduced if gypsum is present in the raw ore, or if caustic lime or calcium sulphate exists in roasted ore. In case the lead is precipitated by soda some of the calcium is also precipitated, as it would be difficult to discriminate between the two, even if the action of carbonate of soda on the one did not begin until the other had been entirely precipitated. The concentration of the solution in these salts, however, does not go on indefinitely, but reaches a maximum and then remains stationary, so that a new stock-solution is never required, the same original stock-solution being used year after year, although very little of the original chemicals may remain, on account of the losses and dilution, by the wash-water, which either precedes and follows or merely follows the stock-solution. The amount of impurities in the stock-solution at the Ontario, after two years' use, was such that the addition of barium chloride to 1.000 c. c. of the stock-solution yielded a precipitate weighing 31.4 gr., after drying and 30.2 gr. after treating with HCl and igniting. The specific gravity of the solution was about 4° Beaumé.

There is, however, a change in the condition of the stock-solution that is of great importance, namely, its assuming a caustic reaction. Silver ores only that carry a comparatively large percentage of antimonial and arsenical compounds may be benefited by treatment with a caustic solution. Although the presence of caustic alkali, even in small quantities, is usually very injurious, the remedy is most simple and inexpensive, being accomplished either by the addition of sulphuric acid or blue-stone to the stock-solution. If the former is used the amount required will vary from one third to one pound per ton of ore, and the cost from one to four cents per ton. The effect of the extra-solution in neutralizing caustic impurities has already been described.

One of the causes of the weakening of the stock-solution is the decomposition of hyposulphite by taking up oxygen from the air. At the Ontario in '83–'84 leaching in the mill was twice interrupted and the rate of decomposition of the mill stock-solution determined. The strength of the solution was in each case 1½ to 2 per cent. In the first case, the solution remained unused in the storage-tank for twenty-six days, during which it decomposed five and a half per cent. In the other case, it remained unused thirty-five days, and the decomposition was 17.4 per cent. In this case, the depth of the solution was about 1¼ times its diameter. The extent of surface exposed in proportion to the depth of the liquid has considerable influence on the rate of decomposition. For instance, the rate of decomposition in volumes whose diameter was seven times the depth was 1.8 times (for sodium hyposulphite) and 2.4 times (for calcium hyposulphite) the rate of decomposition for volumes whose depth and diameter were equal.

b. Volume, Strength, Temperature and Preparation of the Extra-Solution, the Amount of Chemicals Required, and the Efficiency of the Extra-Solution in the Extraction of Silver.

1. *Volume and Depth of the Extra-Solution.*—The volume of the extra-solution and its depth, in proportion to the depth of the ore, varies with the state of the ore, whether raw or roasted, and the method of leaching employed. The volume of the extra-solution varies with the volume required to saturate a ton of ore. This, for raw ores or dry tailings, ranges from 6½ to 9 cubic feet per ton. For roasted ores, except the simple alkaline, the variation is 10 to 14 cubic feet per ton. For raw ores and tailings, if the extra-solution is used on the dry ore, the volume of the extra is the same as the volume to saturate. For all roasted ores, except the simple alkaline, and for raw ores or tailings when the extra is not used first, the volume is 6 to 10 per cent. less than that of saturation, for a reason explained further on. For simple alkaline roasted ore, it is four to six times the volume required to saturate.

The relation of the depth of the extra-solution to the depth of a wet charge, *i.e.*, the depth after leaching has commenced, is as follows: For raw ores and raw tailings, the depth of the extra-solution varies from 36 to 42 per cent. of the depth of the charge of ore; for all roasted ores, except the simple alkaline, it varies from 34 to 46 per cent. of the depth of ore; for simple alkaline roasted ores, it varies from 1½ to 3 times the depth of ore.

2. *Reasons for Adding all the Hyposulphite to the Extra instead of the Ordinary Solution.*—The first reason is that, in treating raw ores, the extra-solution is nearly always used first. As it is

not preceded by any wash-water, but is used on dry ore, no loss of chemicals occurs at this part of the leaching.

Secondly, In treating "simple alkaline" roasted ore, the extraction of a large part of the silver must be done by hyposulphite of soda or lime in the presence of cuprous hyposulphite. But when blue-stone is added to a hyposulphite solution, it absorbs twice its weight of hyposulphite of soda or lime into the cuprous hyposulphite. So, if only that amount of hyposulphite were present, it would all be absorbed in this way, and the extra-solution would fail in this, the most important of its two uses, on simple alkaline ore. The work of the extra, in the use of it here recommended, is more quickly done and at less expense of blue-stone, in proportion (within certain limits) to the strength of the solution in hyposulphite.

Thirdly, When the metals are precipitated from the extra-solution, it becomes converted into ordinary hyposulphite solution, so that the addition of hyposulphite to the extra, is, in all cases, eventually the same as adding it to the ordinary.

Fourthly, As the extra-solution, at the time of making it, must have twice as much hyposulphite as the blue-stone added to it, it is found much more economical to add to the small volume of extra-solution sufficient hyposulphite to raise it to the proper strength (say, 1.75 per cent.) than to maintain the whole volume of stock-solution at that strength.

3. *Strength and Preparation of the Extra-Solutions for Raw, Acid Roasted, and Alkaline Arsenical Roasted Ores.*—The strength of the extra-solutions used on the above ores, varies from 0.7 to 1.1 per cent. in blue-stone, and from 1.5 to 2.3 per cent. in hyposulphite. This percentage of hyposulphite is the percentage existing in the stock-solution at the time of making the extra, plus that added with the blue-stone.

The reason why the extra-solution for roasted ores can be made up on the charge of ore is that it is prevented from sinking into the charge during its preparation by the solution already in the charge, which stands level with the surface of the ore—the outlet of the leaching-tank being stopped while the extra-solution is being prepared. But, in the treatment of raw ores, the extra-solution is generally used on dry ore, which would absorb the extra-solution during the preparation of it, and thus prevent the obtaining of a homogeneous solution.

4. *Strength and Preparation of the Extra-Solution for Simple Alkaline Roasted Ore.*—No rule can be laid down for the preparation

of these extra-solutions, as the volume required may vary from four to six times the amount required to saturate, and the strength from 0.1 to 0.33 per cent. in blue-stone. Besides, for some ores of this class, all or a portion of the blue-stone may be used, as at Lake Valley, in the first wash-water.

5. *Amount of Chemicals Required, and Efficiency of the Extra-Solution in the Extraction of Silver.*—Table XVIII gives the amount of chemicals required for strong, medium and weak extra-solutions, the amount of hyposulphite given in the table varying not only as the amount of blue-stone used, but also as the strength of the stock-solution. The table is for Cusi roasted ore, which has an acid reaction, weighs about 70 lbs. per cubic foot, requires 11 to 12 cubic feet to saturate it, and is treated in charges of about 9 tons. Of course, an ore, requiring only 7 to 10 cubic feet to saturate it, would require less chemicals for the various strengths of stock-solution.

A noticeable fact is the difference in amount of blue-stone and hyposulphite required in the treatment of simple alkaline roasted ore, as compared with acid roasted ore. In Table XIV, the six acid roasted ores require an average of 6.1 lbs. blue-stone, and 4.9 lbs. hyposulphite, while the four simple alkaline ores (Sierra Grande, San Miguel, Veta Grande and Daly) require only 4.5 lbs. of blue-stone and 2.9 lbs. of hyposulphite.

The average maximum extracting-power, in silver per ton of ore of a single medium or strong extra-solution in Table XVIII, containing 6½ to 8 lbs. of blue-stone per ton of ore, is 12.2 oz. per ton. This was determined in twenty-one experiments at Cusi on San Antonio and San Bartolo ore. For these tests charges were selected, having tailings of 20 to 40 oz. per ton after the ordinary solution had done its utmost work and had failed to lower the value of the tailings further. The extra-solution was then used on these charges—sometimes the strong and sometimes the medium of those described in Table XVIII. Of course, on charges yielding tailings of 30 to 40 oz. per ton by the ordinary solution several extra-solutions had to be used to reduce these tailings to 6 or 8 oz. per ton. The results showed that the use of 6½ to 8 lbs. blue-stone per ton increased the average mill-extraction by 12.2 oz. per ton of ore, which is equivalent to 1.68 troy ounces of silver for each avoirdupois pound of blue-stone. At the Ontario, the additional extraction caused by the use of two charges of extra-solution of 9½ lbs. each was an extraction of 39.1 per cent. of 88 oz. ore, more than could be extracted by the ordinary solution in either the mill or the assay-office. This is an additional extraction of 34.4 oz. per ton or 17.2 oz. for each extra-solution with 9½ lbs. blue-stone, or an extraction of 1.81 oz. of silver per pound of blue-stone. Both the Cusi and Ontario were acid roasted ores.

TABLE XVIII.

TABLE OF CHEMICALS REQUIRED FOR EXTRA-SOLUTIONS OF VARIOUS STRENGTHS FOR CUSI ROASTED ORE: WEIGHTS OF CHEMICALS PER TON AND PER CHARGE OF NINE TONS.

	For a Stock-Solution Containing $\frac{6}{10}$ of One Per cent. Hypo.	For a Stock-Solution Containing $\frac{8}{10}$ of One Per cent. Hypo.	For a Stock-Solution Containing One Per cent. Hypo.	For a Stock-Solution Containing 1.2 Per cent. Hypo.	
	1	2	3	4	
Blue-Stone Per Charge.	46 Lbs.	46 Lbs.	46 Lbs.	46 Lbs.	For a Weak Extra - Solution to contain $\frac{7}{10}$ of One Per cent. in Blue-Stone.
Hyposulphite " "	46 "	33 "	20 "	7 "	
Blue-Stone Per Ton........	5 "	5 "	5 "	5 "	
Hyposulphite " "	5 "	3¾ "	2½ "	1 "	
	5	6	7	8	
Blue-Stone Per Charge.	59 Lbs.	59 Lbs.	59 Lbs.	59 Lbs.	For a Medium Extra - Solution to contain $\frac{9}{10}$ of One Per cent. in Blue-Stone.
Hyposulphite " "	84 "	72 "	59 "	46 "	
Blue-Stone Per Ton........	6½ "	6½ "	6½ "	6½ "	
Hyposulphite " "	9¼ "	8 "	6½ "	5 "	
	9	10	11	12	
Blue-Stone Per Charge.	72 Lbs.	72 Lbs.	72 Lbs.	72 Lbs.	For a Strong Extra - Solution to contain 1.1 Per cent. in Blue-Stone.
Hyposulphite " "	111 "	99 "	85 "	72 "	
Blue-Stone Per Ton........	8 "	8 "	8 "	8 "	
Hyposulphite " "	12 "	11 "	9 "	8 "	

Only for a Roasted Ore which requires twelve cubic feet to saturate.
For Cost of these various Extra-Solutions see Table xxiv.

At the Ontario, when allowed to act on tailings from the ordinary solution, but very high in value, or in other words when allowed to show its utmost efficiency, the amount of silver extracted by the extra-solution was in value about fifteen times its entire cost, including chemicals, preparation and application, and the precipitation of the metals from it. When used on simple alkaline roasted ore, the extra-solution may have a still greater efficiency in extracting silver, due to its double action hereafter referred to.

6. *General Rules to be Observed in the Preparation and Use of the Extra-Solution.*—If the extra-solution is made at a temperature of over 120°, it should not be prepared until just before using. Even cold extra-solutions should not be allowed to stand unused more than six or seven hours. For all ores except the simple alkaline roasted ores, the total weight of hyposulphite used, including that existing in the stock-solution, should be at least twice the weight of blue-stone. For simple alkaline roasted ore, the weight of hyposulphite used, including that existing in the stock-solution, should be at least three times that of the blue-stone.

As already stated, the extra solution is made up either in a separate tank, or on the surface of the charge. The first method is always used for raw ores, and for roasted ores also, if the leaching works are to be run to their utmost capacity. In this method about one-third of the total volume of solution required for the extra-solution is run into the storage tank in which the extra is to be prepared. The proper quantity of chemicals is then placed in the chemical box, and the remaining two-thirds of the stock-solution allowed to run in through this box. If this is not sufficient to dissolve all the chemicals, some of the extra-solution in the tank is circulated through the box by means of a No. 4 syphon pump until they are dissolved. The reason for running one-third of the stock-solution into the tank before dissolving the chemicals is to avoid the formation of the yellow precipitate (cuprous hyposulphite), at the bottom of the tank, which would be dissolved only after considerable stirring. In any case, however, the extra-solution is stirred just before it is used. In making up the extra-solution on the surface of the ore, for all except simple alkaline ores, the chemicals are placed in the chemical box and the whole volume of stock-solution required allowed to pass through them. The extra-solution is stirred just before it is used, and whatever yellow precipitate may be left on the ore will be dissolved during the circulation. But in making up the extra-solution for simple alkaline ores, none of this precipitate forms, since it does not appear if, as in the treatment of them, the amount of blue-stone is considerably less than one-half the amount of hyposulphite. In making up the extra-solution for these ores, the manipulation is the same, except that four to six extras are made up, each containing much less chemicals, which are not circulated, and which are applied directly after the first wash-water, instead of after the ordinary or stock-solution, and cold instead of warm. Practical experience has demonstrated that if roasted ores containing an appreciable percentage of caustic lime are treated with ordinary solutions after the first wash-water, as must be done if the old process of leaching is used, the result may be very disastrous, and that the damage done is only partially made good by the subsequent leaching with the extra-solution. But if the extra-solution is applied at once, the cuprous hyposulphite neutralizes and counteracts the deleterious effect of the caustic lime, and leaves the silver in a soluble form. Before the extra-solution follows the first wash-water, it has been found beneficial to impregnate the ore with a small quantity of a strong copper sulphate solution. It can easily be seen that the first portion of the extra-solution, replacing the wash-water, must become more or less diluted by contact with water. This dilution may be so considerable that an insufficient quantity of copper is present to protect the silver from the deleterious influence of the caustic lime. Therefore, after the leaching with water is finished, and just before the extra-solution is to be used, about one or two pounds of blue stone per ton of ore is dissolved in an amount of water equal to from one to three inches on the surface of the ore. As soon as this sinks to the surface of the ore, the extra-solution is used. But for some ores all the blue-stone may be used in the first wash-water. This may be done when the ore contains such a compound (usually carbonate of an alkali or alkali earth) as will precipitate all or a portion of the copper out of the wash-water, in the ore, in such a state that it will be wholly or partially dissolved by the ordinary or stock-solution afterwards used, thus making the extra-solution in the ore. The blue-stone, when thus used, is added to about one charge, (or volume to saturate) of wash-water, which may be either passed through the charge only once or circulated. If desirable, it may be followed by more wash-water, in order to wash out any copper not precipitated in the ore.

c. Differences Between the Solutions.

1. *Difference Between the Ordinary and the Extra-Solution and Between the Extra-Solutions.*—No extra-solutions are kept in stock. They are made up fresh for each charge. The extra-solution for simple alkaline ores is used like the ordinary, *i.e.* without circulating, but differs from it, in containing a fraction of 1 per cent. of cuprous hyposulphite, and in losing its efficiency as an extra when precipitated. For all other ores, both raw and roasted, the extra-solution differs from the ordinary in containing from six-tenths of 1 per cent. to 1.2 per cent. of cuprous hyposulphite, in being much less in vol-

ume, and in being circulated, and also in losing its efficiency as an extra when precipitated.

The extra-solutions differ from each other as follows: For simple alkaline roasted ore, the extra is used cold, weak, in comparatively large volume, is not circulated, and is used immediately after the first wash-water; or part or all of the blue-stone may be used in the first wash-water. For all other ores, both raw and roasted, the extra-solution is generally used stronger, warm, as small in volume as possible, after the ordinary solution, is circulated, is followed by hot ordinary, and no blue-stone is used in the first wash-water.

2. *Reasons for the above Differences.*—In the treatment of simple alkaline roasted ore, the extra-solution has a double use; First, in dissolving and extracting the silver and gold compounds, not soluble in the ordinary solution, and secondly, in protecting the silver (and gold) from the injurious effect of the caustic alkali in the ore, while they are being dissolved by the hyposulphite and are passing out of the charge. The first is the regular use of the extra-solution, but in its second use it is of much more value. In this use of the extra, the extraction of that part of the silver soluble in hyposulphite only, is done under the protection, so to speak, of the weak extra-solution. Hence the blue-stone must be used at the beginning, and its use continued until, as shown by experience, three to six charges of solution have been used. Its use is then no longer necessary. It should be remembered that a " charge of solution " is an amount of solution equal to the amount required to saturate the "charge of ore."

3. *Reasons for Circulating and for Small Volume of Extra.*—For those ores on which the extra is circulated, it must be of a certain strength, say, 0.6 per cent., to accomplish its object. Therefore, the smaller the volume of extra per ton, the less will be the cost of chemicals per ton. The reasons for circulating the extra but not the ordinary, are two: in the first place, the ordinary solution, in passing through the precipitating-tanks, renews its strength as an ordinary, and is ready to be used again. There is, therefore, no reason why it should be circulated before being precipitated. The extra-solution, on the contrary, in passing through the precipitating-tanks, loses its efficiency as an extra, being converted into ordinary solution. Therefore, the whole effect of the extra should be obtained before allowing it to be precipitated, which can only be done by circulating it. In the second place, the volume of the extra is the smallest possible that will saturate the ore, and, on this account, in passing through the charge only once, all particles of the ore might

not be brought in contact with it. On the other hand, the volume of the ordinary solution is practically unlimited, the storage-tanks being always kept supplied from the precipitating-tanks at no additional cost for chemicals.

d. Manipulation of the Extra- and Special Extra-Solutions.—When the extra-solution is to be circulated, the outlet leading to the precipitating tanks is stopped when the surface of the extra-solution has sunk to one-half inch above the surface of the ore. Steam is then turned into the syphon-pump, and the solution returned on to the surface of the ore continuously from four to six hours. For simple alkaline roasted ore the extra is allowed to pass through the ore as if it were the ordinary.

The extra-solution for raw and for acid roasted ore should be followed by hot ordinary. Also, in the treatment of simple alkaline ores, all the blue-stone may sometimes be used in the first wash-water by adding 2 to 4 pounds of blue-stone per ton of ore to about one-half or two-thirds of a charge of wash-water, after the washing is about finished, and allowing it to pass through the ore like the rest of the wash-water. It may sometimes be followed by more wash-water; but usually it immediately precedes the ordinary. In the treatment of alkaline arsenical ores, ordinary solution is used before the extra, until nearly all the silver has been extracted which is soluble in the ordinary solution. The extra-solution is then used, being always allowed to stand from 6 to 12 hours in the charge, although it may then be circulated. It may be used cold or hot, but is always followed by hot ordinary, when used cold. The formation of a yellow precipitate, during the preparation of the extra-solution, which fails to dissolve by the time the extra is ready for use, indicates an insufficient amount of hyposulphite.

Acid ores and simple alkaline ores (according to experience so far) should not be mixed in the same charge. In case such a mixture should occur, the charge should be treated first as if it were alkaline ore, and then as if it were acid.

The special extra-solution is merely an extra-solution which has already been used upon one charge. When used upon a second charge, it is always applied immediately before the regular extra. The main object in using it is, that the regular extra-solution may be diluted by a solution of its own kind, as it is sure to become diluted to some extent with whatever solution is in the charge. Another reason is, that the efficiency of an extra-solution is not usually exhausted by using it on one charge. Its use saves chem-

TABLE XIX.

ORDERS IN WHICH THE LEACHING-SOLUTIONS MAY BE APPLIED IN THE TREATMENT OF RAW AND ROASTED ORES.

1	8-10 cold ordinary	8-10 cold ordinarys	1 special extra	Cold ex. circulated 3-5 hrs.	3-5 hot ordinarys	Raw Ore and Acid Roasted Ore.
2		6-8 hot ordinarys	1 special extra	Hot ex. circulated 2-4 hrs.	4-6 hot ordinarys (1st stands 4-6 hours)	
3		8-10 cold ordinarys	2-3 hot ordinarys	Hot ex. circulated 2-4 hrs.	3-4 hot ordinarys	
4			10-12 cold ordinarys	Cold ex. circulated 3-5 hrs.	3-5 hot ordinarys	
5			6-8 hot ordinarys	Hot ex. circulated 2-4 hrs.	4-6 hot ordinarys (1st stands 4-6 hours)	
6		6-8 hot ordinarys	Hot extra circulated 2-4 hours	The same stands 6-8 hours.	4-6 hot ordinarys (1st stands 4-6 hours)	
7		1 special extra	Hot extra circulated 2-4 hours	The same stands 6-8 hours.	3-5 hot ordinarys	
8			4-6 weak cold extras	Strong cold ex. stands 10-12 hours	10-12 cold ordinarys	Simple Alkaline Roasted Ore.
9			1½ to 4 lbs. Blue-Stone per ton in 1st Wash-Water	4-6 weak cold extras	12-14 cold ordinarys	
10			1½ to 4 lbs. Blue-Stone per ton in 1st Wash-Water	4-6 weak cold extras	10-12 hot ordinarys	
11			All the Blue-Stone in 1st W. W.	1 cold ord. stands 3-6 hours	12-14 cold ordinarys	
12			All the Blue-Stone in 1st W. W.	1 cold ord. stands 3-6 hours	10-12 hot ordinarys.	
13			All the Blue-Stone in 1st W. W.	14-16 cold ordinarys		
14		8-10 cold ordinarys	3-4 hot ordinarys	Hot,extra stands 10-12 hrs.	4-5 hot ordinarys	Alkaline Arsenical Roasted Ore.
15		8-10 cold ordinarys	1 special extra	Cold extra stands 10-12 hrs.	4-6 hot ordinarys	
16		8-10 cold ordinarys	1 special extra	Hot extra stands 10-12 hrs.	3-5 hot ordinarys	
17		6-8 hot ordinarys	1 special extra	Hot extra stands 10-12 hrs.	4-6 hot ordinarys (1st stands)	
18			6-8 hot ordinarys	Hot extra stands 10-12 hrs.	4-6 hot ordinarys (1st stands 4-6 hours)	
19		1½ to 4 lbs. Blue-Stone per ton in 1st Wash-Water	8-10 cold ordinarys	Cold extra stands 12 hours.	4-6 hot ordinarys	Alkaline and Acid Mixed.
20			4-6 weak cold extras	Strong hot ex. cir. 4-6 hrs.	10-12 hot ordinarys (1st stands 4-6 hours)	
21	1½ to 4 lbs. Blue-Stone per ton in 1st Wash-Water	4-6 weak cold extras	Cold extra circulated 3-5 hours.	The same stands 6-8 hours.	12-14 cold ordinarys	Alkaline & Alkaline Arsenical Mixed.
22		1½ to 4 lbs. Blue-Stone per ton in 1st Wash-Water	4-6 weak cold extras	Strong hot ex. stands 10-12 hours	10-12 hot ordinarys.	
23	1½ to 4 lbs. Blue-Stone per ton in 1st Wash-Water	4-6 weak cold extras	2-4 hot ordinarys	Hot extra stands 10-12 hrs.	5-6 hot ordinarys	Raw Ore.
24				Cold ex. circulated 4-6 hrs.	12-14 cold ordinarys	
25				Hot ex. circulated 4-6 hrs.	10-12 hot ordinarys	
26				Cold extra stands 10-12 hrs.	12-14 cold ordinarys	
27				Hot extra stands 10-12 hrs.	10-12 hot ordinarys (1st stands 4-6 hours)	
28			Hot extra circulated 4-6 hours.	The same stands 10-12 hrs.	10-12 hot ordinarys	

icals and causes a higher extraction of the silver, and, for the same amount of blue-stone per ton, the product is of a higher grade if the extra-solution is used as a "special extra."

Also the use of the "special extra" secures a more even extraction of the silver from all parts of the charge of ore. For instance, at Cusi, on San Bartolo ore, the tailings from the lowest six inches of the tub averaged 7.0 oz. On the other hand, while using the special extra, similar samples averaged 5.3 oz.

It may be used on any ore, in the treatment of which the extra-solution would follow the ordinary. It is not circulated, but precedes the extra as if it were ordinary solution, and passes from the charge on which it has been used directly to the precipitating-tanks.

e. Order of Application, Volumes and Temperatures of the Leaching-Solutions.—Table XIX gives the order of applying the leaching-ing-solutions on each of the different classes of ore. The first and second wash-waters are omitted from the table.

The "ordinarys" and "extras" referred to are each "charges" of those solutions, the definition of which term has already been given. The volumes of solutions given have proved sufficient for all ores so far treated by the Russell process.

