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A THESIS

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PHYSICAL AND EXPERIMENTAL CHEMISTRY APPLIED TO METALLURGICAL PROBLEMS

IN CANDIDACY FOR THE DEGREE OF DOCTOR
OF SCIENCE IN PHYSICAL SCIENCE
BY
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Along the lines of the Hydro-metallurgical extraction, chiefly of commercial metals from mineral matter and the like, in all probabilities the unexpected has happened; in reviewing metallurgical practice and advancement up to the close of the year 1912, especially in that of copper, smelting operations held first position, as same has, possibly, since such operations first became a matter of commercial enterprise.

To the uninitiated, possibly a slight apology was proper for giving Hydro-metallurgical advancements the place of honor over the advancements recorded in the time honored dry process, in reviewing and recording the status and achievements in the recovery of the metals of this particular class up to and including the end of the year 1913; to the Hydro-metallurgist possibly the slight excuse was read without being noticed, the emphatic position given his class was expected as a reward for long and careful experimental study, thought, hard work, expenditure of money, and many disappointments.

The prominence given to the exponents of the wet methods of metallic extraction is well deserved, the immense amount of experimental work done in the past two years and the gratifying results obtained especially during the past year, together with the immensity of the projects contemplated by this particular branch of metallic extraction, bears ample testimony to the righteousness of this deserved notice.

It can be well said that this new Hydro-metallurgical development is the first chapter in the new metallurgy of copper and which in all probabilities will prove to be the greatest copper metallurgy period that the world has ever seen.

Some idea as to the reason why wet extraction particularly as to the winning of copper, has so suddenly come to the front and occupied the attention of so many experimenters, when attention is drawn to the fact that immense bodies of low grade silicious copper ore carrying from .6% to 2½% in copper and from almost nothing in gold and silver in some cases to 1 oz. in silver and possibly 25 cents in gold in others; have been developed and proven. The Braden Mine in Chile, South America has developed 100,000,000 tons of low grade oxidized ore. The Utah Copper Co. has, at Bingham Canyon, Utah, U.S. in the neighborhood of 40,000,000 tons of silicious oxidized ore capping averaging .6% copper and from 10 to 25 cents per ton in gold and silver; other mines notably in Arizona and Nevada, U.S. also have opened up large tonnages of similar ore.

The old term "rebellious ores" as applied to smelting ores in the past in certain cases, is practically eliminated from the present miner's vocabulary, however, in attempting to apply flotation or concentration methods to various oxidized copper ores, especially such as oxides, carbonates and silicates, the above term "rebellious" may possibly be still applied.

1st. In the case of the attempted application of the Flotation Process to oxidized ores, it was found that the tension between the oxidized ore particles and those of the gangue was such that a separation by this method of the ore particles from the gangue, was not satisfactory.

2nd. Concentration. Under this heading two divisions can be made.

A. Dry Concentration.

B. Wet Concentration.

A. Under the division of dry concentration no great progress or large installation can be recorded, though some little success can be recorded to its application in dry placer gold mining such as, at the Dull Placer Camp at the mouth of Copper Canyon, eighteen miles South West of Battle Mountain, Nevada, and in other localities in New Mexico and Arizona.

B. The wet method has been the chief factor in the application of concentration to the recovery of metallic values from various ores, notably the immense plant of the Utah Copper Co. at

Magna and Arthur near Salt Lake City, Utah, where at times as much as 24,000 tons of low grade silicious ore are subjected to wet concentration methods, per twenty-four hours. The metallic contents of the ore treated averaging

Copper	1.25%
Gold and Silver	.20 cents per ton.

The product in the form of concentrates averages 17.31% copper. The recovery of copper from the raw ore corresponds to

an average of	63.78%	at the Magna Plant
and	64.18%	Arthur "
Average recovery of	63.98%	
Gold and Silver recovery	50.00%	in round numbers.

Total number of tons treated during year, Magna Plant,	4,142,700.
Arthur "	3,376,692.
Total	7,519,392.

With an average for the raw ore of 1.25% cu.

This gives a total copper content of	187,984,800 lbs.
Less the amount of copper finally recovered =	119,939,809 lbs.
Loss of copper in tailing, etc.	68,044,991 lbs.

At the Nevada Consolidated Mining Co., Nevada, 3,419,178 tons of copper ore was treated

Assaying 1.597% = total copper contents of	100,556,223 lbs.
Less the amount of copper finally recovered	64,972,829 lbs.
Loss of copper in tailings, etc. =	35,583,394 lbs.

At the Miami Copper Co., Arizona, 1,058,784 tons of copper ore was treated

Assaying 2.3% = total copper contents of	48,704,064 lbs.
Less the amount of copper finally recovered =	32,867,666 lbs.
Loss of copper in tailings, etc. =	15,836,398 lbs.