After once selecting the method which gives the best result for the ore from any particular mine, no change in method will usually be required for the ore from that mine or district, unless the wash-water from these ores changes from alkaline to acid or the reverse. For custom-mills treating several different kinds of ore, both acid and alkaline, each one of two or even three methods may have to be used. According to some of the methods given in Table XIX the extra-solution is *required* to stand in the ore, while in others it is unnecessary, but, so far, no injurious effects have been observed due to the standing of the extra-solution *in* any kind of ore. But it should not be allowed to stand more than a few hours, except in the ore.

Table XX gives the results of treating an acid roasted ore as if it were alkaline-arsenical. At the time of making this comparison, some of the leaching-tanks at the Cusi mill had not been fitted with syphon-pumps or ejectors with which to circulate the extra-solution. It was therefore necessary to obtain the best results possible by allowing the extra to stand in the ore. The average difference in mill-extraction by the two methods was 6.9 per cent.

TABLE XX.

EFFECT OF CIRCULATING THE EXTRA-SOLUTION AS COMPARED WITH NOT CIRCULATING (ACID-ROASTED ORE) AT CUSI MILL ON CUSI ORE.

"Circulated" or "Not Circulated."	Number of Charges compared.	Value of Ore in oz. Silver Per Ton.	Per cent. Extraction by Ordin'ry in Ass'y Office.	Per cent. Extraction by Extra in Ass'y Office.	Per cent. Extraction in Mill.
NO CIRCULATION. The extra stood in the ore for twelve hours.	46	34.8	77.1	89.0	78.2
CIRCULATION. The extra was circulated for four hours.	96	34.8	76.6	87.6	85.1

NOTE.—The above is a roasted ore with acid first wash-water.

Table XXI, on the other hand, shows the effect on an alkaline-arsenical ore of circulating the extra instead of allowing it to stand, the difference in mill extraction by the two methods being 4.9 per cent. of the value of the ore.

TABLE XXI.

EFFECT OF ALLOWING THE EXTRA TO STAND TWELVE HOURS IN THE ORE WITHOUT CIRCULATION AS COMPARED WITH CIRCULATING THE EXTRA (ALKALINE-ARSENICAL ROASTED ORE) AT YEDRAS MILL ON YEDRAS ORE.

"Circulated" or "Allowed to Stand Twelve Hours without Circulating."	Value of Ore in oz. Silver Per Ton.	Per cent. Extraction by Ordinary in Assay Office.	Per cent. Extraction by Extra in Assay Office.	Per cent. Extraction in Mill.
CIRCULATION. The extra solution was circulated for four hours.	57.4	77.0	83.3	77.7
NO CIRCULATION. The extra solution stood in the ore for twelve hours.	63.5	70.6	83.1	82.6

NOTE.—The above is a roasted ore with an alkaline wash-water, but with much arsenic in the ore.

F. *Preparation and Manipulation of the Precipitants, Sodium Sulphide and Soda-Ash, and the Amount of Each Used.*

a. Preparation of the Sodium Carbonate (or Soda-Ash) Solution.—In the preparation of the sodium carbonate solution, if the ammonia or Solvay process soda-ash is used, it is dissolved in an iron tank, in a portion of the hyposulphite stock-solution, in preference to water, so that, in precipitating the lead, the strength of the stock-solution is not diminished by dilution. If an aqueous solution of the soda-ash be used, it should be made as strong as possible, and kept heated. If ammonia process soda-ash cannot be obtained, and the ash used contains, as is usually the case, some sodium sulphide, the removal of this sodium sulphide becomes necessary for the following reasons: The sulphur of the sodium sulphide, having a greater affinity for silver than for lead, will precipitate silver out of the leaching-solution when the soda-ash is used to precipitate the lead. Consequently, the precipitate of carbonate of lead will be contaminated with sulphide of silver to the extent of about 100 ounces of silver per ton of lead carbonate, instead of 1.5 ounces per ton as would be the case if the soda-ash were free from sodium sulphide. The removal of this sodium sulphide from the soda-ash, is, however, a very simple matter. As carbonate of copper is perfectly soluble in a hyposulphite solution, while sulphide of copper is not, it is evident that, if the impure soda-ash is dissolved in a hyposulphite solution (such as the stock-solution), and then an amount of blue-stone (or other compound of copper soluble in the hyposulphite solution) corresponding to the amount of sodium sulphide be added to the hyposulphite solution of soda-ash, all the sodium sulphide will be neutralized by the formation of sulphide of copper which sinks to the bottom as a black precipitate. The clear solution is then drawn off and is ready for use. Generally about one pound of copper sulphate is sufficient to purify 100 pounds of soda-ash. If caustic soda is present in the soda-ash, it may be removed by boiling it with a little sulphur, and then adding copper sulphate. The presence of sodium sulphide in the ash may be detected by adding a little of an aqueous solution of the ash to a hyposulphite solution containing silver, a dark discoloration or the formation of precipitate indicating the presence of sodium sulphide.

b. Preparation of the Sodium Sulphide Solution.—In order to obtain a sodium sulphide of maximum precipitating power, the following method should be strictly followed: The whole contents of a drum of caustic soda, 600 to 700 pounds, are broken up into lumps which should not exceed 6 to 8 pounds in weight, and placed in the cast-iron tank with false bottom of lead, hereafter described. About 2 or 3 cubic feet of water are added, and dry steam turned on as full as possible, the tank being covered to avoid spattering of the lye. If sufficient head of steam is used, the dissolving of the caustic soda will be accomplished in ½ to ¾ hours. It would not do to attempt to dissolve it slowly, since too much steam would be condensed. The total volume when the mass is dissolved should be such as to measure not more than as many cubic feet as there were pounds of caustic soda divided by 62.5. In other words, one cubic foot of lye should hold at least 62.5 pounds of caustic soda. As soon as the caustic soda is dissolved, and while the lye is at a temperature of not less than 200° F., which will be the case if the above directions are followed, the sulphur is added, a shovelful at a time. If it is lump-sulphur, it is not necessary to crush the lumps unless most of them will not pass a screen of 1-inch mesh. The sulphur must be added slowly, as the mass boils violently, and swells to several times its original volume while the chemical action is taking place. Should the lye have been either much less concentrated or of lower temperature than stated above, the solution of the sulphur will not be complete.

The sodium sulphide should be made in a cast-iron tank of not more than eight or nine feet area. If through mismanagement the lye is not as concentrated or as hot as above described, and the mass does not boil violently when the sulphur is added, it (with the sulphur) must be boiled with steam for about two hours. The precipitating-coefficient of the finished product is, however, thereby diminished. The sodium sulphide solution obtained as above described is so concentrated, that it would solidify on cooling. It is therefore diluted with hyposulphite stock-solution in preference to water, after being transferred to the storage-tanks by means of a short piece of wooden trough not shown in the drawings. The dilution is carried to such an extent that the total volume measures as many cubic feet as there have been used pounds of caustic soda, divided by 12. If the sodium sulphide still shows a tendency to crystallize, the dilution may be carried still further. The silver-precipitating coefficients for caustic soda and sulphur thus prepared are about 2 to 2.3 for silver and 3.5 to 4 for sulphur.

After the sodium sulphide in the cast-iron tank has ceased boiling, it may be so thick that it will not easily run out, in which case it is diluted by solution or water in the cast-iron tank to the extent of about its own volume.

c. Manipulation of the Sodium Carbonate (*or Soda-Ash*) *Solution.*—The precipitation of lead with sodium carbonate is very easily effected, and the precipitate settles quickly, leaving the solution clear. The precipitate is also more compact than the sulphide. Care should be taken not to add an excess of soda (although the presence of this reagent in the hyposulphite solution is not in the least injurious to the extraction of silver in lixiviation, even if the solution was saturated with it), as it can easily be seen, that, should it become necessary to neutralize any caustic soda in the solution, the consumption of sulphuric acid would be increased by neutralizing sodium carbonate also.

The end of the reaction, or the time at which the addition of the sodium carbonate should be stopped, is best determined by taking some of the solution from the precipitating-tanks in a test-tube, and adding a few drops of an aqueous solution of sodium phosphate, which gives a more delicate reaction than sodium carbonate. After the lead carbonate has settled in the precipitating-tanks, which occupies about half an hour, the clear solution is decanted or pumped by an ejector into the silver-precipitating-tanks. After the final addition of soda-ash solution, the whole contents of the precipitating-tanks are stirred with a hard-wood oar for about half a minute. Also, during the addition of the soda-ash the solution should be stirred. The lead carbonate is removed about once a week, and pressed in a filter-press. The lead carbonate is very pure, unless lime is present in the solution, in which case it will be contaminated to some extent with the lime. In practice, about one pound of soda is consumed in precipitating one pound of lead, or one and one-third pounds of lead carbonate.

d. Manipulation of the Sodium Sulphide Solution.—During the addition of sodium sulphide to the leaching solution, in the precipitating-tanks, and at the end of the operation the solution should be well stirred. The more violent the stirring, the quicker and more compact will be the settling of the precipitate and the clearer the solution. It is hardly necessary to say that the sodium sulphide should not be used in excess. In case this has been done by inadvertence, more fresh silver-bearing solution must be added. Hence, in filling the precipitating-tanks, space should be left for the addition of a few more inches of solution if necessary. The addition of an excess of sodium sulphide is indicated by a smell of hydrogen sulphide, a turbidity of the solution, and a discoloration of the sides of the tank, or the floor around it. If it should happen that an over-precipitated solution has been run on a charge of ore, it is only necessary to follow it with an extra-solution which will dissolve the sulphide of silver precipitated in the ore.

If the solution to be precipitated contains lime, an insoluble calcium monosulphide may be precipitated, and may not only contaminate the precious sulphides, but greatly increase the amount of sodium sulphide, even to the extent of doubling it. But the light yellow calcium precipitate does not appear until all the sulphides of silver, gold and copper have fallen out. In order to determine when the metals have been precipitated, some of the solution is taken in a tumbler or test-tube, and soda-ash solution is added to it in excess. This will precipitate the lime, but leave the copper and silver, if any, in solution. The clear solution from the test-tube is then decanted off into another test-tube, and sodium sulphide is added to it. The reaction is then not obscured by the yellow calcium monosulphide, and a discoloration, turbidity, or precipitate will show that all the silver and copper have not been precipitated. After the sulphides have settled, the clear solution is decanted by means of the surface-discharger into the pump-sump.

TABLE XXII.

DISTRIBUTION OF THE LEAD, SILVER, AND COPPER, IN THE HYPOSULPHITE SOLUTION OF THE OLD OR ORDINARY LEACHING PROCESS AT THE ONTARIO MILL (ONTARIO ORE).

No. of Experiments Made.	Weight of Ore in each Exper.	Strength of Stock-Solution.	Div. into How Many Parts.	Amt. Soda Ash used per ton.	Amt. Caustic Soda used pr ton.	Amt. Sulphur used per ton.	c.c. used for the 1st Twelfth.	c.c. used for the 2d Twelfth.	c.c. used for the 3d Twelfth.	c.c. used for the 4th Twelfth.	c.c. used for the 5th Twelfth.	c.c. used for the 6th Twelfth.	c.c. used for the 7th Twelfth.	c.c. used for the 8th Twelfth.	c.c. used for the 9th Twelfth.	c. c. used for the 10th Twelfth.	c. c. used for the 11th Twelfth.	c. c. used for the 12th Twelfth.	
136	2-3 tons.	1.5 per cent.	12	5 lbs.	5.1 lbs.	3.4 lbs.	340	481	473	477	430	370	354	326	335	374	390	380	Soda-Ash Precipitating-Solution.
							400	1.000	510	377	257	188	124	90	64	45	41	33	Sodium Sulphide Precipitating-Solution.

e. Miscellaneous.—Table XXII shows the comparative amounts of sodium sulphide and soda-ash required at different stages of the leaching (and thus, incidentally, the distribution of the lead, silver, and copper, in the ordinary solution), in treating Ontario roasted ore. The two tanks for precipitating lead were of small size, holding only about 32 cubic feet. There were also two of the same size for precipitating the silver and copper. Each tank was filled and emptied six times during a test. The number of c.c. of soda-ash solution required to precipitate all the lead from each one of the tanks, each time, was noted, and also the number of c.c. of sodium sulphide used each time to precipitate the silver and copper. The Table shows that the amount of lead dissolved by a given volume of solution remained about the same at different periods of the leaching. But the amount of silver and copper dissolved by the second one-twelfth of the solution was nearly double that dissolved by any other one-twelfth, and the amount dissolved by the first one-sixth was 44.7 per cent. of the whole.

In these experiments the leaching with solution was commenced about the middle of the day, stopped during the night, and continued until about 11 A.M. the next day. During the night the solution remained standing in the ore. But the solution which had remained in the ore all night contained no more metal than if it had passed rapidly through. The strength of sodium sulphide solution was much weaker than now used. The strength of the cold stock-solution used was 0.8 to 2.0 per cent., the value of ore 70 to 110 oz., the depth of the charge when dry about 22 inches, and after leaching 16.5 inches. The extra-solution was not used until after the twelfth charge. It was noticed that the volume of solution containing the extra-solution contained nearly as much lead as any other volume. The carbonate of lead produced was pure white, perfectly free from any other metal, except that it contained about 1.5 oz. of silver per ton of lead carbonate. The value of the dried sulphides was from 7500 to 12,000 oz. per ton.

TABLE XXIII.

VALUE OF THE PRECIPITATE OR "SULPHIDES" FROM THE LEACHING-SOLUTIONS AND THE COST OF PRECIPITATING THE SAME WHILE USING THE RUSSELL PROCESS.

Name of Mine or Mill.	Ore or Tailings.	Sulphides "Roasted" or "Dried."	Ounces Silver per ton in the Sulphides.	Ounces Gold per ton in the Sulphides.	Cost of Precipitating per ton of Ore.
Leeds	Tailings.	Dried.	3000	Trace.	34 cents.
Bremen	"	Roasted.	8000	None.	14 "
R. & E. & M. V	"	Dried.	....	....	17 "
Veta Grande	"	Roasted.	5400	20.	30 "
Silver Reef	Raw Ore.	Dried.	3300	Trace.	38 "
Raymond & Ely	"	Dried.	11600	13.2	20 "
Sierra Grande	Roasted Ore.	Roasted.	8000	None.	11 "
San Miguel	"	Dried.	8980	3.2	47 "
Yedras	"	Roasted.	15000	Not Assayed.	86 "
Veta Grande	"	Roasted.	8400	39.0	54 "
Sierra Grande	Roasted Tailings,	Roasted.	7200	None.	11 "
Ontario	Roasted Ore.	Dried.	10500	Not Assayed.	39 "
Sombrerete	"	Roasted.	7400	36.0	55 "
San Antonio	"	Dried.	8200	4.3	88 "
Chloride	"	....	....	....	41 "
San Miguel	"	Dried.	7000	2.8	64 "
San Bartolo	"	Roasted.	8790	8.5	58 "

Table XXIII gives the value in silver and gold of the precipitates or sulphides produced by the Russell process in various mills. The sulphides produced from the Leeds and other Silver Reef ores are low in grade, on account of the presence in the ore of carbonate of copper which is dissolved by the hyposulphite solution.

G.—*Mechanical Treatment of the Products.*

The precipitated sulphides are removed from the three precipitating-tanks for solution about once in three days, and the precipitate from the wash-water tanks at the same time if the wash-water has been precipitated by sodium sulphide, otherwise, usually only once a week. To remove the sulphides, the clear solution is decanted by means of the surface discharge as low as possible. The gate or valve on the rear of the precipitating-tanks is then opened, and the sulphides allowed to run out into the trough or launder leading to the tank for the storage of sulphides. The sulphides are permitted to settle in this tank, and the clear solution again decanted, so as to save running it through the press. The storage-tank, of the size given in the drawing, will hold the sulphides for one week's run, or about 700 tons of average ore. The sulphides may be made more compact and reduced in bulk about two-thirds by using a jet of steam in the storage-tank. But, if this is done, the sulphides should be allowed to cool before pressing, for reasons explained further on. When the sulphides are to be pressed, all the valves between the storage-tank for sulphides and the filter-press (except the steam valves on top of the pressure-tank) are opened, so that the sulphides run freely through the pressure-tank into the press. In this way, two-thirds of the pressing is accomplished by gravity. While the sulphides are running into the press, the contents of the tank are occasionally stirred. When the press is so full that the solution comes only slowly from it, the valve between the sulphide storage-tank and the iron press-tank is closed, and as soon as the liquid in the pressure-tank has sunk below the level of the small valve on the side of the pressure-tank, three or four inches below the top, steam at full pressure is turned into the pressure-tank, until again the clear solution comes only slowly from the filter-press. It is best not to have the pressure-tank quite full when the steam is turned into it, as the solution would be heated by it, and a hot solution has a bad effect on the filter cloths, so that they do not last more than one-half as long as they otherwise would. When the press is full, which is indicated by the pressure gauge, as well as by the slow running of the clear solution from the press, the steam is turned off and allowed to escape from the pressure-tank, and the valve between the pressure-tank and the press is closed. The trough under the press, for conducting off the clear solution, is then removed, and a large iron pan about three feet wide, five inches deep and long enough to reach the whole length of the press between the legs, and having handles, is placed underneath the press. The press is then emptied by sliding the plates apart, and inserting a wooden (not a metal) paddle behind the cakes which causes them to drop into the pan. If distance-rings are used, they are removed from press by the handles and the cakes pushed out. When the press is first opened, a small amount of solution may run out, so that it is best to have the press set in a low tank, about two inches deep, and with a drain-pipe in one corner leading down to the sump under the press.

An 18-inch press has a capacity without distance-rings corresponding to about 250 pounds of dry sulphides per hour. The distance-rings are used when a greater capacity is desired, but the cakes will then contain a greater percentage of moisture.

The appearance of sulphides pressed at 150 pounds per inch does not indicate the presence of more than 10 per cent. moisture, but they have in reality about 38 to 42 per cent. If these sulphides are dried in a reverberatory furnace, 0.5 per cent is likely to be lost in dust, and considerable labor is involved.

The best method of drying is by steam, in which there is no loss, and which requires only a nominal amount of labor. After being dried, which occupies two or three days, the sulphides are sampled on the iron floor beside the steam dryer. The lead carbonate precipitate is pressed and dried in the same manner as the sulphides.

H.—*General Remarks.*

While the first wash-water is draining from the leaching-tank, preparatory to using the leaching-solutions, cracks are likely to be formed, due to a slight settling of the ore. Before the leaching-solutions are used these cracks should be filled up by smoothing over the surface of the charge with a wooden hoe. Also before and during the use of the extra-solution, particular care should be used in filling up any cracks, especially around the sides of the charge; otherwise the effect of the extra-solution may be decreased.

For maintaining a partial vacuum under the filter and thus accelerating the leaching and circulating the extra-solution, the Aller lead-lined siphon-pump is the best. It is applicable in all cases except on such extremely fine material as that at Silver City, another case of which is not likely to be found. The various sizes used are Nos. 2, 3 and 4 for leaching-tanks of 12, 14 and 16 feet diameter, respectively. If the leaching-rate without a vacuum is 6 to 8 inches an hour, the use of a siphon-pump will usually double it. If only one-quarter to one-half an inch per hour, the use of a siphon-pump will often raise it to 6 or 8 inches. If a more positive vacuum is necessary, as was the case at Silver City, a No. 4 or 5 Knowles' pump is used, one being sufficient for a mill of 50 tons' capacity.

The lowest six inches of the charge of ore will sometimes show a less thorough extraction of the silver than the uppermost six inches, or than the average of the whole charge. Therefore, in starting new works, or if the value of the tailings is not satisfactory, the lowest six inches of the tailings should be sampled, and this should also be done whenever two methods of leaching are to be compared. If the tailings are sluiced out the amount of water required for sluicing is twelve to eighteen feet per ton of ore. But if roasted ore is being treated no additional water is required, as the last half of the first wash-water can be used for this purpose, being conducted down to and stored in a suitable tank near the fire-pump.

III. Leaching Plant.

A.—*General Remarks.*

In Mr. Stetefeldt's paper on the Russell process in its practical application (*Transactions*, vol. xv.) a short description was given of the plant required for the Russell process. This, however, in view of the many improvements made in the year and a half since the date of that paper is not sufficient, especially as no plans and only one detailed drawing were given. The working drawings here given comprise the best features of five leaching plants, built especially for the process, and of the Sombrerete plant now being constructed. With the exception of two minor details on the precipitating-tanks there is nothing given but what has stood the test of actual practical experience. The plant is entirely independent and complete as it stands and may be situated wherever convenient, without reference to the position of the crushing plant for raw ores, or for the crushing chlorodizing plant for roasting ores, except as regards the easy transportation of the raw or roasted ore between the two plants.

Consequently when either one of the two above-mentioned plants for the preparation of the ore already exists in connection with amalgamation, the change to the Russell process can be made by the erection of the plant herein described without stopping or removing the amalgamation plant. The plant has its own boiler, and no engine is required.

The following is an index to the parts of the plant as given in Plates I and II. The details are given in Figs. 1 to 3, and Figs. 6 to 19.

Index to Parts of Plant.

A. Leaching-tanks.
B. Solution-precipitating-tank.
C. Wash-water.
D. Solution-pump-sump.
E. Storage-tank for sulphides.
F. Filter-press-sump.
G. Storage-tanks for solution.
H. Storage-tank for hot solution, or for preparation of extra.
I. Cast-iron tank for preparation of sodium sulphide.
J. Sheet-iron tanks for storing sodium sulphide.
K. Iron pressure-tank.
L. Filter-press for sulphides.
M. Steam-dryer for sulphides.
N. Boiler.
O. Fire pump for sluicing.
P. Solution-pump.
Q. Main ore-track to leaching tanks.
R. Side-tracks to leaching-tanks.
S. Side-tracks in chemical room.
T. Grade-line between boiler and steam-dryer.
U. Grade-line through sulphide storage-tank.

a. Tailing-shute (to cars).
b. Sluice-gate for tailings.
c. Sluice-trough for tailings.
d. Box for chemicals for extra-solution.
e. Ore-car.
f. Ore-car truck.
g. Chemical car.
h. Special-extra trough over leaching-tanks.
i. Solution-trough over leaching-tanks.
j. Solution-trough to solution precipitating-tanks.
k. Wash-water trough to wash-water precipitating-tanks.
l. Waste-water trough.
m. Trough to conduct sulphides to sulphide storage-tank.
n. Trough to conduct solution to solution-pump-sump.
o. Waste wash-water trough.
p. Solution-pipe to leaching-tanks.
q. Hot-solution or extra-solution pipe to leaching-tanks.
r. Water-pipe to leaching-tanks.
s. Hose for circulating extra.
t. Hose from leaching-tank to siphon-pump.
u. Hose from leaching-tank to triple launder.
v. Hose from siphon-pump to triple launder.
w. Siphon-pump.
x. Tailing-car.
y. Hose-connections for sluicing.
z. Waste-water pipe from triple launder.

a′. Pipes from triple launder to solution-precipitating-tanks.
b′. Pipes from triple launder to wash-water precipitating-tanks.
c′. Valve or "gate" for sulphides.
d′. Surface-discharge for precipitating-tanks.
e′. Sodium-sulphide pipe.
f′. Pipe for precipitated wash-water.
g′. Connection between solution-pump and sump.
h′. Box and strainer around outlet.
i′. Drip-pipe from solution-pump.
j′. Pipe from pump to storage-tanks.
k′. Connection between sulphide-storage-tank and pressure-tank.
l′. Connection between pressure-tank and press.
m′. Siphon-pump in filter-press sump.
n′. Pipe for returning solution to solution-sump.
o′. Small windlass for steam-dryer.
p′. Boxes over storage-tanks to prevent agitation of the solution.
q′. Boxes and strainers around outlets.
r′. Lead heating-coils in storage-tanks.
s′. Drip from lead coils.
t′. Triple connection between the three storage-tanks.
u′. Main steam-pipe.
v′. Water-pipe from fire-pump for sluicing.
w′. Storage-room for caustic soda and sulphur.
x′. Storage-room for blue-stone and hyposulphite.
y′. Trap-door to filter-press sump.
z′. Gauge-valve on side of pressure-tank.

a′′. Solution-pipe to sodium sulphide storage-tanks.
b′′. Pipe for drawing off sediment from pump-sump.
c′′. Pipe for drawing off sediment from solution storage-tanks.
d′′. Siphon-pump for circulating solution through chemical-box in extra solution-tank.

TABLE XXIIIa.

CAPACITY OF THE PLANT PER DAY FOR VARIOUS ORES AND TAILINGS TREATED BY THE RUSSELL PROCESS.

Name of Ore.	Kind of Material.	Weight of One Cubic Foot in Pounds.	Depth of Charge if Extra Solution is made up in Separate Tank.	Corresponding Weight of Charge in Tons.	Total Capacity of Plant per day if Tailings are Sluiced Out.	Total Capacity of Plant per day if Tailings are Shoveled out.
Silver Reef	Raw ore and tailings.	80.	Full (= 82 inches.)	55.	110 Tons.	100 Tons.
Pioche		100.		68.	155 "	135 "
Parral		80.		55.	110 "	100 "
Lake Valley	Roasted ore.	86.		59.	170 "	140 "
Parral		88.		60.	145 "	120 "
Yedras		60.		41.	80 "	70 "
Ontario		65.		44.	130 "	110 "
Daly		65.		44.	130 "	110 "
Sombrerete		60.		41.	120 "	100 "
Cusi		70.		48.	110 "	100 "
Chloride		79.		54.	140 "	120 "

12 hours allowed for filling each tank.
2 " " " emptying " " if the tailings are "sluiced out."
12 " " " " " " " " " " "shoveled out."

Table XXIIIa gives the capacity of the plant for various raw and roasted ores and tailings in tons per day. This capacity may be further increased to the extent of 30 per cent. by the addition of two more leaching-tanks, without making any change in the rest of the plant. On the other hand, it is not advisable to erect less than six leaching-tanks, since the remainder of the plant should be the same even if only four leaching-tanks were used. In the drawings showing the elevation of the plant, imaginary grade-lines are shown, one being about 25° and the other about 30°. These grades would be the most suitable, but they are by no means necessary. At Sombrerete the same kind of plant is being constructed on a grade of only 12° to 15°.

The building containing the pressure-tank, filter-press, and steam-dryer can be put in front of the sump for the solution-pump and the storage-tank for sulphides, instead of at the side.

In the plan, the floors are supposed to be removed, but are shown in the elevation. Pipes, details and dimensions are omitted in some of the views, in order to avoid a complicated drawing, but are given in the others. Figures alongside of pipe-lines indicate the sizes of the pipes. All tanks of all kinds, and all troughs or launders are accessible on every side.

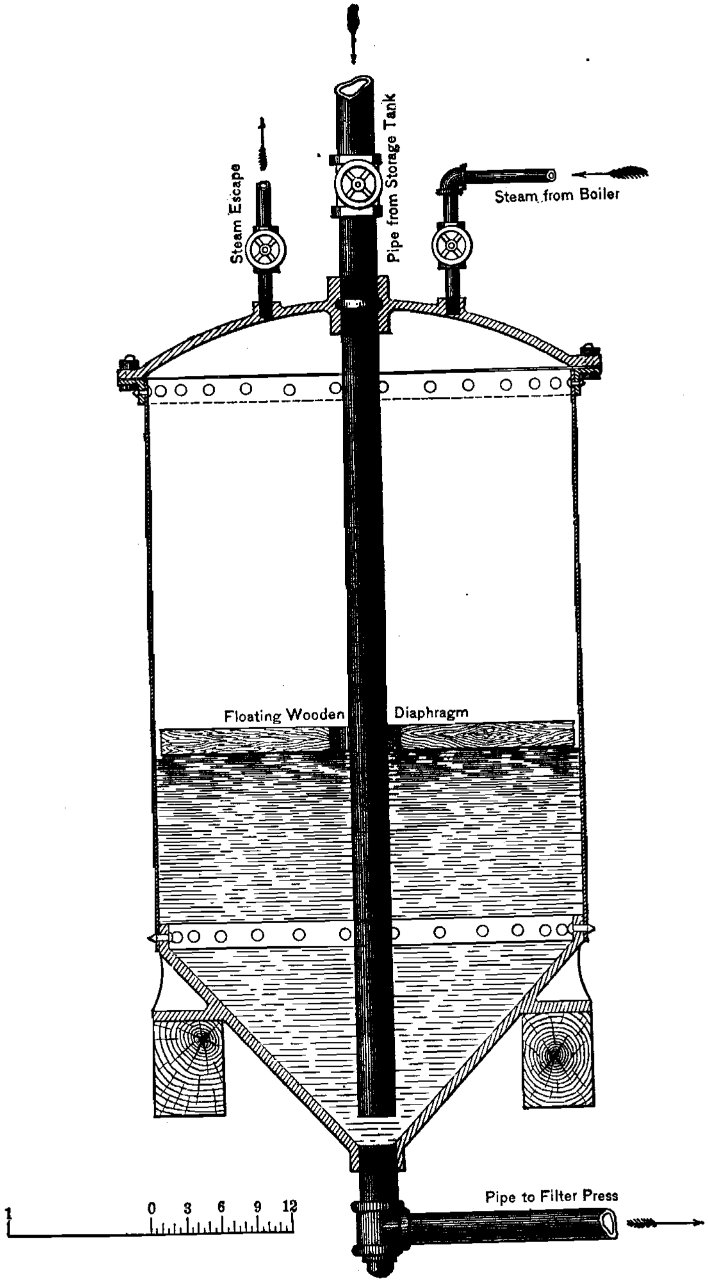
B.—*Leaching- and Storage-Tanks and the Tank for the Preparation of the Extra-Solution.* (Figs. 4 to 17 inclusive.)

All the wooden tanks are usually of white cedar, but this material is not necessary, as was shown at Silver City, the tank lumber being obtained only a few miles from the mill, and put in position while comparatively green. Such a course is not, however, to be recommended. The best plan, except for shipment to Mexico, is to order the tanks made in San Francisco, in diameter about two inches larger than as finally erected, ready to put up, with the exception of having no dowel-pins in the bottom and no gain in the staves. The sides of all the tanks are straight. In making the tanks, even if they have been set up in San Francisco, the staves are dressed to sweep of tank, and of such a length as to allow a 6-inch chime.

The gaining of the staves, one inch deep, is done by hand. No dowel-pins are used in the bottom, but instead the sides are grooved and joined with a tightly-fitting tongue $\frac{5}{8}$ by $1\frac{1}{2}$ inch, which is imbedded in the grooves with white lead.

All joints of the staves and bottoms, and particularly where the bottom enters the gain in the staves, must be fitted with the greatest

Fig. 1.



PRESS TANK.

accuracy, and the surface forming each joint must be covered with white lead just before finally putting them together. Nails or screws must not be used anywhere. Finally, the tanks are covered outside with three coats of white-lead paint, the last being one of a dark color, such as will not show dirt.

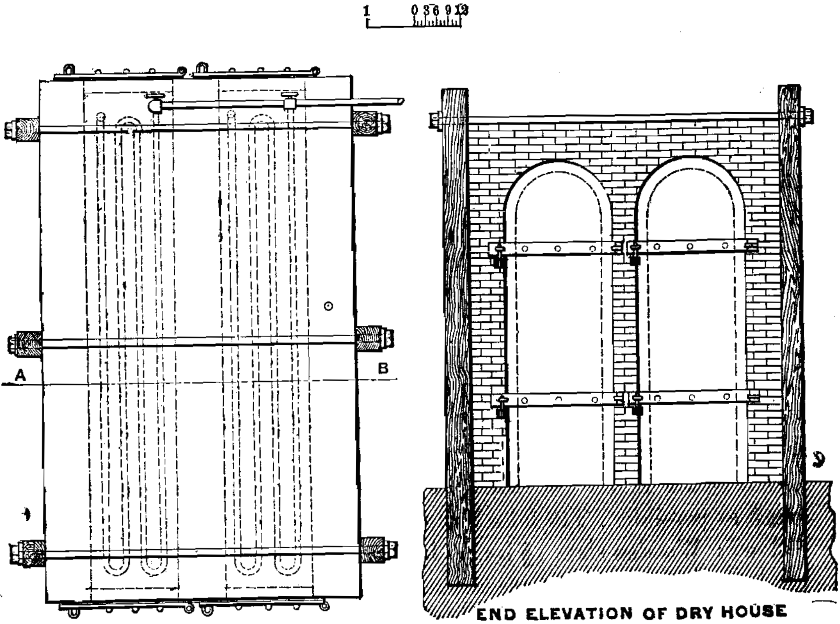
If the lumber is perfectly dry, several coats of paint inside will be beneficial. Too much care cannot be used in the putting together of the leaching-tanks, since not only is an absolutely tight tank required, if the ejectors are to be used to accelerate the leaching, but also because the use of the ejectors will greatly augment any defect in the tank. In the construction of storage- and precipitating-tanks for solution, so much care is not necessary, but will be found advantageous. But the construction of the precipitating-tanks for wash-water requires more care than for any other except the leaching-tank, as the wash-water requires to be heated if the precipitation is by acid and iron. These tanks should be of dry lumber, carefully put together and coated with paint. The dimensions and number of all tanks are given in the drawings, the storage-tanks being of the same dimensions as the precipitating-tanks.

The space under the filter in the leaching-tanks should be as small as possible. The false bottoms for these tanks are prepared as follows: Wooden slats 1½ inches high and 1 inch wide and separated one inch from each other, are fastened to the inside of the bottom of the leaching-tanks by screws imbedded in thick white lead. The side of the slats next to the bottom is cut out, about every 7 or 8 inches, ½ inch deep and 3 inches wide, so that a passage for the solution is established everywhere. Between the end of the slats and the staves a clear space 1⅝ inch wide is left all round, a strip of wood 1½ inches high and 1 inch wide, previously cut by a saw on one side in many places and well soaked in water so that it will bend easily, is fastened around the slats leaving an annular space ⅝ inch wide between the strip and the staves. One thickness of stiff matting (covering the slats and the circular strip, but not the annular space) forms a foundation for the filter cloth proper.

The latter, No. 8 canvas duck, is cut to a diameter 6 inches greater than the inside of the tank so that the edges can be pressed into the annular space described above, and kept in position by forcing down a ½-inch rope. The filter does not require to be removed oftener than once in six or eight months.

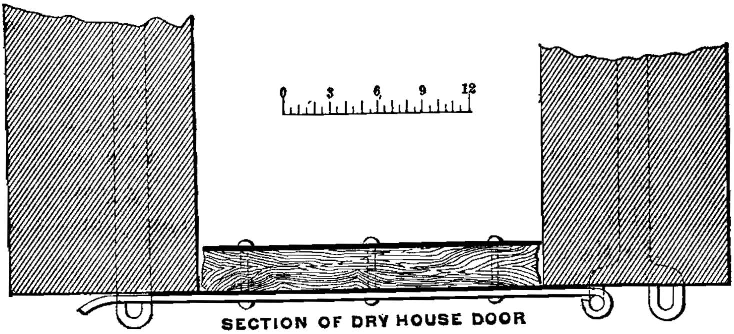
There is only one outlet to the leaching-tanks, which is in the center. It consists of a threaded cast-iron flange, bolted to the bottom, the bolts passing through the bottom, and together with the heads being imbedded in white lead. The details of these flanges, and the manner of connecting the two hose with them, are given in Figs. 12, 13, and 16 of the details.

FIG. 2.



PLAN OF DRY HOUSE

END ELEVATION OF DRY HOUSE



SECTION OF DRY HOUSE DOOR

There are three storage-tanks, the uses of which vary according to circumstances, as follows:

First: If raw ore is being treated, one of them (H) is always used for the preparation of the extra-solution, and the other two

(G, G) for heating and storing warm solution, which is always used in treating raw ores. Second: If roasted ore is being treated and there is ample leaching-tank capacity, the extra is made up on the ore, the storage-tank (H) being used for heating and storing warm stock-solutions, and the other two (G, G) for cold stock-solution, the coils in the bottom of the latter two not being used. Third: But if the leaching-tanks are to be run to their utmost capacity the storage-tank (H) is used for the preparation of the extra-solution and one of the other two for warm stock-solution and one for cold. The bottoms of the storage-tanks are six feet above the tops of the leaching-tanks. The method of heating the solution is by means of a coil consisting of about 100 feet of 1¼-inch lead pipe in each tank, supported on wooden blocks about 4 or 5 inches above the bottom. In regard to the triple connection between the three storage-tanks, in the first and second cases described above, the valves on the two tanks (G, G) are left open all the time, and the valve on (H) shut all the time.

In the third case all the valves are kept closed.

Above the storage-tanks and just under the ends of the pipe which brings the solution up to the storage-tanks from the solution-pump, are boxes about 2½ feet square and 1 foot high inside, with 1 inch holes in the bottom. The boxes over the two tanks (G, G) are for the purpose of breaking the force of the stream of solution as it comes from the pump, which would otherwise disturb the small sediment of sulphides which is still mechanically held in the solution and is gradually deposited on the bottom of the storage-tanks and is removed once every few months.

This sediment, if disturbed, would pass out of the storage-tanks on to the surface of the ore in the leaching-tanks, and any portion of it deposited after the use of the extra-solution might be thrown out with the tailings. But this box has for tank (H) an additional use if the extra-solution is made up in that tank, as it serves as a chemical box in which the chemicals for the extra-solution are dissolved by the solution passing through it.

At the bottom of each storage-tank around the outlets, are small boxes about 6 inches high, with perforated tops, which act, not only to prevent the solution being drawn down below 6 inches, thus keeping back sediment, but also keep chips, etc., from getting into the pipes and valves, over the leaching-tanks. But if the storage-tank (H) is used for the preparation of the extra-solution, this box must be removed so as not to keep back any cuprous hyposulphite, or yellow precipitate, which may be on the bottom of the tank.

The hoops of all tanks are of round iron, flattened at one end and with a lug forged on the outer side. The other end of the hoop is upset, threaded, passed through a hole drilled in the lug and a nut screwed on the end. The method of passing the hoop by the sluice-gate is shown in detail in Fig. 12.

C.—*Precipitating- and "Sump"-Tanks and Storage-Tank for Sulphides.*

The tops of the precipitation-tanks should be about 4 feet below the inside of the bottom of the leaching-tanks. The number of the precipitating-tanks required will vary according to the following circumstances:

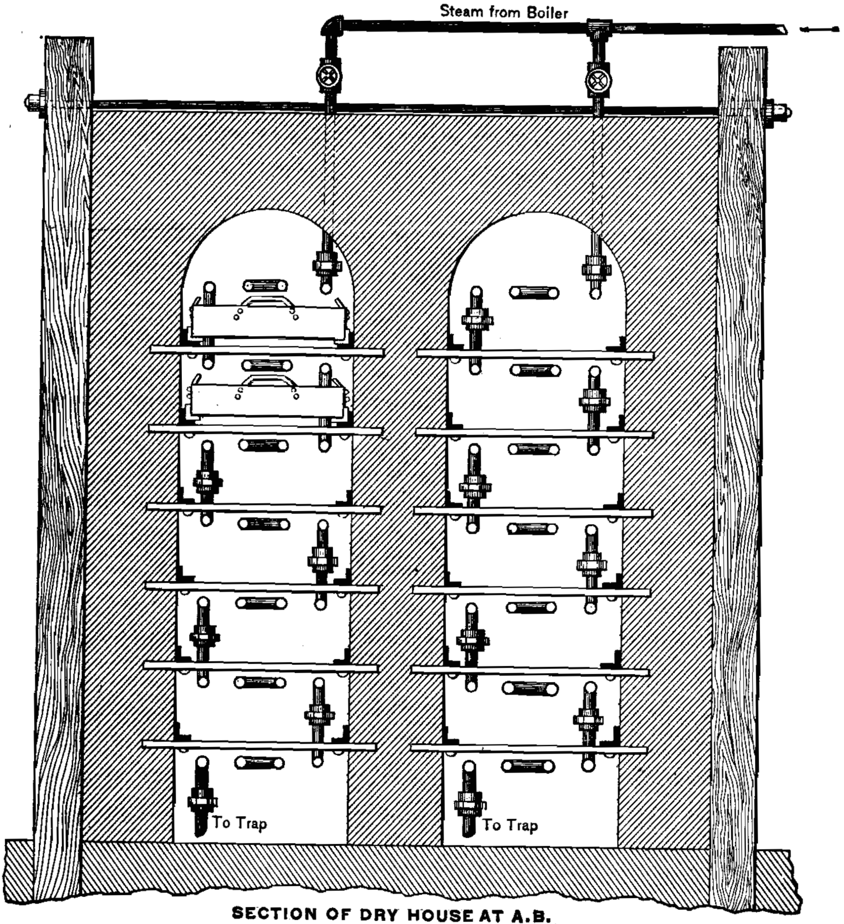
In the treatment of raw ores there is no first wash-water to be precipitated, so that the total number of precipitating-tanks for a raw-ore plant is only three instead of the six shown in the drawings.

In the treatment of roasted ore from which the solution dissolves little or no lead, the number of precipitating-tanks is the same as shown in the drawing.

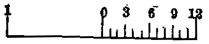
If lead is to be precipitated three additional tanks will be required, and should be placed between the tanks for the precipitation of solution and those for the precipitation of wash-water. They may be placed in the same line and on the same level, and the solution forced from them into the tanks for the precipitation of silver and copper, by means of siphon-pumps. The details of the surface discharge (d') of the precipitating-tanks are given in detail, Figs. 6 and 7. As to the stirring apparatus, a mechanical stirrer is entirely unnecessary and in the way.

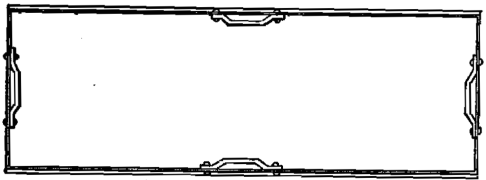
With an ash oar of 16 feet length, the total amount of stirring in twelve hours amounts altogether to only one hour by one man. For easy stirring with the oar, the platform around the precipitating-tanks should not be more than 6 inches below the top of the precipitating-tanks. The "sump"-tank (D) for the solution-pump has a box and strainer around the outlet, for the same purpose as that described under the head of storage-tanks. The storage-tank (E) for sulphides, has no such box. The pipes from the precipitating-tanks should be coated on the outside with asphaltum varnish; particularly is this necessary for the discharge-pipes from the wash-water precipitating-tanks.

FIG. 3.

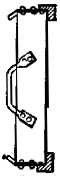


SECTION OF DRY HOUSE AT A.B.





PAN FOR SULPHIDE



CROSS SECTION

D.—*Tanks for the Preparation and Storage of Sodium Sulphide and Soda-Ash.*

The tank, I, for the preparation of sodium sulphide, is of cast-iron, in two sections, the lower being 1¼ inch thick and the upper ¾ inch. The two sections are bolted together through flanges and the joints are leaded. On to the bottom is poured 1 to 1¼ inches of melted lead. The pipe for live steam reaches to within 1½ inches of this lead plate.

The two tanks J, J, for the storage of sodium sulphide are of ⅜-inch boiler-iron, and in the drawing are represented with flanges on top, which are not necessary.

It will be noticed that there is a partition forming a small room around the tank for the preparation of sodium sulphide. This is necessary because in breaking up a drum of caustic soda, pieces are likely to fly and get into the storage-tanks holding the leaching-solution.

If soda-ash is to be used as a precipitant for lead, the same number and size of tanks should be used as for sodium sulphide, and arranged in the same way, but all are of ⅜-inch boiler-iron.

E.—*Pipes, Valves, Launders* (*or Troughs*), *Sluice-Gates and Chemical-Boxes.*

The sizes of all pipes are given in the drawings. The only place where the solution has any material effect on iron pipes, is between the outlet of the leaching-tanks and the precipitating-tanks. Therefore rubber hose is used, which seldom requires renewing. Six-ply steam-hose is recommended, simply because it is more durable and does not kink or sag, or wear out at the ends if continually plugged. The line *u*, leading from the center of the leaching-tank to the triple aunder, is in one piece. Wooden plugs are used for stopping the end. The hose *t*, leading from the center of the leaching-tank to the siphon, is the same size as the other, is attached to the flange below the other hose, and must have a gradual slope, without a sag, down to the siphon-pump, so that any drop of liquid coming from the leaching-tank will run down into the siphon-pump.

The hose leading from the siphon-pump up over the leaching-tank is also of the same size and in one piece, and is used in circulating the extra-solution. The end must be movable from one compartment of the double launder to the other. There is also a short piece of hose (*v*) leading from the siphon-pump to the triple launder, necessary only when the siphon-pump is used to accelerate the leaching. At other times it is kept plugged.

The ends of both pieces of steam-hose lying over the triple launder can be moved from one compartment to another.

The valve in the steam-pipe which furnishes steam to the siphon-pump should be of Jenkins's pattern, and situated 3 or 4 feet above the siphon so that the leaching-solution can not come in contact with it. As described below, the steam for the siphon-pump must be dry and therefore the main and branch lines of steam-pipe must be covered. The 2½-inch pipes (a' and b') from the launders to the precipitating-tanks are also steam-hose.

In order to keep any sand or sediment which might get into the launders, from passing into the precipitating-tanks, nipples or short pieces of 2½-inch pipe are inserted in the outlets of the triple launder and project about one inch above the bottom. The launder (*l*) and the pipe (*z*) are for that part of the wash-water which does not require to be precipitated. The trough (or launder) (*m*), at the back of the precipitating-tanks, conducts the sulphides to the storage-tank for sulphides. It has a grade of about 4 feet in 100. The sulphides are run into it from the precipitating-tanks by a short pipe attached to a flange bolted on to the precipitating-tank just above the bottom, so that all the sulphides can run out.

The valve is a gate-valve, something like a molasses or oil-gate, and is shown in detail in Figs. 8 to 11 inclusive.

The launder (*n*, *o*) in front of the precipitating-tanks is horizontal, and has a division opposite a point between the third and fourth precipitating-tanks, so that one half its length is used for conducting solution to the pump-sump, and the other half to conduct the precipitated wash-water out of the mill.

One side of the double launder above the leaching-tanks has no division, but runs the whole length of the leaching-tanks, having a hole over each, which is closed with a wooden plug. This serves for transferring the special extra from one ore charge or tank to another. The other compartment of the launder has a division between each two leaching-tanks and conducts solution or water to the chemical-box, which has a perforated bottom and serves not only as a receptacle for chemicals, but also to break the force of the stream of solution, so that it does not make a hole in the ore.

The sluice-box or launder conducting the tailings out of the mill should have a slope of 1 inch to the foot. The details of the sluice-gate have been given in *Trans.*, xv., but the scale there given was

Fig. 6.

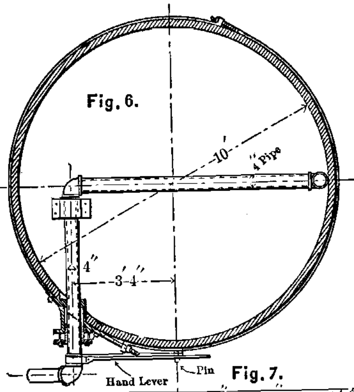


Fig. 7.

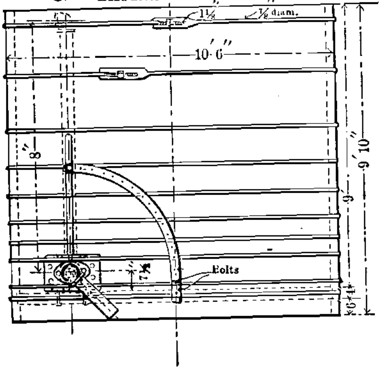
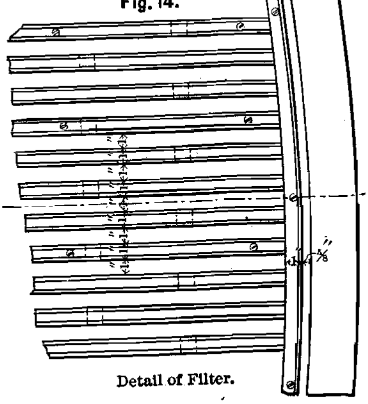
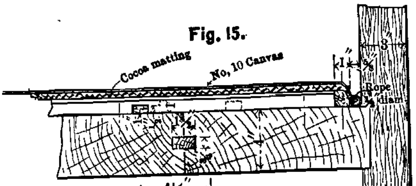


Fig. 14.



Detail of Filter.

Fig. 15.



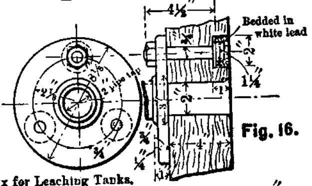
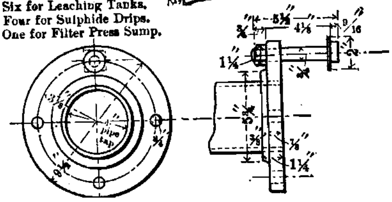


Fig. 16.

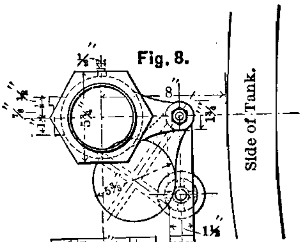
Six for Leaching Tanks.
Four for Sulphide Drips.
One for Filter Press Sump.



Three for Solution Tanks.