At the Ray Consolidated Copper Co., Arizona, 2,366,007 tons of copper ore was treated

Assaying 1.72% = total copper content of	81,390,641 lbs.

Less the amount of copper finally recovered	54,158,309 lbs.
Loss of copper in tailings, etc. =	27,232,332 lbs.

Losses of Copper in Wet Concentration and Smelting.

In summing up four of the largest producers, we find at the Utah Copper Co. a loss in tailing and in smelting the concentrates,

etc. of copper of	68,044,991 lbs.
At the Nevada Consolidated a loss of	35,583,394 lbs.
" " Miami Copper Co.	15,836,394 lbs.
" " Ray Con. Co.	27,232,332 lbs.
A grand total yearly loss of copper of	146,697,111 lbs.

As milling operations have been going on at these various plants for several years in some cases, the loss when so figured up is appalling.

It is well known that the best possible mill methods are closely followed, consistent with financial conditions, nevertheless, the best possible saving under these conditions is from 63.78% to 71.06% of the copper values of the raw ore, in the form of concentrates, and that these concentrates have to be smelted for the production of the metallic copper, in which operation, smelting losses must be allowed for, so that here is to be found a problem wherein chemists and metallurgists can find ample material for mental reflection and the profitable spending of leisure moments.

In looking around for a process that might be available to recover the losses of metallic copper in the aforementioned examplifications, also to avoid future ones, oil flotation, owing to its successful application to zinc ores notably at Butte City, Montana, has attracted the attention of copper producers and is at present being experimented with quite extensively, notably at the Braden Mine in Chile, South America, where a plant treating 600 tons per day has been installed, additions to same are under construction whereby the total treating capacity of the enlarged plant will be increased to 3,500 per day.

By way of exemplifying the advance made in saving the metallic copper values of the raw ore by installing the oil flotation process, the recoveries made during one month by oil flotation = 81.6% while that in the wet concentration plant = 61 %. Laboratory test by flotation showed a possible recovery of 95% though it is hardly expected to maintain this high percent of recovery in actual working conditions. It is thought possible that a minimum of 85% recovery can be maintained.

The Utah Copper Co. at Bingham Canyon, Utah, and other copper interests at Butte City, Montana, are at present experimenting extensively with the oil flotation process with the view of adapting the process for reclaiming past wastes and to prevent future ones. Modifications of "oil flotation" are being tried out and installed in various parts of the world, with great success as compared to the older wet concentration method, in effectiveness and cheapness.

Having briefly reviewed the losses incurred in wet concentratioh of low grade sulphide copper ores, along with a comparison of what extra savings may be effected by the introduction of oil flotation, the winning of metallic zinc from its ores, principally that of the sulphide, will be considered.

Taking that of the Butte Superior Plant at Butte City, Montana, as affording the best example of efficiency and high recoveries, together with possibly the largest tonnage per diem of any oil flotation zinc plant in America.

Here, a little over 1,000 tons per diem of zinc lead ore is treated. The head ore entering the mill assays from 17% to 21% zinc, and an intermediate lead product is secured by means of wet concentration, which concentrates are shipped direct to the lead smelters, by this preliminary treatment 12% of the crude ore is eliminated, so that of the original but 88% is in reality treated by oil flotation.

The "heads" entering the "flotation plant" assayed as follows:

Zinc	%	13.91
Lead	"	.77
Copper		.19
Iron		2.73
Manganese		1.44
Silica (insoluble)		68.87
Silver	ozs. per ton	7.73
Gold		.02

Concentrates.

Zinc	%	50.71
Lead		4.43
Copper		.45
Iron		1.39
Manganese		2.72
Silica (insoluble)		10.82
Silver	ozs. per ton	26.55
Gold	" " "	.05

Tailings.

Zinc	%	2.06
Lead		.15
Copper		.04
Iron		.94
Manganese		.85
Silica (insoluble)		91.50
Silver	ozs. per ton	1.62
Gold	" " "	.004

Thus a saving of 90.67% of the total values of the ore was effected at a cost of treatment of $4.225 per ton, ~~comparing this~~ as compared with the average saving of 60 to 65% by the wet concentration system alone. In the Joplin zinc districts in Missouri, we find that "flotation" recovers an extra 25 to 30% zinc as in the case of Butte zinc ores, and that, from a non-profitable to a moderate profit making mining and milling venture, a very profitable one has evolved. Heretofore at Butte City the finding of considerable amounts of zinc ore was almost generally a sign to stop.

In comparing the results in the district we find:

Butte	concentrates	averaging	zinc	%	50.71
Joplin	"	"	"	%	54. to 63.
Butte	tailings		"	%	2.06
Joplin	"			%	2 to 3.
Butte	recoveries			%	90.67.
Joplin	"			%	60 to 65.
Butte	loss in tailings			%	9.23
Joplin	" " "			%	25 to 30.