Fig. 17.

Fig. 8.



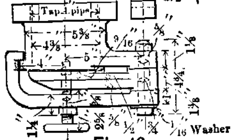


Fig. 10.

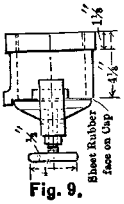


Fig. 9.

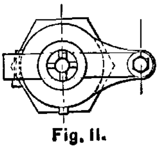


Fig. 11.

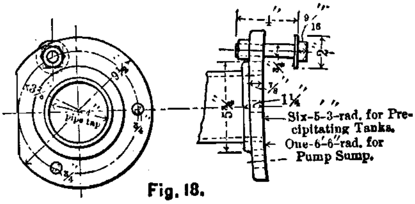


Fig. 18.

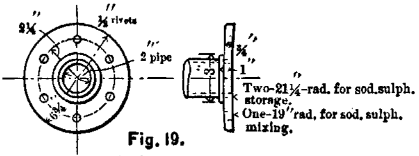


Fig. 19.

LEACHING PLANT FOR THE RUSSELL PROCESS.
DETAILS.

Fig. 7, Precipitating-tank; Figs. 8, 9, 10, and 11, Gate for discharging sulphides; Figs. 1 and 15, Detail of filter; Figs. 16, 17, 18, and 19, Discharge-pipes for the various tanks.

wrong. The opening should be about 12 inches high by 18 inches long. The hose-connections for sluicing are shown at *y*. The two lines of hose are each 30 feet long, each doing duty for three leaching-tanks.

The sluice-gate shown in detail in Figs. 12 and 13 is packed with a rubber gasket. The solution-pump sets in a low tank about 6 inches high, which catches any solution that may leak from it, and a pipe conducts it to the sump-tank under the filter-press.

The water-pipe *r* is for leaching. There is no occasion for any water-pipe to the storage-tanks, as no water is there required after once making up the stock-solution. On that occasion the water may be obtained by running water into one of the leaching-tanks and then down to the solution-pump and pumping it up to the storage-tanks.

All the valves in the triple connection between the storage-tanks are gate-valves; also those on the pipe-line leading up from the solution-pump to the storage-tanks and those near the storage-tanks on the pipe-lines leading from the storage-tanks to the leaching-tanks, and all those on the sodium-sulphide tanks and pipes; but all those over the leaching-tanks are Jenkins's rubber-faced globe-valves. The valves between the solution-pump and "sump" and between the storage-tank for sulphides and the pressure-tank, and between the pressure-tank and filter-press, and on the pipes which convey the sediment from the solution-pump sump and the solution storage-tanks, and on the line of water-pipe for sluicing tailings, are also gate-valves. At every angle in the pipes for conveying sodium sulphide should be a cross instead of an elbow. A slight sediment of silver sulphide gradually collects in the solution-pump sump and in the solution storage-tanks which must be occasionally removed. This is done by means of the pipes *b''* and *c.''* The valves should be near the tanks. The pipe from the solution-pump sump ends near the bottom of the pressure-tank, and is connected with it by a hose-coupling only when needed; a permanent connection not being advisable as a leak in the valve might occur without being detected, thus allowing sulphides to be forced back into the solution-pump sump.

F.—*Siphon-Pumps (or Ejectors), Solution and Sluicing-Pumps.*

The siphon-pumps, made by A. Aller, 109 Liberty Street, New York, are lined with hard lead and have a platinum steam-nozzle. Size No. 4, costing $58, is the most suitable for a 16-foot leaching-tank. In using the siphon-pumps for leaching two precautions are necessary: first, the siphon-pump must be upside down (suction up) so that all solution coming from the leaching-tanks, running down the sloping hose, may fall directly into the steam in the ejector. Otherwise, or if the hose has a sag in it, the siphon will act only intermittently, ejecting first steam, then solution, then steam, etc. Secondly, the steam used must be dry, which can only be secured in this case by covering the steam-pipes. The difference in effect between wet and dry steam for siphon-pumps was illustrated at a mill in Arizona. The steam was conducted about 200 feet in an uncovered 1¼ inch pipe. The result was that the siphon-pump would not lift six inches; and was therefore useless. It was then used near the boiler, and lifted 20 feet, and forced the solution an additional 20 feet.

In addition to one siphon-pump for each leaching-tank, there is also one for returning the clear solution coming from the filter-press to the sump of the solution-pump.

If lead is to be precipitated with soda-ash, one siphon-pump (which should be a No. 6) will also be necessary for each of the three lead precipitating-tanks. A siphon-pump is also sometimes necessary in making up the extra-solution in one of the storage-tanks; for in case the chemicals in the box with perforated bottom should not be dissolved by the time the right volume of solution had run into the storage-tank through that box, some of that solution would have to be forced up and circulated through the chemical-box until the chemicals were dissolved.

The solution-pump should be a Knowles 6½ double plunger, with 8-inch steam cylinder, 5-inch plunger and 12-inch stroke, and a 4-inch suction and discharge. The fire-pump for sluicing should be a Knowles (or corresponding size of other make) single cap pattern with 10-inch steam and 5-inch water-cylinder, 12-inch stroke and 4-inch suction and discharge.

G.—*Apparatus for the Mechanical Treatment of the Sulphides.*

This consists of a pressure-tank, Johnson filter-press, and steam-dryer. The details, the pressure-tank, and steam-dryer, are shown in Figs. 1 to 3 inclusive. The filter-press, "circular pattern," is furnished by John Johnson, Franklin Square and Cherry Street, New York. The size should be 18 inches, with 24 chambers, 12 two-inch distance-rings, one dummy-plate, pressure-gauge, indicating as high as 150 pounds, one dozen extra tension-hooks and screws, 3 sets of "double-chain" filter-cloths, but no pump, pulleys, shafting or hangers. The cost of the above is about $290 f. o. b., New York. Much trouble has hitherto been encountered in using a pump for forcing sulphide into the filter-press, but the pressure-tank gives no trouble. If the pressure-tank and press are arranged as shown in the drawing, very little steam-pressure is needed, as most of the sulphides run by gravity into the press. It is likely that compressed air will eventually take the place of steam for this purpose.

If the sulphides are to be dried by steam the use of the filter-press cannot be avoided. Near the front of the steam-dryer is a small windlass by which the pans of sulphides are lifted into, or removed from the steam-dryer. At the side of the steam-dryer is the sampling-floor and scales. The sampling-floor is best made of iron plates, the larger the better, closely jointed.

H.—*Miscellaneous.*

Experience has shown that a separate steam-boiler for the leaching-plant is advantageous, even when the regular mill-boilers are available. The steam for the leaching-plant can thus be maintained at a greater pressure than would be advisable in the regular mill-boilers, the filter-press, siphon-pumps and steam-dryer doing much better work with high-pressure steam. Moreover, the leaching-plant is thus rendered independent of the rest of the plant, and can be run when the mill-boilers are shut down.

In the rear of the leaching-tanks is a car-track having a slight inclination, which answers not only for bringing ore from the rest of the mill-plant, but also for conveying chemicals from the chemical-room to the leaching-tanks and chemical-boxes if the extra-solution is made up on the ore. The car for chemicals runs over the scales in the chemical-room, where the blue-stone and hyposulphite are stored. The caustic soda and sulphur are in the upper chemical-room, in which is also a pair of platform scales. In addition to the scales already mentioned, there must be another set for weighing the ore delivered to the leaching-tanks.

If the tailings cannot be sluiced out, owing to scarcity of water, they should be removed by cars running under the leaching-tanks, as shown in the drawing.

The whole building should be well lighted, particularly over the precipitating-tanks, a skylight being placed in the roof over each of these tanks.

The platform around the leaching-tanks should be about 2½ feet,

Fig. 12.

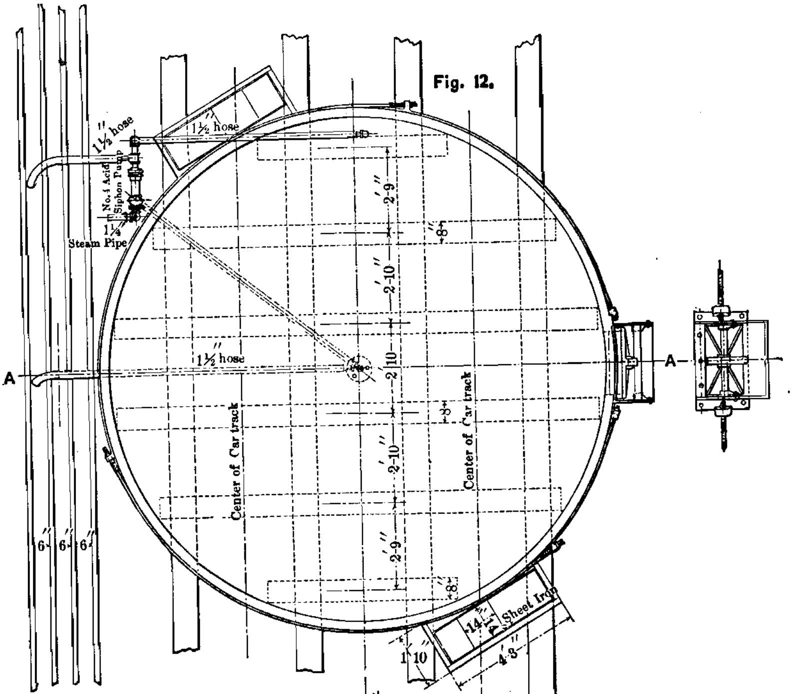
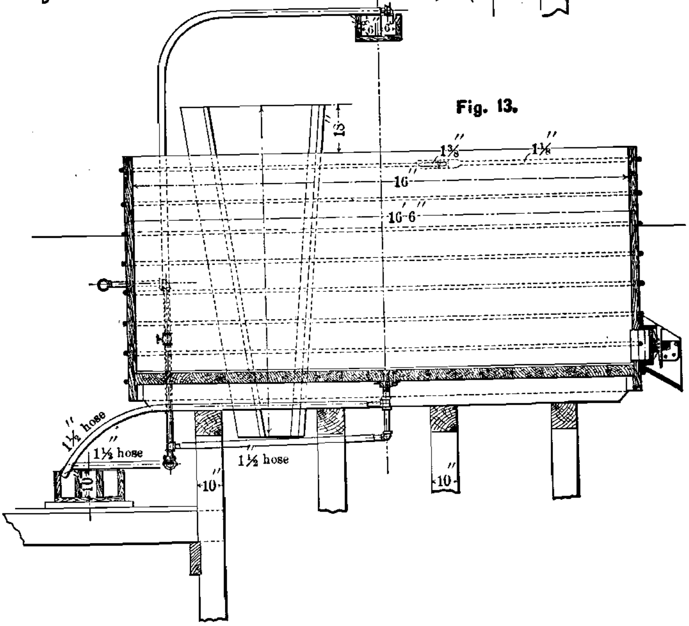


Fig. 13.



Leaching-tank for Russell process. Fig. 12, Plain; Fig. 13, Section.

and that around the precipitating-tanks 4 to 6 inches, below the top of these tanks respectively.

All treatment of the sulphides below the sulphide storage-tank takes place in a room shut off from the rest of the mill.

IV. Expenses.

A.—*Comparative Values of Chemicals in the United States and Mexico.*

The following table gives the average values of chemicals, per hundred pounds avoirdupois, in Mexico, near the railroad, as compared with similar places in the mining regions of the United States.

	Mexico.	United States.
Hyposulphite of soda,	$8.70	$4.25
Blue-stone,	9.50	5.75
Caustic soda,	10.75	5.50
Sulphur,	5.60	3.50
Soda-ash,	7.00	3.50
Sulphuric acid (1700 pound drums),	16.00	3.75
	$9.60	$4.37

The cost of all the chemicals averages about twice as great in Mexico as in the United States. But the use of the acid can nearly always be omitted when the extra-solution is used. The value of chemicals in Mexico therefore averages 8.3 cents per pound, or 1.9 times as much as in the United States.

B.—*Cost of the Stock-Solutions.*

The total amount of hyposulphite for stock-solution for works of various capacities has been given as 3281, 1875 and 1406 pounds for works of 100, 50, and 25 tons respectively.

The corresponding costs in the United States of these stock-solutions are therefore $140, $80 and $60.

C.—*Cost of Precipitating the Silver from the First Wash-Water by Various Methods.*

Table XVII gives the cost of precipitating the first wash-water from roasted ore, and the values of the precipitates. Except in Mexico, where acid costs about five or six times as much as in the United States, there is little difference in cost between the method by acid and iron and that by sodium sulphide. In the United States, if the wash-water is acid, the cost is in favor of acid and iron by 3 or 4 cents per ton. In cases where the method by dilution can be used, of course the expense of precipitating is merely nominal. But generally, on account of the small cost and greater thoroughness of the first two methods, one of them should be adopted in preference to the method by dilution.

D.—*Cost of Precipitating the Metals from the Leaching-Solutions.*

In all the mills mentioned in Table XXIII, except the Rubio in Parral working on Veta Grande tailings, sodium sulphide was used as a precipitant, and the cost of precipitating per ton of ore, as there given, is the cost of the sodium sulphide.

The comparison in cost between sodium sulphide and calcium sulphide is more favorable for the former in the United States, than in Mexico, as the cost of lime in Mexico and the United States is the same, while the price of caustic soda is very high in Mexico, being nearly—sometimes more than—twice as great as in the United States. Nevertheless, even in Mexico, the sodium sulphide is much superior, both in cost of chemicals and the time, labor and heat required in its preparation. A comparison between sodium sulphide and calcium sulphide, made at Cusi, may be found in the MINING AND ENGINEERING JOURNAL, vol. xlii., page 383, showing the sodium sulphide to be the cheaper by 22 cents per ton. A comparative test at Sombrerete, also made by Mr. Watson, resulted in the adoption of sodium sulphide in place of calcium sulphide, although caustic soda, at that time, cost 14 cents per pound (in Mexican silver).

E.—*Cost of Various Extra-Solutions for Acid Roasted Ores.*

A preceding section gives the weights of blue-stone and hyposulphite required per ton of ore for extra-solutions of various strengths. Table XXIV gives the corresponding costs of these chemicals and extra-solutions per ton of ore. The cost of the extras, including the precipitating of the metals contained in them, varies from 49 cents to $1.24, the latter representing the strongest extra-solution ever used, and used, moreover, under the most unfavorable circumstances, as regards the quantity of hyposulphite required. A corresponding table of extra-solutions for raw ores would show less cost for each extra, because, for the same amount of hyposulphite per ton, the stock-solution will average in strength about 40 to 50 per cent. higher than in the case of an acid roasted ore, while the volume of extra-solution per ton of ore is less. No table can well be given for simple alkaline roasted ore. A table for alkaline arsenical roasted ores would be about the same as for acid roasted ores.

TABLE XXIV.

COST OF CHEMICALS FOR EXTRA-SOLUTIONS OF VARIOUS STRENGTHS AND OF THE PRECIPITATION OF THE SAME.

	For a Stock-Solution containing $\frac{6}{10}$ of 1 Per cent. Hypo.	For a Stock-Solution containing $\frac{8}{10}$ of 1 Per cent. Hypo.	For a Stock-Solution containing One Per cent. Hypo.	For a Stock-Solution containing 1.2 Per cent. Hypo.	
	1	2	3	4	
Cost of Blue-stone Per Ton	28.7 cts.	28.7 cts.	28.7 cts.	28.7 cts.	For a weak Ex. Solution to contain $\frac{7}{10}$ of 1 per cent. in Bl' stone.
" " Hyposulphite " "	21.2 "	15.9 "	10.6 "	4.2 "	
" " Precipitating " "	16.0 "	16.0 "	16.0 "	16.0 "	
Tot'l Cost of using Ex. Solut'n	$0.66	$0.61	$0.55	$0.49	
	5	6	7	8	
Cost of Blue-stone Per Ton	37.4 cts.	37.4 cts.	37.4 cts.	37.4 cts.	For a medium Ex. Solution to cont'n $\frac{8}{10}$ of 1 per cent. in Blue-stone.
" " Hyposulphite " "	39.3 "	34.0 "	27.6 "	21.2 "	
" " Precipitating " "	24.0 "	24.0 "	24.0 "	24.0 "	
Tot'l Cost of using Ex. Solut'n	$1.01	$0.95	$0.89	$0.83	
	9	10	11	12	
Cost of Blue-stone Per Ton	42.0 cts.	42.0 cts.	42.0 cts.	42.0 cts.	For a str'ng Ex. Solution to contain 1.1 per cent. in Bl' stone.
" " Hyposulphite " "	51.0 "	46.7 "	38.2 "	34.0 "	
" " Precipitating " "	31.0 "	31.0 "	31.0 "	31.0 "	
Tot'l Cost of using Ex. Solut'n	$1.24	$1.20	$1.11	$1.07	

F.—*Comparison Between Raw, Acid-Roasted and Alkaline-Roasted Ores, as to Cost of Chemicals.*

The difference in cost of chemicals per ton of ore is much in favor of alkaline, as compared with acid ores. For the former, the cost of blue-stone is about 27 per cent., of hypo, 43 per cent., and of sodium sulphide, 35 per cent., less than for the latter, as is shown in Tables XIV and XXIII. For raw ores, the average cost of chemicals is still smaller, being about 20 per cent. less than for alkaline-roasted ore.

G.—*Disposal of the Sulphides.*

The following statistics are furnished by the manager of one of the largest mills using the Russell process, the product being from 60,000 to 80,000 ounces of silver per month.

The monthly shipments are divided into two parts, one being sent to the Chicago and Aurora Smelting and Refining Co. at Chicago, and the other to E. Balbach & Sons, Newark, N. J. The division is made for the double purpose of having two checks on the work of the assayer at the leaching-mill and also of having two refiners as checks upon each other. As the monthly shipments occupy a number of packages, the division is made by sending the even numbered cases to one refiner and the odd to the other, thus securing an average sample of the whole lot for each refiner. The result of this method is that the returns from both refiners are entirely satisfactory, the assays made at the mill and those by the buyers being virtually identical. As soon as the shipments are received by the buyers they are weighed, sampled and assayed, the returns being made within a week after the shipments are received. Within a week after instructions are received by the refiners to work the sulphides, the payments are made by check on New York, the silver being paid for on the basis of New York price on day of settlement, and gold at the rate of $20.60 per ounce.

The Balbach Co. pay 99 per cent. of the value of the silver and full price for gold, and charge $100 per ton for treatment. The Aurora Works pay 98 per cent. of the value of the silver and charge nothing for treatment. Their rates for gold are not known, as the sulphides in question contain only a trace. The Balbachs pay for the least trace of gold. The rate paid for copper is generally $1.00 per unit, although not in the case above mentioned, as very little copper is present.

The sulphides from the Sombrerete mill are treated at the Argo Works, Denver, Col. The price paid by these refiners is 97 per cent. of the gold and silver (silver at N. Y. quotations and gold at $20 per oz.) without any charge for treatment.

The sulphides from the Sombrerete mill contain $700 to $800 in gold per ton. Taking into account the silver entering the cupel, the mill-assays for silver average about 0.6 per cent. above those of the refiners. Without taking this into account, the refiners' assays are a little higher than the mill assays. The average of the assays on gold agree within 0.1 per cent.

H.—*Details and Summary of Cost of Running the Plant.*

The following statement gives the approximate amount of labor required per day of twenty-four hours, in the leaching department only, for a plant of 100 tons capacity, as demonstrated in various mills:

2 men on leaching-tanks, at	$4.00	$8.00
3 " as helpers, at	2.50	7.50
2 " on precipitating-tanks, at	4.00	8.00
2 " on pump, press, dryer, boiler, etc.,	4.00	8.00
2 " as helpers, at	2.50	5.00
Total,		$36.50
Cost of labor per ton,		36½ cents.

The three helpers at the leaching-tanks attend also to the preparation of the sodium sulphide, which occupies about three hours once every two days, and also to the preparation of the extra-solution, which occupies one man about two hours, two or three times each day. Of the three helpers, one is on night-shift, and the other two on day-shift, the sluicing out of the tailings being done only on day-shift, and occupying one man about four to five hours each day. For 50-ton tanks, the two men and three helpers are more than sufficient, as the labor is much less than where ten-ton tanks are used. Also the two men and two helpers are more than sufficient for the solution-pump, press, dryer, boiler, sampling of the sulphides, etc. If the plant is arranged as described, the pressing of the sulphides will occupy only one man three days per week; the removing of the dry sulphides from the steam-dryer and replacing with wet sulphides occupies two men about four hours every three days, and the sampling also two men about six hours every three days. The boiler needs but little attention, the steam required being about as follows: A very small amount nearly all the time for heating the solution in the storage-tank and for the steam-dryer; steam to run two siphon-pumps or ejectors, five hours per day: to press sulphides, three days per week: to run the fire-pump (if tailings are sluiced), three to four hours each day: to pump the solution to the storage-tanks all day, at intervals; and for the preparation of sodium sulphide one hour every two days. At Yedras, the amount of wood used per day was three-fifths of a cord, for pumping solutions and for making calcium sulphide. Four times this amount (or two and a half cords) would therefore probably be enough for all the above purposes.

As to the removal of tailings, the cost of shovelling out and removal at Cusi, was ten to eleven cents per ton of ore. At Sombrerete, the filling of the tanks from the cooling-floor, and shovelling out and removal of the tailings cost only fifteen cents per ton, labor being cheaper than at Cusi.

At Yedras, the cost of labor and material for repairs on the leaching-plant was only fifty dollars per month or about four and a half to five cents per ton. At Cusi, it was several times as great per ton, owing to the poor arrangement and condition of the plant.

The silver-bearing solution coming from the ore-tanks to the precipitating-tanks should be conducted in rubber hose. But at the Cusi mill this silver-bearing solution was conducted from the ore-tanks to the precipitating-tanks by means of iron pipes, and these pipes, which are practically the only ones attacked by the solution, had to be periodically replaced. The solution-pump was set on top of the solution-sump instead of level with the bottom, and required constant repairing in order to produce a vacuum sufficient to lift the solution. The twenty-two small leaching-tanks and corresponding valves, piping and siphon-pumps also required considerable attention.

For the plant described and for 100 tons per day, the total cost of labor and material for repairs should not exceed $150 to $200 per month, or five to seven cents per ton.

The total cost of the various operations and chemicals required for 100 tons per day is given in the following statements, which show approximately the minimum and maximum running expenses per day of twenty-four hours at the rate of 100 tons per day, in the United States, as shown by actual experience. The cost of charging the ore into the tanks is not included. It would vary probably from ten to eighteen cents per ton. The removal of the tailings is supposed to be by sluicing; if by shovels and cars, the expense may be put at about fifteen cents more per ton.

Approximate Cost of Running Leaching Plant at 100 *Tons Capacity per day of twenty-four hours.*

Minimum.	Per Day.	Per Ton.
Labor, 11 men at $3.00,	$33.00	$0.33
Fuel, 2½ cords at $4.00,	10.00	0.10
Chemicals,	56.00	0.56
Repairs $150 per month,	5.00	0.05
Assaying, ½ total for mill,	7.00	0.07
Total,	$111.00	$1.11

Maximum.	Per Day.	Per Ton.
Labor, 13 men at $3.00,	$ 39.00	$0.39
Fuel, 3½ cords at $6.00,	21.00	0.21
Chemicals,	100.00	1.00
Repairs $450 per month,	15.00	0.15
Assaying, ½ total for mill,	8.00	0.08
Total,	$183.00	$1.83

I.—*Total Mill-Expenses of Various Mills While Using the Russell Process.*

Table XXXVI gives the total mill-expenses per ton of roasted ore in the Cusi mill for nine months, while using the old or ordinary leaching-process, the Russell process, and both processes together (mixed months). The total expenses, including assaying, crushing, drying and stamping, roasting, leaching, roasting and refining of the sulphides, and production of bullion 980 fine, while using the Russell process, were $12.08.

At Parral, also in Mexico, the total expenses for the treatment of tailings from ore which had originally been roasted, and leached by the ordinary process, were $2.10 per ton for 40 tons per day. For roasted ore, at the rate of 10 tons per day, the total expenses were $9.15.

It should be observed that the expenses at Cusi much exceed, are indeed probably double, what they would be in a well-constructed mill in the United States.

In Table XXV the Cusi expenses are taken as a basis, from which are deduced the corresponding expenses for acid-roasted ores and raw ores in the United States.

The Cusi mill, however, was built so as to utilize, in part, old plant, and to work San Bartolo ore by the ordinary process. It was ill-adapted, particularly in roasting and dust-saving facilities, for economical work on the baser San Antonio ore, by any process, and in its leaching department was unsuited to the Russell process. The

$12.08 on which the estimates are based is the average mill-expense per ton while using the Russell process. In the second estimate of Table XXV, the reduction in cost, due to improvements in arrangement and construction, amounts to 30 per cent.

TABLE XXV.

COMPARISON OF TOTAL MILL-EXPENSES PER TON OF ORE IN THE UNITED STATES AND MEXICO, BASED ON ACTUAL EXPENSES AT CUSI.

FIRST ESTIMATE. Total Mill-Expenses for Cusi Ore and Mill if the same Ore and Mill were in the U. S.			
Actual total mill-expenses of Cusi (see Table XXXVI)		$12.08	
Deduct for difference in cost of salt and chemicals	$2.54		
Deduct for difference in expense due to shipping instead of refining prod.	.93		
Deduct for difference in cost of supplies, other than salt and chemicals	.60	4.07	
Estimated expense per ton of ore.			$8.01
SECOND ESTIMATE. Total Mill-Expenses for Cusi Ore in a Well-Constructed 75-Ton Roasting Mill in the U. S.			
Total mill-expenses for Cusi ore and mill in the U. S. (see last estimate)		$8.01	
Deduct for difference in cost of running 1 Stetefeldt as compared with 3 Howells	$1.00		
Deduct for difference in expense due to substitution of 6 50-ton for 22 9-ton leaching-tanks	.65		
Deduct for difference in expense due to increase of capacity from 55 tons to 75 tons	.86	2.51	
Estimated expense per ton of ore.			$5.50
THIRD ESTIMATE. Total Mill-Expense for Dry-Crushing, Raw-Leaching Mill in the United States.			
Total mill-expense for Cusi ore in a well-constructed 75-ton mill in the U. S.		$5.50	
Deduct cost of chloridizing, salt and difference in amount of chemicals per ton	$2.40	2.40	
Estimated expense per ton of ore.			$3.10

In a well-constructed raw-leaching mill, the total mill-expenses per ton for 75 tons per day should not exceed $3.00 per ton, and under favorable circumstances would fall as low as $2.50, particularly if the crushing were done by rolls instead of stamps.

At Silver Reef, Utah, the total mill-expense, including also assaying and general expense, was $1.65 per ton for raw tailings, at the rate of 40 tons per day, and $3.00 to $4.00 for raw ore, at the same rate per day. The corresponding expenses for 75 tons per day would be about $1.25 for raw tailings, and about $2.85 for raw ore.

For roasted ore, at Lake Valley, the total mill-expenses for 60 tons per day, were estimated by the general manager, Mr. Hadley, at $4.65 per ton, including assaying, etc.

V. Applicability of the Process.

A.—*Values of the Materials Treated, in Gold, Silver and Lead.*

Table VI gives the average values of all the tailings and raw and roasted ores on which the Russell process has been or is now in use. The values of the tailings, in silver, range from 8 to 12 ounces, of raw ore 14 to 15 ounces, and of roasted ore 12 to 88 ounces per ton. In gold, the value has been from 0 to $5 and $6 per ton. In lead, the range is from 0 to 23 per cent.; ore containing the latter proportion having been treated at Cusi, and also at Sombrerete.

B.—*Special Applicability to Ores Containing Lime, Antimony or Arsenic.*

The superiority of alkaline-roasted, as compared with acid-roasted, ores in regard to amount of chemicals required per ton, has already been illustrated.

So far as extraction by hyposulphite solution is concerned, the most desirable elements which can be present in material which is to be roasted, are antimony or arsenic, or both of them, in combination with the silver; since, in that case, both antimoniate and arsenate of silver are formed during the roasting, both of which are soluble in the leaching-solutions. Of the two elements, arsenic is the most desirable, not only because arsenate of silver is about 3.57 times as soluble by a given weight of hyposulphite as antimoniate of silver, but also because the solubility of arsenate of silver is increased 2.4 times by the presence of caustic alkali, while that of antimoniate of silver is raised only one-sixth. The latter peculiarity causes the presence of arsenic in combination with silver, to be particularly desirable in the treatment of alkaline-roasted ores.

On account of the solubility of antimoniate and arsenate of silver, in the leaching-solutions, some ores can be roasted entirely without salt, and still yield a high percentage by the Russell process (as at Yedras and Ontario).

The Solace ore in Idaho, from which nothing can be extracted in the raw state, yields 92 per cent. after an instantaneous (or 5 seconds) roasting without salt in the muffle, on account of the presence of arsenic and antimony.

C.—*Special Applicability, Either With or Without Concentration, to the Treatment of Raw Ores.*

The advantages in favor of raw leaching as compared with leaching after roasting are as follows:

1st. The cost of plant for raw leaching is much less than that required for a treatment which includes roasting.

2d. Less power and less labor required in handling.

3d. No losses or discrepancies due to dust or volatilization in roasting.

4th. No salt.

5th. No acid, and less chemicals required for leaching.

6th. Less water, the amount per ton averaging only one-fourth to one-fifth that required for roasted ores.

The practicability of raw treatment depends upon the value of the ore, supposing the percentage-extraction to be the same for raw ore of different values. For instance, for a well-arranged mill, the losses occurring in the treatment of raw ore would probably be about 1 per cent., in addition to the loss in tailings, from the time the ore enters the mill up to and including the production of dried sulphides. In a roasting-leaching-mill the corresponding losses would be, probably, about 3 per cent.

In expenses, the advantages are in favor of raw leaching by about $3.00 per ton. If the ore to be treated, therefore, has a value of 20 oz. per ton, 73 per cent. extraction by raw leaching is equivalent to 90 per cent. by roasting and leaching. But if the ore has a value of 50 oz., the extraction from the raw ore must be 82 per cent., to be equivalent to 90 per cent. by roasting and leaching.

The great difference between the result by the ordinary and by the extra-solution, on raw ore, is illustrated in the case of some ores lately experimented upon by Mr. F. M. Watson, Superintendent of the Sombrerete mill. These results were as follows:

Name of Mine.	Value in Ounces.	Per cent. Extracted in Assay Office.	
		By Ordinary.	By Extra.
Buena Vista	60.0	25.0	70.0
San Geranimo	219.0	2.8	81.1
Penoles	38.0	8.0	86.0

If, however, the tailings from leaching, as is frequently the case, contain the precious metals in such a combination as to admit of concentration, a combined leaching and concentration will frequently be more applicable than roasting and leaching, even though the leaching alone may not yield more than 30 or 40 per cent. of the value of the ore. The additional cost of concentrating the tailings from the leaching will seldom be more than 50 to 70 cents per ton. When leaching and concentration are combined, if the concentration is by the wet method, it must follow the leaching; since, if it preceded, the result would be a separation of the slimes and coarse ore, which would make the leaching practically impossible on account of the mechanical difficulty of introducing the leaching-solutions into or of removing them from the ore.

D.—*Applicability to Tailings from Amalgamation or from the Ordinary Leaching Process.*

The Russell process extracts silver-compounds which cannot be extracted by amalgamation. This is shown in the case of raw tailings at Silver City, New Mexico, which had already been treated twice by amalgamation, and once by concentration, and from which amalgamation, even in assay-office tests, would still extract only 22 per cent., while the extraction by the extra-solution was 72 per cent.

Also at Silver Reef, Utah, the raw tailings, which had already been twice treated by amalgamation and from which practically nothing could be extracted by amalgamation, yield 40 to 80 per cent. by treatment with the extra-solution. This is in spite of the fact, that the Silver Reef ores, from which these tailings were produced, are the only raw ores so far met with, which yield a higher percentage by amalgamation on the first treatment, than by the extra-solution, although the expenses are less for the latter.

At the Minas Prietas, in Mexico, the extraction from the raw ore by amalgamation in a Boss mill, is 76 per cent. of the gold, but only 9.2 per cent. of the silver. The extraction from the tailings by the Russell process is 54.6 per cent. of the gold and 30 per cent. of the silver.

E.—*Applicability of the Process to Dry or Remote Regions, or Where Salt is Expensive.*

The amount of chloride of silver present in an ore is practically immaterial so far as the results of the extra-solution are concerned. This was particularly noticeable at the Ontario, on roasted ore, the mill-extraction by the extra-solution, on ore with a very low chlorination, being 34 ounces per ton more than could be extracted by the ordinary solution, even in the assay-office (the extraction by the latter representing approximately the amount of silver present as chloride). Also at Yedras, the extra-solution in the mill extracted as high as 28 ounces per ton more than could be extracted by the ordinary solution even in the assay-office; and the difference was much greater, if mill-results were compared. At the Ontario, the mill-results by the extra-solution, on ore roasted with no salt, were only 4 per cent. less than by amalgamation on ore roasted with 18 per cent. salt, and on ore roasted with 9 per cent. salt were 3.5 per cent. higher than by amalgamation on ore roasted with 16 per cent. salt. Hence the Russell process can be used on ore with a very low chlorination, or on ore chloridized with less salt than would be necessary if the ordinary process or amalgamation were to be used. The applicability of the Russell process to remote regions is illustrated by the fact that over 90 per cent. of the leaching-plant, in bulk and weight, is of wood and can be constructed wherever wood can be obtained.

Its great applicability to dry regions, as compared with amalgamation, is shown by the fact that in the treatment of roasted ore leaching requires only one-sixteenth as much water as amalgamation, and in the treatment of raw ores only one-thirtieth as much as amalgamation.

VI. Comparative Applicability of Various Roasting Furnaces to the Preparation of the Ore for the Process.

A.—*Effect of a Very Short Muffle-Roast, With Free Exposure to Air and Heat, on the Extraction and Loss of Silver by Volatilization.*

Table XXVI shows the effect on Yedras ore of a very short roasting in the muffle. In these experiments the ore crushed in the battery through a 26-mesh screen was sifted quickly, or spread in a thin layer on an earthenware dish and exposed in the muffle to a medium red heat. The Table illustrates the fact, that, as far as the extraction by the extra-solution is concerned, the operation of roasting requires not more than five seconds, if each particle of the ore is freely exposed to the heat and furnace-gases at the same time, and that the loss in silver under such circumstances is practically nothing. On the other hand, deeper layers of ore roasted for half an hour to an hour showed a loss of 11 to 20 per cent., and in the case of concentrates as high as 40 per cent. In the experiments in Table XXVI, all the roasted ore was assayed, so that the determination of the percentage lost required no calculation. A still higher extraction in the same time was obtained on Solace ore, but no record of the loss of silver in these experiments was preserved. The above are the only two ores on which the quick roastings have so far been tried.

TABLE XXVI.

EXPERIMENTS ON VERY SHORT TIME OF ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS, YEDRAS ORE.

Thickness of Layer of Ore.	With or Without Stirring.	Per cent. of Salt Used.	Total Time of Roasting.	Value of Roasted Ore. Oz. Silver per Ton.	Per cent. Extracted by Extra-Solution.	Loss of Silver by Volatilizat'n. Per cent.
⅛ inch deep.	Stirring.	7	½ minute.	64.3	66.9	0.0
" " "	"	7	1 "	64.0	78.3	0.3
" " "	"	7	1½ "	64.3	75.3	0.0
" " "	"	7	2 "	65.0	....	0.0
" " "	"	7	2½ "	65.2	77.9	0.0
1/16 inch deep.	"	7	½ "	64.5	66.4	0.0
" " "	"	7	1 "	64.6	69.9	0.0
" " "	"	7	1½ "	64.6	74.5	0.0
" " "	"	7	2 "	64.8	77.6	0.0
" " "	"	7	2½ "	64.8	86.7	0.0
1/24 inch deep.	No stirring.	7	15 seconds.	64.5	86.1	0.0
" " "	" "	7	5 "	64.5	83.5	0.0
" " "	" "	3½	5 "	64.5	67.2	0.0
" " "	" "	0	5 "	64.5	56.9	0.0

B.—*Effect of Various Sizes of Crushing on the Furnace-Results.*

Table XXVII compares the Stetefeldt, Howell, reverberatory, and Brückner furnaces as to the results of using various sizes of screen. The table shows that the Stetefeldt furnace gives about as good results with a 16- as with a 20-mesh screen, and the Howell with a 12- as with a 26-mesh, but the reverberatory gives a little better results with an 8- than with a 16- or a 30-mesh, although the coarse part alone does not give as good results as the fine part alone. With the Brückner, the change from a 26- to a 10-mesh produces very little difference in the results. The table also gives the percentage of salt used in each case. The Ontario ore is of higher grade than the others, and has therefore a little more salt.

C.—*Differences in Extraction Between Ore From Furnaces and Ore From the Cooling-Floor or Vaults.*

The cooling-floor or vault-samples are the only ones which can be taken as a standard of comparison for furnace-work. Evidently if the percentage of extraction decreases as soon as the ore leaves the furnace, a high extraction from samples taken in the furnace counts

TABLE XXVII.

EFFECT OF DIFFERENT SIZES OF CRUSHING ON THE ROASTING IN VARIOUS FURNACES.

Name of Mine.	Kind of Furnace Used.	Mesh of Screen.	Value of Ore. Oz. Silver Per Ton.	Per cent. of Salt.		Place from which Sample was taken.	Per cent. Extraction by Extra-Solution in Assay Office.	
Ontario.	Stetefeldt.	30	75.1	9	11.7	Cooling-Floor.	94.6	95.2
"	"	20		12½		"	97.0	
"	"	16		13½		"	94.1	
San Antonio.	Howell.	26	47.0	10	10.0	"	89.2	89.1
"	"	12		10		"	89.0	
San Miguel.	"	26	53.0	8	8.0	"	91.9	90.5
"	"	12		8		"	89.1	
Sombrerete.	Reverberatory.	30	35.9	10	10.0	"	87.8	88.5
"	"	16		10		"	88.8	
"	"	8		10		"	90.5	
"	"	8*		10		"	91.8	
"	"	8†		10		"	83.6	
Yedras.	"	26	54.9	7	7.0	"	85.9	85.9
"	Brückner.	26	61.7	7	7.0	"	70.4	68.9
"	"	10		7		"	67.5	

* Fine half only of the screenings taken.
† Coarse " " " " "

TABLE XXVIII.

FURNACE-SAMPLES FROM BRÜCKNER FURNACE COMPARED WITH THE CORRESPONDING VAULT (OR COOLING-FLOOR) SAMPLES AT YEDRAS MILL.

FINAL SAMPLES FROM THE FURNACE.

No. of Charge.	Mesh of Screen.	Per cent. of Salt.	Per cent. extracted by Ordinary in Assay-Office.		Per cent. extracted by Extra in Assay-Office.		Difference. Per cent.	
115	10	7	32.5		84.1		51.6	
116	26	7	37.7		69.0		31.3	
117	26	5	57.0		68.8		11.8	
118	26	5	20.0		57.2		37.2	
119	26	7	32.7	50.4	63.5	70.4	30.8	20.0
120	26	7	40.8		57.0		16.2	
121	26	7	53.9		83.8		29.9	
122	26	7	76.8		82.4		5.6	
123	26	7	63.1		70.9		7.8	
124	26	3	54.3		74.4		20.1	
125	26	7	30.0		60.0		30.0	
126	26	7	51.4		71.4		20.0	
Total Average,			45.8		70.2		24.4	
Ave. of 119-126,			50.4		70.4		20.0	

CORRESPONDING SAMPLES FROM THE COOLING-FLOOR.

No. of Charge.	Mesh of Screen.	Per cent of Salt.	Per cent. extracted by Ord'y in Assay-Office.	Per cent. extracted by Extra in Assay-Office.	Difference. Per cent.
115	10	7	19.0	67.5	48.5
116	26	7	17.9	70.4	52.5
117	26	5	20.3	63.2	42.9
119 to 126	26	3 to 7	37.9	61.8	23.9
Average.........			32.8	63.2	30.4

for nothing. Such was the case at Yedras with the Brückner furnace. Table XXVIII, with this particular ore, and the Brückner furnace, shows that the percentage obtained from cooling-floor samples is 8 to 17 per cent. less than that from the final furnace-sample. On the other hand, the average of a month's run with the reverberatory shows a decrease of only 0.8 per cent. after leaving the furnaces.

The percentage of extraction from ore roasted in a Howell or Stetefeldt increases after the ore leaves the furnace. At Cusi the average increase in extraction from ore roasted in Howell furnaces was 26 per cent after the ore had left the furnace. At the Ontario, with a Stetefeldt, the average was 6 per cent. after leaving the furnace.

D.—*Effect on Brückner Furnace-Results of Introducing the Salt into the Furnace With the Ore, as Compared With Introducing Salt After the Roasting has Begun.*

The experiments in the Brückner furnaces at Yedras may be divided into two classes: first, those in which the salt was introduced into the furnace at the same time with the ore, and, secondly, those in which the salt was added during the roasting.

The first of the above sets of experiments was made particularly to test the applicability of the Brückner furnace to the roasting of the ore for the ordinary leaching process, as that was the process then in use and continued in use for the next six months, on ore roasted in reverberatory furnaces. For this reason, leaching-tests by the extra-solution were not made until afterwards, on vault-samples which had been saved. The average of the assay-office results on these vault-samples was 72.1 per cent. by the ordinary solution and 84.0 per cent. by the extra-solution. This does not include the ore from the dust-chambers, which yielded much less. All the ore produced in these experiments (except the dust) was then leached in the mill by the old or ordinary leaching process, with the results shown in Tables XXIX and XXX. These results in the mill, by the ordinary process, were so extremely poor that the attempt to use the furnaces for the ordinary process had to be abandoned. One great difficulty was the formation of balls of all sizes and of the hardness of brick. Some of these balls were too large to be extracted from the furnace until broken up with crowbars. The percentages of the various sizes of balls were as follows:

Description of the sizes.	Per cent. of each size.
Larger than base-balls,	22.2
Base-balls to marbles,	22.2
Marbles,	9.7
Large gravel,	3.1
Gravel,	42.7
Finer than gravel,	0.1
	100

TABLE XXIX.

THE REVERBERATORY AND BRÜCKNER FURNACES COMPARED BY THE MILL-RESULTS FROM THE OLD OR ORDINARY LEACHING PROCESS, AT THE YEDRAS MILL.

Duration of Mill Run.	In what Furnace the Ore was Roasted.	Value of Ore.	Per cent. by Ordinary in Assay-Office.	Per cent. by Extra in Assay-Office.	Oz. Silver per Ton in Mill-Tailings.	Extraction in Mill Per cent.	Time of First Washing with Water, Hours.	Time of Leaching with Solution. Hours.	Total Time in Hours.	Total Time in Days.
4 Days.	Reverberatory.	50.3 oz.	63.3	76.4	16.6	69.3	17	40	57	2.4
4 "	Brückner.	62.5 "	67.8	79.4	28.4	54.6	11	96	107	4.5
4 "	Reverberatory.	57.1 "	70.6	85.0	14.5	74.6	17	40	57	2.4
4 "	Brückner.	56.9 "	53.0	66.6	44.4	22.2	33	73	106	4.4
4 "	Reverberatory.	63.3 "	72.6	85.3	15.8	75.2	17	40	57	2.4
4 "	Brückner.	69.5 "	66.6	86.8	34.4	50.5	27	128	155	6.4
3 "	Reverberatory.	52.5 "	74.9	86.5	16.6	68.4	17	40	57	2.4
3 "	Brückner.	64.2 "	74.4	91.1	20.5	68.2	24	104	128	5.3
7 "	Reverberatory.	56.7 "	76.6	87.8	12.6	73.6	24	36	60	2.5
7 "	Brückner.	55.3 "	55.1	83.2	25.9	53.2	58	158	216	9.0
6 "	Reverberatory.	60.2 "	74.8	89.8	14.4	76.1	27	46	73	3.0
6 "	Brückner.	67.4 "	79.2	89.0	27.4	59.3	37	98	135	5.6
Aver'ge	Reverberatory.	56.7 oz.	72.1	85.1	15.1	72.8	19.2	40.3	60	2.5
	Brückner.	62.6 "	66.0	82.7	30.2	51.3	32.0	108.0	140	5.9

The second set of experiments, in which the salt was introduced into the furnace after the roasting had begun, showed that the formation of balls could be avoided, but only at the expense of a great decrease in extraction by the ordinary solution, and on considerable decrease in extraction by the extra-solution, even from the furnace-samples, to say nothing of the 8 to 17 per cent. decrease in extraction after leaving the furnace. Tables XXVII, XXVIII, and XXXI give the results obtained in the second set of experiments. Table XXVIII shows the average of the results on vault or cooling-floor samples for the second set of experiments, the average extraction in the assay-office being 32.8 per cent. by ordinary solution and 63.2 per cent. by the extra-solution. The extraction by the ordinary solution being so poor, and the Russell process not having then been introduced, this ore was not leached in the mill. Whether it was subsequently treated, after the Russell process was introduced, or re-roasted in the reverberatory furnace, is not known.

TABLE XXX.

COMPARISON OF THE NET EFFICIENCY OR ECONOMY OF REVERBERATORY AND BRÜCKNER FURNACES WHEN USING THE OLD OR ORDINARY LEACHING PROCESS, AT YEDRAS MILL.

Mill-Results on Ore from Reverberatory Furnaces.	
Average Value of Roasted Ore Per Ton (see Table XXIX)	56.7 oz.
Per cent. Extracted in Mill " " "	72.8
Ounces Per Ton Extracted in Mill	41.5
Number of Tons Per Charge	34
Number of Ounces Extracted Per Charge	1411
Average Time of Leaching Per Charge (see Table XXIX)	2.5 days
Number of Charges which could be treated per month	33
Ounces Silver Produced per month	46563
Mill-Results on Ore from Brückner Furnaces.	
Average Value of Roasted Ore Per Ton (see Table XXIX)	62.6 oz.
Per cent. Extracted in Mill " " "	51.3
Ounces Per Ton Extracted in Mill	32.4
Number of Tons Per Charge	34
Number of Ounces Extracted Per Charge	1102
Average Time of Leaching Per Charge (see Table XXIX)	5.9 days
Number of Charges which could be treated per month	14
Ounces Silver Produced per month	15428

E.—*Comparison of Reverberatory and Brückner Furnaces as to their Economy, and Leaching-Tests on Flue-Dust from Various Furnaces.*

Table XXIX shows the comparison between the results on ore roasted in a reverberatory and that roasted in a Brückner furnace at Yedras, for six periods amounting to 28 days. The table illustrates the difference in assay-office results on the charge-samples, in extraction in the mill by the old or ordinary leaching process, and in the total time required in leaching. The mill-results are in favor of the ore roasted in a reverberatory furnace, to the extent of an additional extraction of 15.1 ounces per ton or 21.5 per cent. of the value of the ore. In the time required in leaching the charges, the difference is in favor of ore roasted in a reverberatory by 3.4 days—the time for reverberatory ore being 2.5 days as against 5.9 days for ore roasted in a Brückner furnace. The rate of leaching on ore roasted in reverberatory furnaces and leached without using a vacuum under the filter was three times that of ore roasted in Brückner furnaces and leached with the help of a vacuum.

In Table XXX the above results are arranged in a different form, and illustrate the difference between the two furnaces as to their net efficiency or economy for the Yedras leaching-works, run to their utmost capacity with the ordinary process. The difference in favor of the reverberatory furnaces is about 31,000 ounces per month additional product, or 203 per cent. more than from ore roasted in Brückner furnaces.

Table XXXa gives statistics of reverberatory roasting at Yedras. The first section given in the table represents the average of 70 days, the second the average of 31 days. Further data of loss of silver in roasting may be found on page 109. Although the furnace losses are not absolutely correct, they are comparatively so.

The method of calculating the loss is so well known that it need not be described here. It is, of course, applicable only in cases where very little dust is formed. Otherwise, the proportion between the amount of ore and dust should be known. The only accurate method is to weigh and assay the ore sent to the furnaces and all the ore and dust obtained from them. Any other method is merely approximate at the best, and is useful only as a means of comparison. In the new mill now being erected by the Sombrerete Mining Co., by the manager of that company, Clarence King, and by W. R. Eckart of San Francisco, particular attention will be paid to the determination of the furnace-losses. The mill will have a screening capacity of 200 tons per day and a leaching capacity by the Russell process of

TABLE XXXa.

RESULTS OBTAINED BY ROASTING IN VARIOUS REVERBERATORY FURNACES AT YEDRAS. AMOUNT OF WOOD USED IN ROASTING, FURNACE-LOSSES AND RESULTS OF LEACHING-TESTS.

Date of Run.	Number of Furnace.	Kind of Furnace.	Cords of Wood per ton.	Per cent. Extracted by Ordinary in Ass'y Office.	Per cent. Extracted by Extra, in Ass'y Office.	Per cent. Loss by Volatilization in Roasting.
Section 1, before July 1st, 1887.	1 and 2.	5 Hearth Flat.	.17	76.3	85.8	13.5
	3	5 " Step.	.18	70.9	82.2	17.9
	4	5 " "	.18	72.5	83.1	15.6
	5	3 " Flat.	.25	76.8	86.2	14.8
	6	3 " "	.26	75.5	86.9	15.3
	7	4 " "	.19	78.7	86.9	14.1
	8	4 " "	.20	74.2	84.9	16.0
Section 2, since July 1st, 1887.	1 and 2.			75.5	86.6	7.5
	5			71.8	86.3	7.6
	6			72.7	86.5	8.8
	7			71.2	85.7	4.1
	8			63.4	85.8	3.8

150 tons. The ore will be crushed with rolls, dried in shelf dry-kilns, and roasted in the largest Stetefeldt furnace so far constructed. All the ore will be weighed and assayed before each operation in the mill, thus affording absolutely correct statistics.

Reverberatory furnaces have so far been used at Sombrerete, and careful statistics have been kept for a year and a half by Mr. Watson, superintendent of the mill. An opportunity will thus be afforded of completing the comparison given in this paper, between the Stetefeldt, Howell, and Brückner furnaces, respectively, and the reverberatory. This is the only kind of comparison (comparing each furnace with the reverberatory) which can probably ever be made, as an opportunity of direct comparison of all four furnaces on the same ore is not likely to occur.

Table XXXb gives the value and the results of assay-office leaching-tests, on samples from the dust-chambers of both the reverberatory and Brückner furnaces at Yedras. The "lower dust-chambers" are the nearest to the furnace. The dust from the reverberatories was much greater in value than that from the Brückner, but in amount was not more than $\frac{1}{10}$ as great. Table XXXc, furnished by Mr. Wilson, superintendent of the Marsaac mill, gives the values, results of assay office leaching-tests, and the comparative amounts of the ore and dust deposited in various parts of a Stetefeldt furnace and dust-chamber, at Park City. By changing the amount of draft, of course, the comparative distribution would be altered. Each of the dust-chambers, except the last, is 10 feet long by 12 feet high. The wing, or arch in the middle of each dust-chamber, reaches half-

TABLE XXXb.

ASSAY-OFFICE RESULTS ON ORE FROM DUST CHAMBERS OF THE REVERBERATORY AND BRÜCKNER FURNACES AT YEDRAS MILL.

	Reverberatory Furnaces.			Brückner Furnaces.			Brückner Furnaces.		
	Salt Added with Ore.			Salt Added with Ore.			Salt Added during the Roasting.		
	Value in Ounces.	Per cent. by Ordinary Solution.	Per cent. by Extra-Solution.	Value in Ounces.	Per cent. by Ordinary Solution.	Per cent. by Extra-Solution.	Value in Ounces.	Per cent. by Ordinary Solution.	Per cent. by Extra-Solution.
Lower dust-chambers......	144.2 oz.	46.7	45.8	55.9 oz.	75.1	84.3	72.9 oz.	36.2	66.7
Middle dust-chambers.....	130.5 "	69.1	88.5	57.8 "	63.7	85.3	70.2 "	51.8	76.7
Upper dust-chambers......	120.2 "	86.0	86.0	114.4 "	60.8	71.8	67.8 "	55.6	80.4

way down from the top, and the entrances and exits in the diagonally opposite upper corners are four feet square. The comparative weights given are the results of an 8 days' run. In passing through each dust-chamber the current containing the dust changes its direction 4 times, the whole number of right-angle changes of direction being 52. This is the important point, which is nearly always neglected in small mills and in many large ones. For instance, at Cusi the dust-chambers consist of a flue averaging about 5 feet wide, and probably 7 feet high, but with only about four right-angle changes of direction. The result is a dust-loss at Cusi probably 8 or 10 times as great as at Ontario, Daly, Lexington, Blue Bird, or anywhere else where a Stetefeldt furnace is used. This may be ascribed to the fact that no Stetefeldt furnace has ever been erected without suitable dust-chambers, which, however, are just as necessary for other furnaces as for a Stetefeldt.

TABLE XXXc.

DISTRIBUTION OF ROASTED PULP THROUGH A STETEFELDT FURNACE AND THE ASSAY-OFFICE RESULTS ON CORRESPONDING SAMPLES AT THE ONTARIO AND MARSAAC MILLS, PARK CITY, UTAH.

	Salt and Screen used.	Shaft.	Return Flue.	Hopper.	1st Dust-Chamber.	2d Dust-Chamber.	3d Dust-Chamber.	4th Dust-Chamber.	5th Dust-Chamber.	6th Dust-Chamber.	7th Dust-Chamber.	8th Dust-Chamber.	9th Dust-Chamber.	10th Dust-Chamber.	11th Dust-Chamber.	12th Dust-Chamber	13th Dust-Chamber.
Distribution by Weight.	16 pr ct. salt & 26 screen	59.9 pr ct.	21.0 pr ct.	3.3 pr ct.	2.7 pr ct.	2.7 pr ct.	2.5 pr ct.	1.3 pr ct.	1.0 pr ct.	0.8 pr ct.	0.6 pr ct.	0.6 pr ct.	0.6 pr ct.	0.5 pr ct.	0.4 pr ct.	0.4 pr ct.	1.4 rp ct.
Assay Value in Ounces.		33.6	40.6	42.6	41.1	45.5	48.5	49.1	49.7	49.2	52.2	52.6	54.8	50.8	50.8	53.0	51.
Per cent. by Ordinary in Assay Office.		82.8	89.7	91.3	92.3	92.7	94.1	92.7	93.3	92.2	93.5	91.8	91.6	91.8	88.6	88.6	86.0

The great capacity of the Stetefeldt furnace for roasting has been noticed. The results of experiments at the Ontario with this furnace show that with a 16-mesh screen and 17 per cent. salt, and roasting at the rate of 70 tons per day, the extraction on the large scale was 97 per cent. The capacity of the furnace could not be tested further, as only 20 stamps are connected with each furnace and 70 tons was the utmost which could be crushed. It is no more than reasonable to suppose that at least 90 per cent. could have been obtained from ore roasted at the rate of 80-90 tons per day.

The average amount of pulp treated in one Stetefeldt furnace at the Marsaac mill is 70 to 74 tons per day. In this case also, this is the limit of the crushing, and not of the roasting capacity.

F.—*Comparison Between the Howell and the Reverberatory at Cusi.*

The reverberatory furnaces formerly used at Cusi were replaced by Howell furnaces, and the leaching results may be thus compared:

	Mesh of Screen.	Percentage of Salt.	Time of Roasting.	Result by Ordinary Solution in Assay-Office.
Reverberatory..........	30 to 40	5 to 6	18 hours.	80 per cent.
Howell's....................	12 to 20	8 to 10	35 minutes.	85 "

G.—*Comparison of the Four Roasting-Furnaces as to the Time Required in Roasting.*

Table XXXI shows the difference in time of exposure to the fire required by the different furnaces to produce the given results by

TABLE XXXI.

COMPARISON OF STETEFELDT, HOWELL, REVERBERATORY AND BRÜCKNER FURNACES WITH REGARD TO TIME REQUIRED IN ROASTING AND THE RESULTS OBTAINED.

Name of Mine and Kind of Furnace Used.	By which Process Leached.	STETEFELDT		HOWELL.		REVERBERATORY AND BRÜCKNER.				Per cent. after Lying in Vault or on Floor.
		Per cent. Extraction after 5 Seconds in Furnace.	Per cent. after Lying on Cooling Floor.	Per cent. after 35-45 Minutes in Furnace.	Per cent. after Lying in Vault.	Per cent. after 2-4 Hours in Furnace.	Per cent. after 5-7 Hours in Furnace.	Per cent. after 8-10 Hours in Furnace.	Per cent after 11-15 Hours in Furnace.	
Ontario; Stetefeldt	Russell.	87.5	93.5							
	Ordinary	71.9	87.2							
Cusi; Howell	Russell.			65.6	90.2					
	Ordinary			28.5	78.1					
Sombrerete; Reverberatory	Russell.					29.1	39.4	55.4	90.0	About the same.
	Ordinary					9.4	11.3	14.1	80.9	
Yedras; Reverberatory	Russell.					68.3	64.6	75.1	86.7	$\frac{8}{10}$ of 1 per cent. less.
	Ordinary					36.5	39.1	58.9	67.9	
Yedras; Brückner	Russell.					71.8	76.4	74.7	80.2	8 to 17 per cent. less.
	Ordinary					49.6	44.3	46.0	36.3	

both extra and ordinary. The times of exposure to the fire, and corresponding final results by the extra-solution are as follows:

Kind of Furnace.	Time of Roasting.	Extraction by Extra-Solution.	
		Furnace Sample.	Cooling Floor.
Stetefeldt......................	5 seconds.	87.5 per cent.	93.5 per cent.
Howell........................	35 to 45 minutes.	65.6 "	90.2 "
Reverberatory..............	11 to 15 hours.	88.3 "	88.3 "
Brückner....................	11 to 15 hours.	80.2 "	80.2 "

Although higher extraction from furnace-samples can undoubtedly be obtained on some ores, with a Howell furnace in thirty-five to forty-five minutes, or in a Brückner furnace in five to six hours, yet the table shows that, as far as the Russell process is concerned, roasting is, in a Stetefeldt furnace, an operation of only a few seconds—in a Howell, of minutes—but in a reverberatory or Brückner it requires hours.

H.—*General Comparison of the Four Roasting-Furnaces.*

a. Comparison of Reverberatory and Brückner.—The comparative merits of the two furnaces, as illustrated at Yedras, were as follows:

1. *As to the Mechanical Leaching of the Ore.*—In the case of the reverberatory furnaces, whether the salt was added with the ore, or afterward, no hard balls were formed, and the leaching charges remained soft, and were rapidly leached. In the case of the Brückner furnace, the addition of salt with the ore was disastrous, not only converting the ore almost entirely into hard brick-like balls, but producing a hardening of the charge as soon as liquid was turned upon it, so that the mass could not be removed except by picks, thus rendering the leaching extremely slow.

2. *As to Extraction,* the addition of salt with the ore in the reverberatory furnaces produced about the same result as when added in the furnace. But with the Brückner, when salt was added with the ore, the mill-extraction was 17.0 per cent. less than on ore roasted in the reverberatory.

3. *As to Extraction again,* the decrease in extraction from reverberatory ore, after leaving the furnaces, was only 0.8 per cent. no matter when the salt had been added, but the extraction from the Brückner ore, when salt had been added in the furnace, decreased from 8 to 17 per cent. after leaving the furnace.

4. *The Time of Leaching Brückner ore* was 2.3 times as long as the leaching of ore roasted in the reverberatory.

5. *The Mill-extraction from Brückner ore* averaged 21.5 per cent. of the value of the ore less than the extraction from ore roasted in the reverberatory.

6. *The gross product per month* of the Yedras leaching-works, from ore roasted in reverberatories, was at the rate of three times that from ore roasted in the Brückners.

b. Comparison of the Howell and Reverberatory.—The only direct comparison available between these two furnaces was at Cusi, the results of which have already been given. The Howell used 3 to 5 per cent. more salt, but, using a 12- to 20-mesh screen, yielded 5 per cent. higher extraction than the reverberatory with a 30-mesh screen, and in less than one-thirtieth of the time required by the reverberatory. At Cusi was also illustrated, indirectly, the disastrous mechanical effect of the roasting in a Brückner furnace on the leaching of the ore; indirectly, because the one ore, roasted in the Brückner furnace, hardened during the leaching, while, on the other hand, the ore from all the other three producing mines in the camp, roasted in Howell and reverberatories, gave no such trouble, and at the same time yielded a higher percentage of the silver.

c. Comparison of the Stetefeldt with the Other Three Furnaces.—In *a* and *b*, Brückner and Howell furnaces have been compared with reverberatories. The only opportunity of making the same comparison for the Stetefeldt will be at Sombrerete, Zacatecas, Mexico; where reverberatories have thus far been in use, and where a Stetefeldt is now being erected. The results of the very short roasting in the muffle on Yedras ore are, however, a fairly reliable indication that very high results could be obtained in the Stetefeldt with that ore. It should be recollected, also, that it is a peculiarity of the Stetefeldt, as compared with the reverberatory or Brückner, that its effect upon the ore is not limited to the time the ore remains in the furnace, but the extraction from ore roasted in it continues to rise for twelve to fourteen hours after the ore has been taken from the furnace, and at no additional expense. In short, the advantages in favor of the Stetefeldt furnace, as compared with the Howell, reverberatory and Brückner, in regard to the preparation of ore for the Russell process are as follows:

1. *In Capacity.*—The Stetefeldt has a capacity of 70 to 100 tons per day, while that of a Howell rarely reaches 40 tons, a Brückner 20 tons, and a reverberatory 10 tons per day.

2. *In a Better Condition of the Roasted Ore for Rapid and Thorough Leaching.*—As illustrated at Yedras and other places, the effect of a long-continued rolling motion, as in the Brückner, not only often results in the formation of balls, but in a hardening of the leaching-charge, like cement, as soon as water or solution is turned upon it, which was not the case with the reverberatory or the Howell, and never has been with a Stetefeldt furnace.

3. *In Fuel.*—The amount required for roasting in a Stetefeldt is about one-tenth of a cord of wood per ton of ore, while for the Howell it is one-fourth to one-third, and for the reverberatory one-sixth to one-fourth of a cord.

4. *In Power.*—The amount required to run a Stetefeldt is only that required to shake the screen at the top of the shaft, which is probably not more than one-thirtieth of that required to run a Brückner, or one-fifteenth of that required for a Howell furnace, per ton of ore-capacity.

5. *In Extraction.*—Table III shows that the average extraction from ore roasted in the Stetefeldt is 4.2 per cent. above that roasted in the Howell, 4.8 per cent. above the reverberatory, and 6.9 per cent. above that roasted in the Brückner.

6. *In Time of Roasting* (*of Exposure to the Fire*).—Table XXXI shows the time required to obtain approximately the given results.

VII. COMPARISON OF THE RUSSELL PROCESS WITH THE ORDINARY LEACHING PROCESS; WITH AMALGAMATION; AND WITH COMBINED AMALGAMATION AND CONCENTRATION.

In discussing the merits of different processes, the rank which any of them shall hold should be based solely on its value *as compared with the others*, not merely on its economic success on any particular material, which is dependent on many conditions entirely outside of the process itself.

A.—*Comparison with the Ordinary Leaching-Process.*

a. At the Ontario Mill, Park City, Utah.—The results of these experiments, Table XXXII, illustrate, not only the greater extraction of silver by the extra-solution, but also the injurious effect of caustic alkali entering the ordinary solution from the precipitant, and show separately the beneficial effects of the use of the acid and of bluestone, which constitute the most important chemicals peculiar to the

Russell process. In these tests, which were made on a scale of 2 to 3 tons each, the first wash-water being acid, the caustic alkali could not have entered the solution from any other source than the precipitant. The first set of experiments in Table XXXII shows an assay-office extraction by ordinary solution, of 89.0 per cent., but a mill-extraction by ordinary solution of 81.8 per cent. Samples of the mill-tailings were then leached in the assay-office with ordinary solution, which brought the extraction up to only 82.8 per cent. instead of 89 per cent., as it would have been if some of the silver, which had originally been soluble in ordinary solution, had not been rendered insoluble during the leaching in the mill. This insoluble percentage was 6.2 per cent. of the value of the ore.

TABLE XXXII.

COMPARISON OF THE RUSSELL PROCESS WITH THE OLD OR ORDINARY LEACHING PROCESS AT THE ONTARIO MILL, IN 1883-4 (ACID-ROASTED ORE).

Value of Ore. Oz. Silver per Ton.	Per cent. extracted by Ordinary Solution in Assay-Office.	Per cent. Extracted by Extra-Solution in Assay-Office.	Per cent. Extracted in Mill.	Per cent. of Silver Made Insoluble in Ordinary Solution by Caustic Alkali.	Difference Between Extraction in Mill and Extracti'n by Extra-Solution in Assay-Office. Per cent.
AVERAGE RESULTS OF THE FIRST FOUR MILL-TESTS WITH THE ORDINARY PROCESS. (NEITHER ACID NOR EXTRA-SOLUTION WAS USED.)					
85.0	89.0	90.5	81.8	6.2	8.7
AVERAGE RESULTS OF THE SECOND FOUR MILL-TESTS WITH THE ORDINARY PROCESS. (NEITHER ACID NOR EXTRA-SOLUTION WAS USED.)					
86.5	84.3	92.5	66.2	14.6	26.3
AVERAGE RESULTS OF THE NEXT TEN MILL-TESTS, IN WHICH ACID WAS USED BUT NO EXTRA-SOLUTION.					
90.4	82.6	90.0	82.5	0.0	7.5
AVERAGE RESULTS OF SUBSEQUENT MILL-TESTS, IN WHICH BOTH ACID AND EXTRA-SOLUTION WERE USED.					
72.0	87.0	92.6	92.5	0.0	0.1

During the next four tests, in Table XXXII, the caustic alkali in the stock-solution had increased to such an extent as to cause the mill-extraction to fall to 66.2 per cent., or 18.1 per cent. below the extraction by ordinary solution in the assay-office. By leaching samples of these tailings in the assay-office, this was raised to 69.7 per cent., which still left 14.6 per cent. discrepancy, the amount of silver rendered insoluble during the leaching in the mill.

Up to this time no part of the Russell process had been used.

The whole stock-solution was now rendered neutral by the addition of sulphuric acid, the amount of acid used to neutralize all caustic alkali which had so far accumulated, being 0.1 per cent. by weight of the stock-solution. Always afterward, the stock-solution was kept neutral by the addition of acid to the extent of from three-fourths to one pound per ton of ore treated. For the next ten tests after the neutralizing of the caustic alkali, and the use of one pound of acid per ton, the extraction in the mill by the ordinary solution varied from the ordinary in the assay-office only 0.1 per cent., showing that caustic alkali no longer had any effect. By comparing these figures with the preceding, it will be seen that the use of an amount of acid costing only three to four cents per ton of ore has raised the mill-extraction 13 to 15 ounces per ton. But the mill-results, although equal to the ordinary in the assay-office, were 7.5 per cent. less than by the extra in the assay-office. The extra-solution was then used in the mill in addition to the acid, with the results shown in the last line of the table. Comparing this with the second line of figures, we see that the Russell process (both acid and extra-solution) made a difference in the mill of 26.2 per cent. which, on 86.5 oz. ore, is 22.6 oz. per ton. As the extra-solution acts as a neutralizer of caustic impurities, less acid is required when the extra-solution is used. On most ores the use of the acid for this purpose can thus be entirely avoided.

TABLE XXXIII.

COMPARISON OF THE RUSSELL PROCESS WITH THE OLD OR ORDINARY LEACHING PROCESS AT SILVER CITY, N. M. (RAW TAILINGS ALREADY TWICE AMALGAMATED AND ONCE CONCENTRATED).

Fineness of the Tailings — Per cent. Passing a Screen of 22,500 Holes per sq. inch.	Average Rate of Leaching in Inches per Hour with vacuum of 14″ of Mercury.	Value in Ounces Silver per Ton.	Percent. Extracted by Concentration in Assay-Office.	Percent. Extracted by Amalgamation in Assay-Office.	Percent. Extracted by Ordinary in Assay-Office.	Percent. Extracted by Extra in Assay-Office.	Percent. Extracted by Ordinary in Mill.	Percent. Extracted by Ordinary and Extra-Solution in Mill.
87.8	0.667	11.6	0	22	42.5	64.7	38	61.7

b. At Silver City, New Mexico (*Raw Tailings*).—Table XXXIII gives the average results on the raw Bremen tailings at Silver City for first month's run. The average difference between the results of the ordinary process in the mill and the extra-solution in the mill, was in favor of the extra-solution by 23.7 per cent. of the value of the material treated. The second month, the average difference was 20.9 per cent., and the third month 25.9 per cent. Afterwards, the tailings became too fine for any further leaching, the average rate per hour for the second month being only one-seventh of an inch, and for the third month one-tenth of an inch per hour.

c. *At Lake Valley, New Mexico.*—Table XXXIV gives the average mill-results by the ordinary leaching-process and by the Russell process (two methods). The Table illustrates not only the difference between the results of the ordinary and the Russell process in the mill, but, also, the difference in results caused by using a strong, warm extra-solution with circulation, after the ordinary, as if the ore were an acid ore, as compared with the results of using a cold, weak extra-solution, before the ordinary and without circulation. As shown in the table, the results of the mill-tests were 30 per cent. of the value of the ore in favor of the Russell process, as compared with the ordinary process; although the difference between the ordinary and extra in the assay-office was only 10.5 per cent. The mill-workings by ordinary solution were below the assay-office results by ordinary solution on account of the injurious effect of the caustic lime in the roasted ore. No separate clean-up was made for the mill-tests by ordinary solution. The clean-up from the Russell process averaged 0.5 to 1.5 per cent. short of that called for by the "apparent" extraction. Although the mill was built for a capacity of 80 tons per day, and the capacity of the leaching part of the plant was shown to be 80 to 100 tons, the crushing part of the plant failed to crush more than 20 tons, thus limiting to that figure the amount which could be treated. The average value of the ore treated for

TABLE XXXIV.

COMPARISON OF THE RUSSELL PROCESS WITH THE ORDINARY LEACHING PROCESS AND WITH AMALGAMATION, AT LAKE VALLEY, NEW MEXICO (ALKALINE-ROASTED ORE).

Per cent. Extraction by Ordinary in Assay-Office.	Per cent. Extraction by Extra-Solution in Assay-Office.	Per cent. Extraction by Amalgamat'n in Mill.	Per cent. Extraction by Old Ordinary Process in Mill.	Per cent. Extraction by Russell Process in Mill if Extra was Used after the Ordinary.	Per cent. Extraction by Russell Process in Mill Using a Weak Extra before the Ordinary.
71	81.5	70.6	53	74.4	83

the first two months was about 17 ounces per ton, and for the last 5 months about 12 ounces. Although the expenses per ton for 20 tons per day were large compared to what they would have been for 80 tons per day, yet this mill continued to run on 12-ounce ore until the supply of ore was exhausted.

d. At Sombrerete, Zacatecas, Mexico.—Table XXXV gives the results of the full course of preliminary mill-tests made at Sombrerete to determine the applicability of the Russell process, the ore being roasted in reverberatory furnaces and leached in charges varying from 325 pounds to 4.5 tons. The first three leaching-tests were made on very fine and pure pyrites alone. On account of its fineness, and not containing any gangue, it became matted during the roasting. Hence the mill-results by the extra-solution fell short of the extra in the assay-office, by 5.9 per cent. The fourth set of mill-tests illustrates the injurious effect, on the results by extra-solution, of wetting down the roasted ore while it is red-hot, the mill-results by the extra-solution falling 4 per cent. below the results in the assay-office by the extra-solution.

TABLE XXXV.

COMPARISON OF THE RUSSELL PROCESS WITH THE OLD OR ORDINARY LEACHING PROCESS AT SOMBRERETE, ZACATECAS, MEXICO (ACID-ROASTED ORE).

Description of Charges.	Weight of Charge.	Value in Ounces Silver per Ton.	Per cent. Extraction by Ordinary Solution in Assay-Office.	Per cent. Extraction by Extra-Solution in Assay-Office.	Per cent. Extraction by Russell Process in Mill.
Finest Pyrites from the Jigs, crushed through an 8-screen—3 Charges	325 lbs.	75.8	73.5	90.9	85.0
Coarse and Fine Jig-Products mixed half and half, crushed through an 8-screen—1 Charge	325 lbs.	65.4	67.4	91.3	90.8
Coarse Jig-Products, crushed through an 8-screen—5 Charges	325 lbs.	33.4	76.7	86.2	86.5
Normal Mixture of Jig-Products, crushed through an 8-screen—4 Charges	325 lbs.	44.9	80.6	90.6	91.3
Normal Mixture of Jig-Products, wet down while red hot—3 Charges	4½ tons.	35.4	77.5	84.6	80.6
Normal Mixture of Jig-Products, *not wet down while red hot*—5 Charges	4½ tons.	36.5	74.0	89.3	88.0

At the time of making all the above tests, the mill-results by the ordinary solution were averaging very much below the results by the ordinary solution in the assay-office, so that the actual difference in the mill-results in favor of the Russell process, as compared with the ordinary process, averaged 22.6 per cent. of the value of the ore. The difference in actual clean-up in silver was a little greater, as the clean-up from the ordinary process fell a little short, while the clean-up from the Russell process was slightly in excess of that called for by the apparent extraction. Since Mr. Watson has had charge of the Sombrerete mill both the mill and the assay-office results have been very much higher.

e. At Cusihuiriachic, Chihuahua, Mexico.—Table XXXVI gives the comparison of mill-results at Cusi by the Russell process and the ordinary leaching-process, during a period of 9 months. The "mixed" months, referred to in the table, are those in which the extra-solution was used on a portion (about one-half) of the charges. During the other months either the extra-solution alone or the ordinary process alone was used for the whole month. The comparison between the two processes is in favor of the Russell process, as follows:

By the use of the extra-solution the time of leaching is reduced 34.8 per cent.; the apparent extraction is increased 6.5 per cent.; the expenses per ton are reduced $1.29; the net mill-profits per ton are nearly doubled; the gross mill-product per month is more than doubled, and the net mill-profits per month are more than trebled. It

TABLE XXXVI.

THE RUSSELL PROCESS COMPARED WITH THE OLD OR ORDINARY LEACHING PROCESS, AT THE CUSI MILL. COMPARISON OF "ORDINARY," "MIXED" AND "RUSSELL PROCESS" MONTHS FOR NINE MONTHS (SEPT. 12, 1886, TO JUNE 1, 1887).

Months and Process Used.	"Ordinary," "Mixed" or "Russell" Process Months.	Value of Roasted Ore in Ounces Silver per Ton.	Average Total Leaching Time. Hours.	Per cent. Apparent Extraction in Mill (by Tailings).	Value of Product in Ounces per Ton.	Total Mill-Expenses per Ton, Leached.	Total Mill-Product per Month.	Net Mill-Profit per Ton.	Net Mill-Profit per Month.
Jan. 1887... Ordinary... Feb. 1887... Ordinary...	"Ordinary" Months.	35.1	66	78.3	8.681	$13.37	$19,900	$ 7.81	$ 7,336
Sept. 1886.. ⅓ R. P...... Oct. 1886... ¾ R. P...... Mar. 1887... ⅜ R. P...... May, 1887.. ½ R. P......	"Mixed" (or part Ordinary and part Russell Process) Months.	36.8	46	82.3	7.985	12.62	27,690	12.42	13,724
Nov. 1886... R. Process Dec. 1886... R. Process Apr. 1887... R. Process	"Russell Process" Months.	39.9	41	84.8	8.201	12.08	42,233	15.11	23,436

will be noticed that the ore during the months when the extra-solution was used was 4 to 5 ounces higher in value than while the ordinary process was in use. Making corrections accordingly, *i. e.*, reducing the value of the ore to 35.1 ounces, the comparison would be in favor of the Russell process by 50 per cent. greater net profits per ton, 86.2 per cent. additional gross product per month and 149 per cent. additional net profits per month.

During the nine months referred to a discrepancy nearly always existed between the apparent and the actual extraction, both while using the ordinary and the Russell process. Part of this (3.2 per cent.) was due to volatilization and loss by dust in roasting the sulphides. This was remedied by substituting steam-drying for roasting. Another portion (2.8 per cent.) was found to be due to incorrect assaying of the sulphides. Another portion was due to incorrect determination of the weight of ore leached. Except in the special tests given in Table XXXVII, the weight of ore actually leached could not be accurately determined, as there were no scales for weighing the ore, and practical difficulties existed in the way of introducing them for general use. The only means of approximately determining the weight of ore treated was by the cubic-foot system, which consists in calculating the weight of the charge from the weight of one cubic foot of the ore from each charge and the number of cubic feet in the charge. Careful experiments showed that this method gave 6 per cent. more ore than was actually present in the charge. But this approximate method was better than none. At intervals careful experiments were made to determine whether any discrepancy really existed between the apparent (*i. e.*, the extraction calculated from the value of the tailings as compared with the value of the ore) and the actual extraction (*i. e.*, the product in silver). In all, six of these tests were made during a period of nine months, the weight of ore used each time, varying from 18 to 94 tons, being carefully determined. All these tests are given in Table XXXVII, of which No. 2 was by the ordinary process. The total amount of ore treated in these special tests was 279.1 tons. The average "actual" extraction (or clean-up) was 87.9 per cent., which is 1.7 per cent. above the "apparent" extraction. No. 2 (of the 6 special tests) shows both an apparent and an actual extraction of over 79 per cent. by the ordinary process.

On San Antonio ore, or on a mixture of San Antonio and San Bartolo, the extraction by the ordinary process averaged much less than this, the difference in extraction between the ordinary and the Russell

TABLE XXXVII.

SIX SPECIAL MILL-RUNS AT CUSI TO DETERMINE THE AMOUNT OF DISCREPANCY BETWEEN "APPARENT" AND "ACTUAL" EXTRACTION IN THE MILL.

No. of Mill-Run.	Kind of Ore Treated.	Mesh of Screen on Battery.	Per cent. of Salt Used.	Total Weight of Ore Leached. Tons.	Value of Ore in Ounces per Ton.	Per cent. Extraction by Ordinary in Assay-Office.	Per cent. Extraction by Extra in Assay-Office	Apparent Extraction in Mill. Per Cent.	Actual Extraction in Mill. Per Cent.	Condition of Sulphides.
1	San Bartolo and San Antonio.	16	9	19.6	39.8	85.5	92.0	85.3	84.2	Roasted.
2	San Bartolo and San Antonio.	16	10	18.7	38.4	83.4	91.9	79.7	79.2	Furnace-Dried.
.3	San Bartolo and San Antonio.	12	11	46.8	83.9	84.8	91.6	82.6	87.9	Furnace-Dried.
4	San Bartolo and San Antonio.	12	8	94.	41.7	82.7	90.4	85.9	86.5	Steam-Dried.
5	San Bartolo and San Antonio.	26	10	50.	46.9	85.0	89.2	88.5	91.5	Steam- and Furnace-Dried.
6	San Miguel.	26	8	50.	53.0	89.0	91.9	90.4	91.8	Steam- and Furnace-Dried.
	Average.					84.8	90.8	86.2	87.9	

NOTE.—The Second Mill-Run was by the Ordinary Leaching Process.

process being 15 to 34 per cent. of the value of the ore, as shown in Table XXXVIII.

This set of special mill-runs was made to determine the exact difference caused by the use of the Russell process as compared with the ordinary leaching process, both in regard to the value of tailings and in actual product in silver. The ore was mixed—San Antonio and San Bartolo. In order to get the best possible results by the ordinary process, the percentage of salt used in chloridizing was increased, and each charge of ore was leached continuously (some of them for five or six days) by the ordinary process, until for two consecutive days the tailings remained the same and it was evident that no further reduction in the value of the tailings could be accomplished by the use of the ordinary process. The extra-solution was then used on the same charges, and reduced the value of the tailings to the extent of 9.8 ounces per ton. The increase in the percentage extracted, due to the use of the Russell process, was 22.4

per cent. of the value of the ore. The increase in "actual" extraction (product in silver) due to the use of the extra-solution exceeds the amount called for by difference in tailings to the extent of 2.2 per cent. in dried sulphides and 0.8 per cent. in roasted sulphides. The difference caused by the use of the extra-solution was 22 to 24 per cent. both in apparent and actual extraction.

TABLE XXXVIII.

SPECIAL MILL-RUNS BY RUSSELL PROCESS ON CHARGES ALREADY TREATED BY THE ORDINARY OR OLD LEACHING PROCESS, AND ON WHICH THE VALUE OF THE TAILINGS COULD NOT BE FURTHER REDUCED BY THAT PROCESS.

Number of Charges Treated.	Number of Tons Treated.	Value of Ore in Ounces per ton.	Per cent. of Salt Used.	Per cent. Extracted by Ordin'y in Assay-Office.	Per cent. Extracted by Extra in Assay-Office.	Mill Tailings after Ordin'y Process and before Russell Process.	Mill Tailings after Russell Process.	Mill Per cent. after Ordin'y Process and before Russell Process.	Mill Per cent. after Russell Process.	Difference bet. Ordinary and Russell Process in Mill in oz. per Ton.	Difference in Per cent. of Value of Ore.	Amt. of Silver called for by Difference in Tailings. Total Oz.	Silver obtained in Furnace-Dried Sulphides. Ounces.	Silver obtained in Roasted Sulphides. Ounces.	Per cent. Discrepancy based on Dried Sulphides.	Per cent. Discrepancy based on Roasted Sulphides.
51	450	44.0	11.2	81.2	......	13.4	6.5	69.6	85.2	6.9	15.6	No separate clean-up in sulphides				
11	97	40.5	10.0	78.1	89.6	15.0	7.2	63.0	83.0	7.8	19.2	"	"	"	"	"
11	82	39.3	12.6	82.5	88.5	15.6	7.8	61.1	81.0	7.8	19.9	640.	747.8	732.	+3.3	+2.9
19	174	48.3	13.5	72.7	86.8	24.5	7.7	49.3	84.1	16.8	34.8	2922.	2890.0	2860.	—1.1	—2.1
92	803	43.0	11.8	78.6	88.3	17.1	7.3	60.7	83.3	9.8	22.4	3562.	3637.8	3592.	+2.2	+0.8

f. At Yedras, Sinaloa, Mexico.—Table XXXIX gives the comparison in mill-results of the extra-solution with the ordinary leaching process for a period of two months at Yedras. This was with an experimental run, the Russell process being used on charges of only one ton each. The difference in mill-results is 18.1 per cent. or 11.2 ounces per ton in favor of the Russell process. Table XL gives the

TABLE XXXIX.

COMPARISON OF MILL-RESULTS BY RUSSELL PROCESS AND BY THE ORDINARY LEACHING PROCESS IN THE YEDRAS MILL FOR A PERIOD OF TWO MONTHS.

By what Process Treated.	Value in Ounces per Ton.	Per cent. by Ordinary in Assay-Office.	Per cent by Extra in Assay-Office.	Mill Per cent.
Ordinary Leaching Process	60.5	68.9	82.7	64.5
Russell Process..............................	63.5	70.6	83.1	82.6

comparison between the extra-solution and the ordinary leaching-process as to mill-results, expenses and net profits per ton.

Run No. 1 is the one referred to in Table XXXIX. As no exact record of additional expenses was kept for this run, the expenses are taken the same as in Run 2, in which the record of expenses was carefully kept.

The second run gives the comparison of the ordinary process for five months previous to November 1st, 1887, with the Russell process for the three weeks between November 1st and 22d, the process not having been introduced in the mill until November 1st, 1887. Yedras being so remote and inaccessible, the price of chemicals per pound is very high, the average price being 10.5 cents per pound as compared with 4.75 cents in the mining camps of the United States. The weight of chemicals used per ton is 13 pounds for the ordinary process and 6.1 pounds additional for the Russell process. The average of the runs shows the superiority of the extra-solution to the extent of $9.48 gross profit or $7.83 net profit per ton.

TABLE XL.

COMPARISON OF THE RUSSELL PROCESS WITH THE ORDINARY LEACHING PROCESS OF THE YEDRAS MILL. COMPARATIVE MILL-RESULTS, EXPENSES AND NET PROFITS.

No. of Mill Run.	Value of Ore. Oz. Silver per Ton.	Ordinary in Mill short of Ordinary in Assay-Office. Oz. per Ton.	Ordinary in Mill short of Extra in Assay-Office. Oz. per Ton.	Extra in Mill above Ordinary in Assay-Office. Oz. per Ton.	Extra in Mill short of Extra in Assay-Office. Oz. per Ton.	Gross Difference per Ton in favor of the Russell Process. Oz.	Additional Cost per Ton of Russell Process.	Net Difference per Ton in favor of Russell Process.
1st.	60–63	2.7	11.3	7.44	0.13	10.10	$1.65	$8.45
2d.	55–64	3.7	9.0	5.33	0.17	8.86	1.65	7.21

Table XLa gives the comparison for the months of November and December, 1887, the first two months during which it was introduced at that mill. All tests previously made had been on a scale of only one or two tons at a charge. During the above two months the plant for the Russell process was still in process of construction and all the ore crushed and roasted could not therefore be treated by the Russell process, the excess over the capacity of the unfinished plant being treated in the old plant by the ordinary process, thus affording a valuable opportunity of comparing the two processes. In constructing the new plant very large tanks could not be built owing to the difficulty of obtaining large timber. The leaching-tanks are 11.8 feet diameter and 5.2 feet deep, and hold about 15 tons of roasted ore. The ore was crushed by stamps through a 20-mesh screen, and roasted in reverberatories with 7 per cent. salt. All efforts to roast the ore in Brückner furnaces were entirely unsuccessful, and these furnaces (four of which had been erected at considerable expense) have been permanently abandoned. As shown in the tables the mill-extraction at Yedras is not as high as in some other mills. This is due to the fact that the roasting in the reverberatories, owing to the large loss of silver by volatilization in these furnaces, must be conducted with reference to this loss, as well as to the extraction by leaching. The roasting can be so conducted as to yield 90 per cent. to 92 per cent. instead of 83 per cent. of the value of the roasted ore, by leaching; but such a roasting would cause a loss of about 18 per cent. or 20 per cent. by volatilization, instead of 7 per cent.

In another section are given the losses in silver for each of the reverberatory furnaces at Yedras under the management of R. D. Rhodes, the mill superintendent. Considering the long exposure (12 to 16 hours) necessary in roasting in reverberatory furnaces the losses by volatilization are extraordinarily low.

Table XLa gives the comparison of the Russell process and old process at Yedras for November and December, not only as to assay-office and mill-extraction, but also as to the amounts of chemicals (calcium sulphide being used for the ordinary, and sodium sulphide for the Russell process) and the additional gross and net saving due to the use of the Russell process, the gross saving being

TABLE XLa.

COMPARISON OF THE RUSSELL PROCESS WITH THE ORDINARY LEACHING PROCESS, AT YEDRAS, FOR NOVEMBER AND DECEMBER, 1887.

Month.	By which Process Treated.	Value of Roasted Ore per Ton.	Per cent. by Ordinary in Ass'y Office.	Per cent. by Extra in Assay Office.	Per cent. Extraction in Mill.	Weight of Chemicals Used.	Additional Extraction per Ton by Russell Process.	Additional Net Saving per Ton by Russell Process.
November.	Ordinary	56.9 oz.	69.4	82.0	63.7	11.0 lbs.	9.85 oz.	$8.20
	Russell.	55.7 "	70.4	82.6	81.2	20.5 "		
December..	Ordinary	61.15 "	71.9	84.0	67.3	17.4 "	8.89 "	$7.41
	Russell.	58.15 "	67.9	83.8	82.2	21.1 "		

$9.37 and the net $7.80 per ton. In all clean-ups, so far, the actual extraction in the mill has slightly exceeded the apparent. The value of the sulphides from the Russell process averages 15,000 to 16,000 ounces per ton, with a small amount of gold, the ore containing only a trace of gold. The amount of first wash-water averages 32, of ordinary solution 100, and of extra 14.5 cubic feet per ton.

This amount of extra-solution is unusually large, as the Yedras ore requires more liquid to saturate it than any other.

The rate of leaching is 7 inches per hour, and the strength of stock-solution is maintained at 1.8 per cent. although only one-half pound of hyposulphite per ton is used.

The maintaining of the stock-solution at such a strength with so small a consumption of hyposulphite, is due to the following causes: First, so little base metal occurs in the first wash-water from the Yedras ore, that all the metals are precipitated from it, not only at very slight expense in sodium sulphide, but also with the production of sulphides just as high in grade as those from the regular leaching-solution. Consequently, the amount of silver entering the wash-water is immaterial, and the first weak silver-bearing solution which comes from the tank after the first wash-water, can be allowed to pass into the said wash-water.

Secondly, in determining the point at which the dilute silver-bearing solution shall be turned from the wash-water precipitating-tanks, into those for the regular leaching-solution, Mr. Johnson makes use of the iodine test (the test being made by the man who attends to the precipitating), the weak silver-bearing solution being allowed to run into the wash-water precipitating-tanks until it shows a strength in hyposulphite equal to one-half that of the regular stock-solution. The actual amount of silver extracted per ton of ore by the first wash-water is only 1.2 ounces or 2 per cent.; but the actual amount obtained in the wash-water precipitate, owing to the use of the above method, is 4.5 ounces per ton.

Of course, in using the above method, the first wash-water is not allowed to drain from the charge, but, as it disappears below the surface of the charge, is immediately followed by solution. The use of the above method has reduced the consumption of hyposulphite per ton from 3.6 pounds, formerly used, to about one-half a pound per ton.

The best results on Yedras ore by the extra-solution are obtained by allowing the extra to stand 12 hours in the ore, or by circulating 6 hours and allowing to stand 6 hours.

The extra-solution is made up on the ore, which is charged dry to a depth of 62½ inches, and sinks 8 inches (or 13 per cent.) during the first washing. The depth of extra required is about 24 inches or about 44 per cent. of the depth of the wet charge. When the tanks are charged full, the extra must therefore be made up in 3 charges of 8 inches each, or else made in a separate tank.

The following statement shows comparison of the mill-extraction by the ordinary process at Yedras for 7 months, June to December inclusive, with that of the Russell process for November and December.

MILL-EXTRACTION AT YEDRAS.

Month (1887).	By Old Process.	By Russell Process.
June	71.1 per cent.	
July	66. " "	Process
August	66. " "	not in use till
September	73.1 " "	November.
October	63.1 " "	
November	63.7 " "	81.2 per cent.
December	67.3 " "	82.2 " "

B.—*Comparison with Amalgamation.*

a. At Lake Valley, New Mexico.—Table XXXIV gives the mill-extraction by the Russell process and by amalgamation on roasted ore at the Lake Valley mill, the difference in favor of the Russell process being 12.4 per cent. of the value of the ore. The leaching-charges are 18 to 20 tons, while the amalgamation-charges are 1.5 to 2 tons. The amount of water for amalgamation was about eight times that used in the leaching. The difference in expenses could not be ascertained, as the amalgamation was at the rate of only 3 or 4 tons per day. Probably the difference in expenses would not have been less than $5 per ton in favor of the Russell process. In this table is illustrated a fact which will be again noted, namely, the percentage obtained by amalgamation in the mill agrees very closely with the percentage obtained by the ordinary solution in the assay-office, but bears no relation to the percentage extracted by the extra in the assay-office, while the percentage obtained in the mill by the extra-solution agrees with the latter very closely.

b. At the Ontario in 1883 *and* 1884.—Table XLI illustrates the comparison in mill-results between the Russell process and amalgamation on Ontario roasted ore during Mr. Russell's experiments at that mill. The amalgamation-tests were necessarily confined to ore crushed through a 30-mesh screen and roasted with 16 to 18 per cent. of salt. But the leaching-tests were made on ore crushed as coarse as a 16-mesh and with all percentages of salt, from 18 down to 0. The table shows that the results of leaching on ore roasted without any salt were only 4 per cent. less than the results by amalgamation on ore roasted with 18 per cent. of salt. Also, that on ore crushed through a 20-mesh screen and roasted with 12.5 per cent. of salt, the extraction by leaching was 4.5 per cent. greater than by amalgamation with finer crushing and 3.5 per cent. more salt. The average results by leaching on ore crushed through a 16-mesh screen with 12 per cent. of salt, were also higher than those by amalgamation on ore crushed through a 30-mesh screen and roasted with 17.5 per cent. of salt. The amount of water required for amalgamation was 2.3 times as great as that used in leaching. The milling expenses by amalgamation for the year during which these experiments were made averaged about $15 per ton. On the other hand,

TABLE XLI.

THE RUSSELL PROCESS COMPARED WITH AMALGAMATION. RESULTS ON THE LARGE SCALE AT THE ONTARIO MILL, PARK CITY, UTAH, 1883 AND 1884.

Process Used.	Per cent. of Salt.	Size of Screen.	Weight of Charges. Tons.	Per cent. Extracted.
Russell Process	0.	30	2	84.9
Amalgamation	18.2	30	1¼	89.4
Russell Process	0.	30	2	85.7
Amalgamation	18.2	30	1¼	89.3
Russell Process	9.	30	2	96.2
Amalgamation	15.9	30	1¼	92.5
Russell Process	12.5	20	2	97.0
Amalgamation	15.9	30	1¼	92.5
Russell Process	8.	16	2	92.0
Amalgamation	17.9	30	1¼	96.1
Russell Process	12.	16	2	97.1
Amalgamation	17.3	30	1¼	92.5
Russell Process	16.	16	2	95.7
Amalgamation	17.5	30	1¼	95.5
Russell Process	18.	16	2	95.1
Amalgamation	17.5	30	1¼	93.3

as shown by the expenses for a similar ore in the United States, deduced from the Cusi expenses, the corresponding cost in a mill using the Russell process would probably be at least about $5 per ton less.

c. At the Marsaac Mill (*Daly Ore*), *Park City, Utah.*—Table XLII gives all the mill-tests so far made in the series of experiments now being conducted by Mr. W. A. Wilson, Superintendent of the Marsaac mill, to test the applicability of the Russell process to the ore treated in that mill. In all cases the ore was roasted for amalgamation—not for leaching. If the ore had been roasted with reference to the Russell process, the results by that process would probably have been 2 to 5 per cent. higher. The weight of ore used in the leaching-charges was 2 tons, and in those for amalgamation 1.5 tons each. The amount of water used in amalgamation is 7 to 8 times the amount used in leaching. The comparison in expenses has not been made; but the Marsaac mill-expenses are about $11, and the difference will probably be $4 to $6 (see table) in favor of the Russell

TABLE XLII.

COMPARISON OF THE RUSSELL PROCESS WITH AMALGAMATION AT THE MARSAAC MILL, PARK CITY, UTAH, IN 1887, ON "DALY" ORE ROASTED FOR AMALGAMATION.

No. of Mill-Run.	Total Time Covered by Mill-Runs.	Value of Roasted Ore. Oz. Silver per Ton.	Size of Screen Used.	Per cent. of Salt Used.	Per cent. Extracted by Ordinary in Assay-Office.	Per cent. Extracted by Extra in Assay-Office.	Per cent. Extracted by Amalgamation in Mill.	Per cent. Extracted by Russell Process in Mill.
1	October 20th to November 20th, 1887. These tests are now being continued.	33.22	20	10	93.4	94.4	90.0	88.9
2		35.43	"	10	87.2	91.0	90.7	88.5
3		43.03	"	10	89.9	91.2	89.9	87.5
4		40.04	"	10	88.8	90.8	89.0	86.6
5		38.87	"	10	89.4	91.6	90.7	88.2
6		39.34	"	10	86.9	88.9	90.6	86.3
7		40.87	"	10	88.9	90.9	90.5	90.8
8		42.32	"	10	87.1	91.0	93.5	90.8
9		35.94	"	9	88.5	90.7	92.3	90.9
10		40.13	"	10	88.0	90.5	89.8	91.8
11		39.67	"	12	89.4	91.6	89.9	91.0
Averages.					88.9	91.1	90.6	89.2

process, the ore being a simple alkaline ore and requiring only a small amount of chemicals. Experiments are now being made with less salt and coarser screens for the leaching-tests, a 20-mesh screen and 10 per cent. of salt being the least which can be used for ore to be treated by amalgamation.

d. At the Ontario in 1887–88.—Table XLIII gives all the mill-tests so far made in the series of experiments now being made similar to those in the Marsaac mill on Daly ore. These tests also are being conducted by Mr. Wilson to determine the applicability of the Russell process to the Ontario ore. In all the tests, the crushing, the percentage of salt used, and the roasting are fixed with reference to obtaining the best possible results by amalgamation, and not at all with reference to the leaching. Nevertheless, the mill-extraction by the Russell process averages 9.4 per cent. of the value of the ore above the results of amalgamation. The weight of each leaching-charge was 2 tons and that of each amalgamation-charge about 1.5 tons. The Ontario ore has always required about three times as

TABLE XLIII.

COMPARISON OF THE RUSSELL PROCESS WITH AMALGAMATION AT THE ONTARIO MILL, PARK CITY, UTAH, IN 1887, ON ONTARIO ORE ROASTED FOR AMALGAMATION.

No. of Mill-Run.	Total Time Covered by Mill-Runs.	Value of Roasted Ore. Oz. Silver per Ton.	Size of Screen Used.	Per cent. of Salt Used.	Per cent. Extracted by Ordinary in Assay-Office.	Per cent. Extracted by Extra in Assay-Office.	Per cent. Extracted by Amalgamation in Mill.	Per cent. Extracted by Russell Process in Mill.
1	November 20th, 1887, to January 1, 1888. These tests are now being continued.	55.78	26	11	91.6	91.8	73.6	91.8
2		40.28	"	11	83.8	91.9	71.3	92.1
3		36.49	"	13	91.9	94.1	75.5	86.3
4		43.72	"	14	93.1	94.2	86.3	89.5
5		37.06	"	13	93.8	94.4	89.6	94.9
6		39.08	"	12	87.1	91.8	81.1	90.8
7		43.50	"	12	89.5	92.2	89.6	91.3
8		51.40	"	14	83.9	89.9	86.8	89.2
9		43.10	"	15	79.0	91.4	76.6	90.6
10		38.10	"	16	94.0	95.9	81.8	95.1
11		52.90	"	15	93.8	94.2	91.3	94.8
Averages..............		43.76			89.2	92.9	82.1	91.5

much water for leaching as any other roasted ore, but still the amount of water used in amalgamation for Ontario ore is 2.7 times that required for leaching the same ore. On all other roasted ores, the amount of water required for amalgamation averages sixteen times that used in leaching. The Ontario mill expenses are about $13 per ton. Those for leaching would be about $5 to $6 less, or a difference in favor of the Russell process of $5 to $6 in expenses, which together with the additional extraction of 4.11 ounces makes a total net difference of $9 to $10 per ton.

For January, as in December, the experimental plant at the Ontario mill was run by Mr. Wilson on Ontario ore alone, it not being considered advisable to run the plant on both Ontario and Daly ore at the same time. The following statement gives the average results for January, 1888, by the Russell process and by amalgamation:

Extracted by:	Per cent.
Ordinary in assay-office,	92.1
Extra-solution in assay-office,	93.8
Russell process in mill,	93.0
Amalgamation in mill,	83.1

The following statement shows the comparison between the extraction by the ordinary solution in the assay-office, and that by amalgamation in the mill, for the last ten days in January.

Ordinary solution in assay-office,	92.0
Amalgamation in mill,	78.1

The best results by the Russell process on Ontario ore were obtained while using a stock-solution of about 1.5 per cent. hyposulphite, at a temperature of 90° to 120° F., and with an extra-solution of about the same temperature, and using about 6½ pounds of bluestone per ton. The weight of charge treated was 2 tons. The depth of the dry charge before leaching was 24 inches and after leaching 18 inches. For 2 tons of Daly ore, the depth is 22 inches and after leaching 16 inches. The temperatures of the solutions are the same as for Ontario ore; but the amount of blue-stone used is only 4½ pounds per ton. Although both Ontario and Daly ore yield an alkaline first wash-water, the extra-solution can be circulated (and usually is), as if they yielded an acid first wash-water. The amount of gold extracted by the Russell process from both Ontario and Daly ores is the same as by amalgamation. The amount of gold in the ore is, however, quite small, amounting to only about 0.1 ounce per ton for shipping-ore and probably about one-half that amount for milling-ore.

e. At Cusihuiriachic, Chihuahua, Mexico.—Table XLIV gives the comparison in mill-results, both in apparent and actual extraction, between the Russell process and amalgamation. The ore for each test was weighed with the utmost care. In this case also, the crushing, use of salt, and the roasting were conducted solely with reference to obtaining the best results by amalgamation. The results showed, in the case of amalgamation, a discrepancy between the apparent and the actual extraction, the actual extractions being 6.7 per cent. less than the apparent. The results of leaching, on the other hand, showed a *plus* discrepancy, the actual extraction averaging 2.2 per cent. more than the apparent. The comparison in actual extraction in silver is in favor of the Russell process by 11.4 per cent. of the value of the ore or 5.7 ounces per ton. The difference in expenses between the Russell process and amalgamation is less for Cusi than for most other ores, being about $2.50 per ton in favor of the leaching, making a total net difference of over $8 per ton in favor of the Russell process.

TABLE XLIV.

COMPARISON OF THE RUSSELL PROCESS WITH AMALGAMATION AT THE CUSI MILL. SPECIAL MILL-RUNS ON SAN ANTONIO AND SAN MIGUEL ORES.

Name of Ore Treated.	Mesh of Screen Used.	Per cent. of Salt.	Value of Ore. Oz. Silver per Ton.	Per cent. of Salts Soluble in Water.	Per cent. by Ordinary in Assay-Office.	Per cent. by Extra in Assay-Office.	Number of Tons Treated.	Apparent Extraction in Mill. Per cent.	Actual Extraction in Mill. Per cent.	By which Process.
San Antonio and San Bartolo	26	10.	46.96	16.5	85.0	89.2	50	88.5	91.5	Russell Process.
							50	86.3	79.6	Amalgamation.
San Miguel......	26	8.	53.08	9.5	89.0	91.9	50	90.4	91.8	Russell Process.
							50	87.5	80.8	Amalgamation.

C.—*Comparison Between the Russell Process and Combined Amalgamation and Concentration.*

All the foregoing comparisons have been based on mill-results. On the ore referred to in Table XLV, the data for which were furnished by Mr. G. J. Rockwell, no tests have yet been made on the large scale by the extra-solution, hence the comparison is between mill-results by amalgamation and concentration, on the one hand, and assay-office results by the Russell process on the other. The previous comparisons have been on roasted ore, but this is on raw ore. As no difficulty has been experienced elsewhere in obtaining as good results (within 2 to 3 per cent.) in the mill as in the assay-office on raw ore by the extra-solution, there is no reason to expect any in this case. Table XLV shows in Run I a difference in favor of the Russell process of 53.5 per cent. of the value of the ore as compared with the Boss process, and of 34.8 per cent. of the value of the ore as compared with the combined Boss process and concentration. In Run II no leaching-tests by extra-solution were made. In Run III, the difference is in favor of the leaching as compared with the Boss process by 39.6 per cent. of the value of the ore, and 38.3 per cent. as compared with the combined Boss process and concentration.

TABLE XLV.

COMPARISON OF THE RUSSELL PROCESS WITH AMALGAMATION AND WITH COMBINED AMALGAMATION AND CONCENTRATION, AT THE BIG SANDY MILL, ARIZONA.

Number of Run.	Description of Run.	Value Oz. Silver. per Ton.	Per cent. Extraction by Concentration.	Per cent. Extraction by Amalgamation.	Total Mill-Extraction.	Per cent. Extraction by Ordinary in Assay-Office.	Per cent. Extraction by Extra in Assay-Office.
I.	4 Days Run without Chemicals . 3 Days Run with Chemicals....... Amalgamation by Boss Process *before* Concentration.	21.8 21.5	28.7 8.7	25.3 45.3	54.0 pr ct. 54.0 "	49.0 50.5	85.1 92.5
II.	4 Days Run .... Concentration *alone*.	20.8	43.0		43.0 "	No leaching tests made.	
III.	6 Days Run with Chemicals....... Amalgamation by Boss Process *before* Concentration. Coarse Ore Concentrated, Fine Ore Amalgamated.	30.2	63.7	51.9	53.2 "	50.4	91.5
IV.	3 Days Run with Chemicals....... Amalgamation by Boss Process *after* Concentration.	25.0	44.8	7.6	52.4 "		87.8
V.	4 Days Run with Chemicals ...... Amalgamation by *Charge* Process after Concentration.	20.7	34.4	28.8	63.2 "	No leaching tests made.	

Run IV shows in favor of the Russell process 43 per cent. when compared with concentration, or 35.4 per cent. compared with concentration and Boss process combined.

The average difference in assay-office results by the ordinary and extra-solutions is in favor of the extra-solution by 38 per cent. of the value of the ore. Here also, as has been noted in the case of roasted ore, the amalgamation-results agree closely with the results by the ordinary solution.

D.—*Relation of the Mill-Results Obtained by Amalgamation, and Those by the Russell Process, to the Results Obtained in the Assay-Office by the Ordinary and Extra-Solutions.*

In making the comparisons between the mill-results by amalgamation and by the Russell process, the relations of the former to the ordinary and of the latter to the extra in the assay-office, have been noted. Table XLVI gives the results of mill-runs on five different kinds of ore by amalgamation and by the Russell process, and the corresponding assay-office results by the ordinary and extra. The extractions from San Antonio and San Miguel are actual clean-ups in the mill, the others being apparent extractions in the mill.

TABLE XLVI.

RELATION OF MILL-RESULTS BY AMALGAMATION AND RUSSELL PROCESS TO THE ASSAY-OFFICE RESULTS BY THE ORDINARY AND EXTRA-SOLUTIONS.

Name of Ore.	Per cent. Extracted by Ordinary in Assay-Office.	Per cent. Extracted by Extra in Ass'y-Office.	Per cent. Extracted by Amalgamation in Mill.	Per cent. Extracted by Russell Process in Mill.
Lake Valley	71.0	81.5	70.6	83.0
San Antonio and San Bartolo	85.0	89.2	79.6	91.5
San Miguel	89.0	91.9	80.8	91.8
Daly	88.9	91.1	90.6	89.2
Ontario	89.2	92.9	82.1	91.5
Average	84.6	89.3	80.7	89.4

The mill-results by amalgamation average 80.7 per cent., which is 3.9 per cent. below the ordinary in the assay-office. On the other hand, the mill-results by the Russell process average 89.4 per cent. or 0.1 per cent. above the extra-solution in the assay-office, and 8.7

per cent. above amalgamation in the mill. These results indicate that amalgamation-results on roasted ore are almost as dependent on a good chloridizing roast as is the ordinary leaching process. But the Russell process, on the contrary, may be entirely independent of it.

VIII. SUMMARY, BEING A GENERAL COMPARISON OF THE RUSSELL PROCESS WITH THE ORDINARY LEACHING PROCESS, AND WITH AMALGAMATION.

A.—*With the Ordinary Leaching Process.*

1. The Russell process permits a coarser crushing of the ore.

2. It requires a less careful chloridizing roasting. Hence less salt, and sometimes no salt, need be used.

3. The roasting may be accomplished in a suitable furnace almost instantly; but the roasting for the ordinary process requires time.

4. The results obtainable by the ordinary process are dependent upon a good chloridizing roast, while the Russell process is sometimes entirely independent of the chloridization, and always exceeds in extraction, the amount actually chlorodized.

5. The extra-solution allows a much more systematic leaching and gives more regular results.

6. The results by the extra-solution are not affected by the presence of caustic alkali in the ore or solution, while even minute quantities affect very powerfully the results by the ordinary process.

7. The Russell process is much more applicable to ores containing lime or lead.

8. It produces sulphides free from lead, and the lead as a by-product.

9. From raw ores it extracts often two to ten times as much as can be extracted by the ordinary solution.

10. Hence, ores which would require chloridizing for treatment by the ordinary process can often be treated raw by the Russell process, thus allowing the erection of a cheaper plant and saving the additional expense and losses incident to chloridizing.

B.—*With Amalgamation.*

1. The cost of plant for the Russell process is much less than that for amalgamation.

2. It permits a much coarser crushing of the ore, the size required

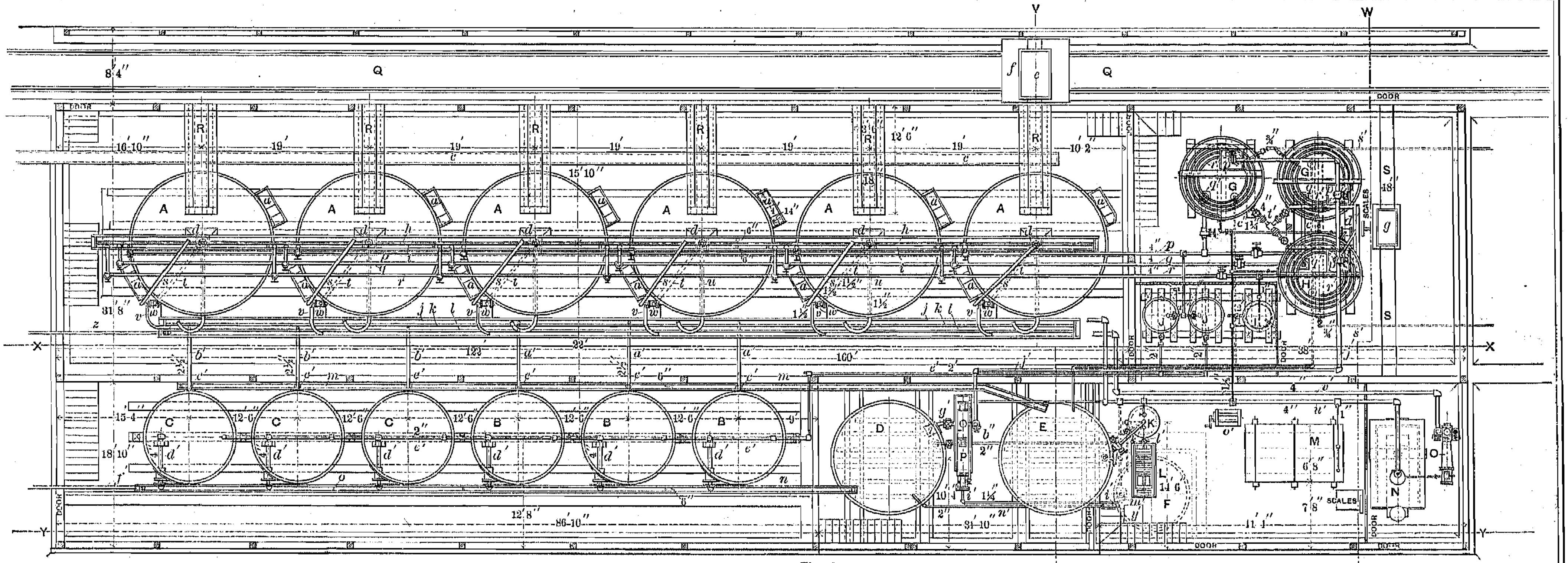


Fig. 4. PLAN.

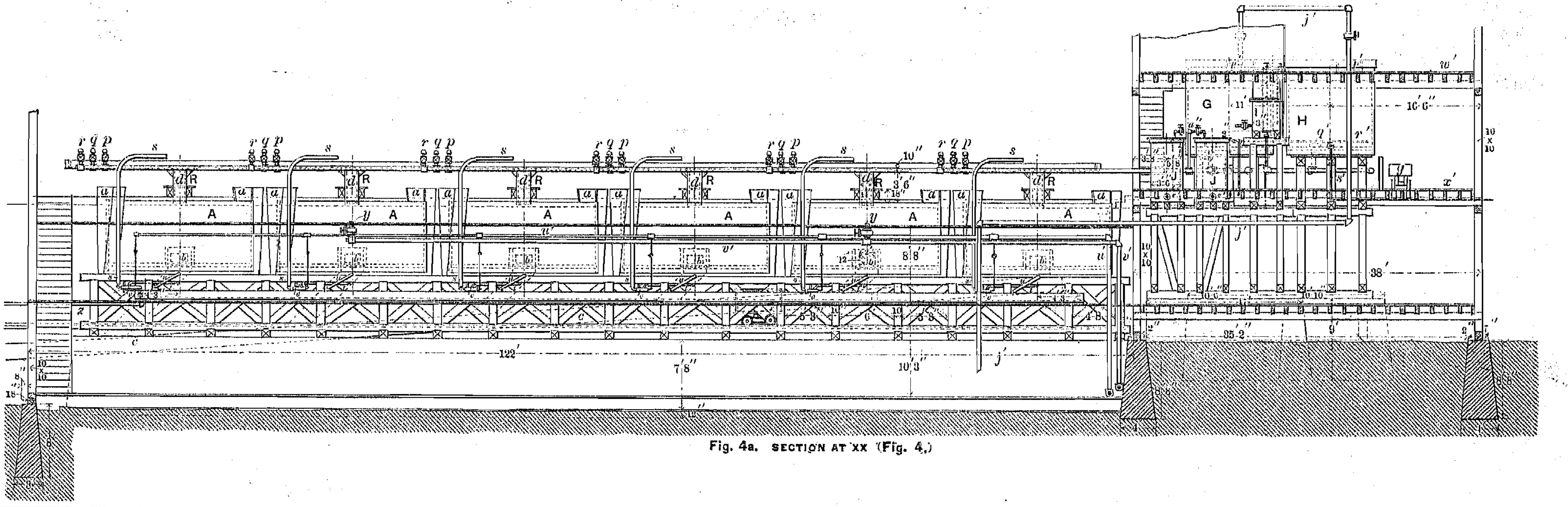


Fig. 4a. SECTION AT XX (Fig. 4.)

WORKS FOR THE RUSSELL PROCESS.

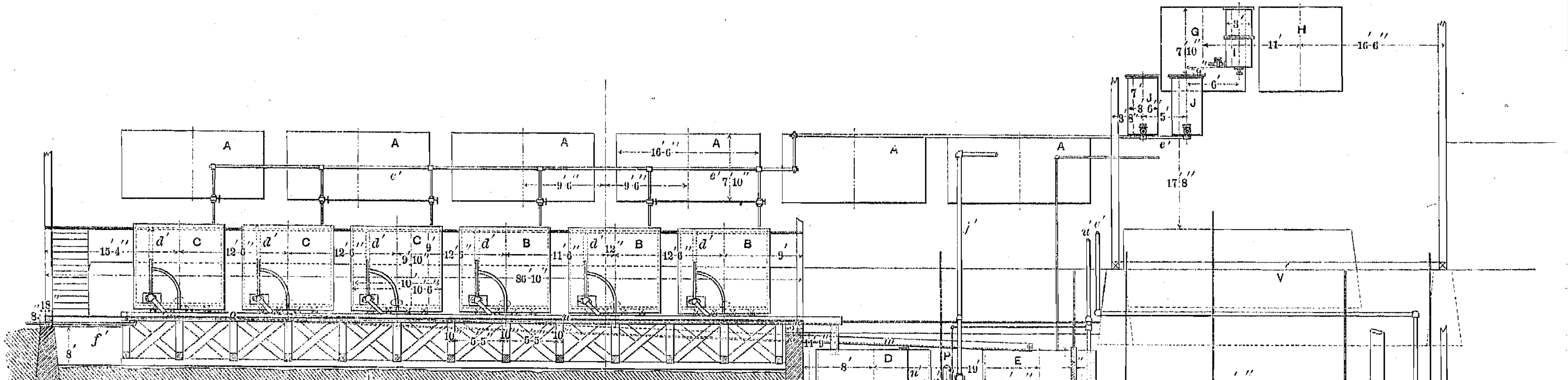


Fig. 5. SECTION AT YY (Fig. 4)

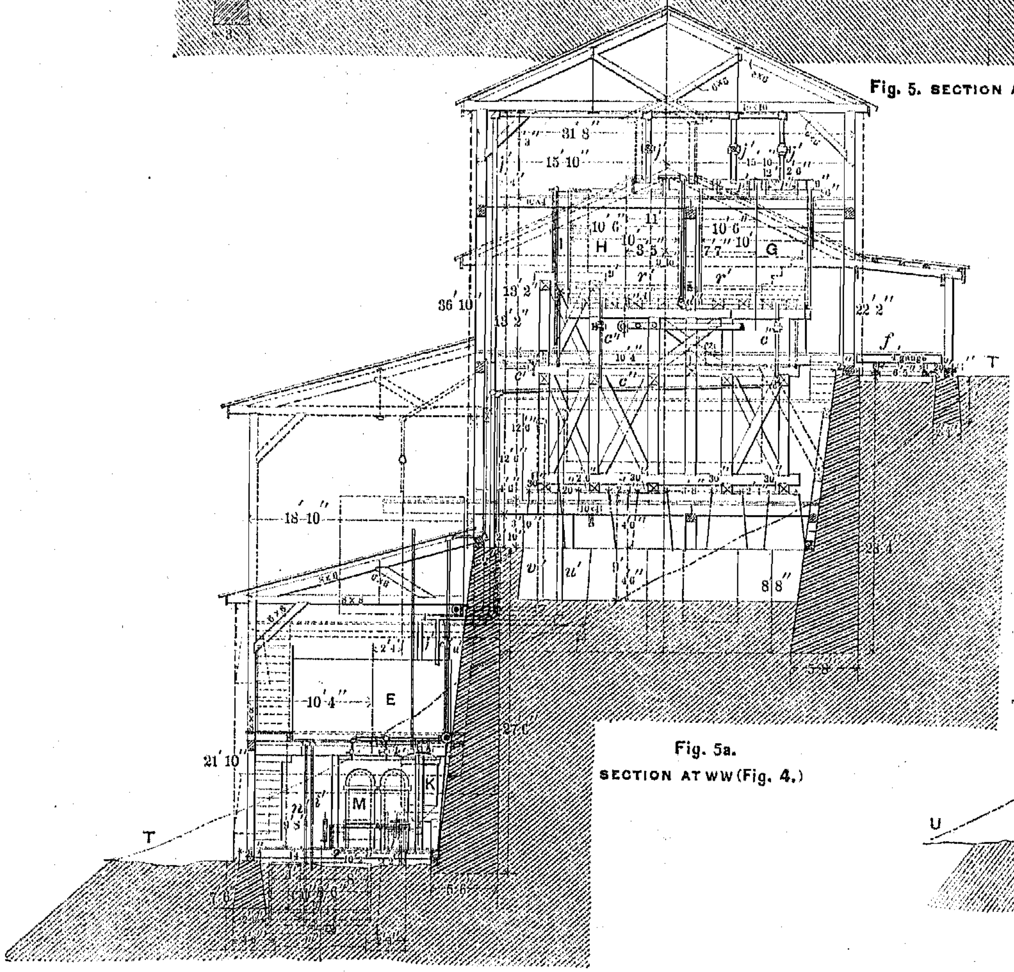


Fig. 5a.

SECTION AT WW (Fig. 4.)

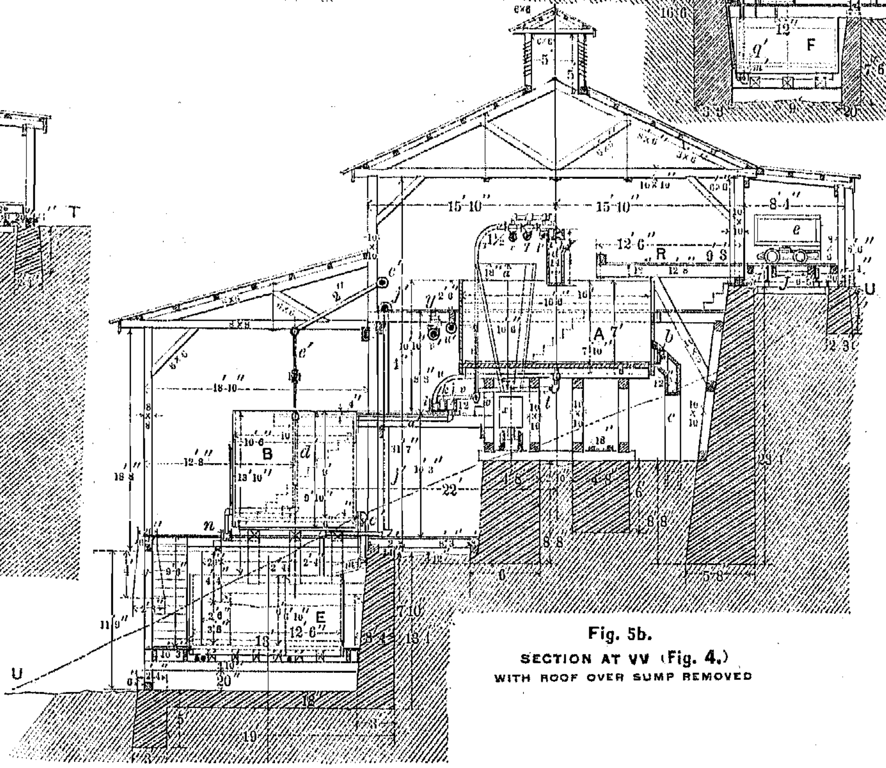


Fig. 5b.

SECTION AT VV (Fig. 4.)

WITH ROOF OVER SUMP REMOVED

WORKS FOR THE RUSSELL PROCESS

for amalgamation being rarely less than a 26-mesh and usually a 30-mesh screen, while leaching, on the other hand, requires only an 8- to 16-mesh screen.

3. The results of amalgamation are closely dependent upon a good chloridizing roast, while the Russell process is often independent of it.

4. Hence a less careful roasting is required, and therefore less, and sometimes no, salt.

5. The roasting for amalgamation requires time, while that for the extra-solution can be accomplished almost instantly.

6. The extraction in the mill by the Russell process is almost always higher, and the expenses are always less, than by amalgamation.

7. Raw ores that can be successfully treated by amalgamation yield a higher percentage by the extra-solution (except Silver Reef) and always at less expense.

8. The Russell process permits the extraction, from roasted ore, of copper and lead as valuable by-products.

9. It is not injurious to the laborers' health, as is apt to be the case with amalgamation.

10. It extracts compounds of silver which are not attacked by amalgamation.

11. It requires no machinery in the leaching, and no moving parts, except a single pump.

12. The charges treated are from ten to thirty times the size of those treated by amalgamation.

13. The amount of water required for roasted ores averages only one-sixth, and for raw ores only one-thirtieth of that required for amalgamation.

14. In large mills, the quantity of quicksilver in circulation represents a capital of $30,000 to $40,000, which is more than 150 times the cost of the stock-solution for the Russell process.