So that an examination of the figures given will prove quite clearly, that the Butte Superior from a non-profit maker to a moderate one under wet concentration system of metallic value saving as aforementioned, has, by the addition of oil flotation, been turned into a splendid profit maker.

The separation of metallic sulphide ore particles by oil flotation, consists in agitating by air or otherwise a mixture of finely ground ore with water together with the predetermined requisite quantity of oil, due to the affinity between the oil and valuable metallic sulphide ore particles, especially when assisted by the buoyant effect of air, by hydrogen, and possibly other gases, the flotation effect of said metallic particle is effected on the surface of the water and same can be easily skimmed off and recovered; whereas the gangue and non-sulphide particles are not effected and sink to the bottom of the tank in which the separation is being effected, and can be therefrom suitably eliminated.

How simple, efficient and well known scientific knowledge, that could easily and efficiently be applied to the conservation of natural resources and the betterment of conditions in life in general, can be permitted to lay dormant and unused for years, nay even centuries, is well exemplified by the oil flotation process; though the Romans of old were aware of the affinity that existed between metallic sulphide particles and oil; it remained for an American school mistress to rediscover and patent same; even then, the patent rights ran out before advantage was taken of the principle, for the treatment of various ores and the recovery of metallic values therefrom.

Doubtless, metallurgical establishments have effected a small saving at a great loss, in refraining from carrying on the research work in the past, that they should have done, though at present this condition is somewhat changed, nevertheless in the meantime millions have been permitted to go to absolute waste, barring what may be recovered at an expensive rehandling, together with the loss represented by labor and money invested in the partially treated material. Modifications of the oil flotation process are being constantly introduced, the latest being a combination of flotation without oil and with electrolysis of the pulp ground from twenty to forty mesh, liquified with water in proportion of ten tons of ore to which twelve tons of water is added, along with a small amount of acid (for the purpose of increasing the conductivity of the solution) together with some salt. The solution is electrolyzed at a low voltage meanwhile being agitated by a current of compressed air. Under this treatment water is decomposed and a stream of hydrogen is evolved which assists in buoying and bringing to the surface of the solution a black scum of metallic sulphides, which is suitably skimmed off, recovered, and sent to the smelter to be smelted to metallic copper. It is claimed that in crushing the ore, that crushed to forty mesh, contains more sulphur than the ore crushed to twenty mesh, so that in this way, the sulphur contents of the concentrates can be somewhat regulated to suit the purpose of the smelters. Where a small amount of salt is contained in the solution, decomposition is effected by the electrical current, and chlorine gas is liberated which has a solvent effect on the sulphides, but principally on the oxides, carbonates and silicates of copper, the ultimate effect of which is, that the copper is deposited in metallic form on electrodes suitably placed in the solution; the recovery of the metallic copper contents of the ore is said to be high, in the neighborhood of 97%, while the concentrates made, averaged over 18% in copper, it is thought that metallic copper can be produced for 5 cents per pound.

Some advantages of the flotation process are, that it can treat slimes successfully, in fact in most cases the ore is ground to a fineness of 80 mesh, so that the process becomes almost essentially a slime process. Obviously, the reason why a better extraction can be obtained by a combination of the two, can be explained by the fact, that the best of the two processes can be used in conjunction with one another; as much as possible of the coarse and fine rich particles of metallic sulphides, being recovered by wet concentration in the preliminary stages of coarse crushing, as it is found profitable, by determinations made in each particular case, to recover. As an example of this practice, it may be cited that the Butte Superior Mill treats 12% of the crude ore by wet concentration and the balance 88% is treated by flotation.

The balance of the crude ore excepting that which may have been rejected in the preliminary treatment by wet concentration, as worthless, is ground to the proper degree of fineness and subjected to flotation treatment. The amount of oil used, though generally given as about 1%, varies more or less from this stated amount, in accordance with the nature of the ore, the proper amount of oil having been experimentally predetermined.

Hydro-metallurgy.

Hydro-metallurgy may be defined as the profitable extraction of metals by the solvent action of water or water solution of dissolving reagents, followed naturally, by the recovery of the desired metal.

In looking back under the heading of Losses of Copper in Wet Concentration and Smelting we find that at the plants of the Utah Copper Co., Nevada Consolidated, Miami and Ray Con. that the yearly loss of copper in wet concentration followed by the recovery of metallic copper by smelting amounts to 146,697,111 lbs., equal to a loss of 35% of the total amount of copper contained in the original ore, at 15¢ a pound this means in round numbers $22,000,000, and by no means represents the total amount involved, as there are several other plants in different parts of the country such as at Anaconda, Montana, the Inspiration Plant in Arizona, and several others of lesser importance, further, the yearly loss would have to be multiplied by the number of years that the various plants have been in operation respectively, deductions, in case that some recovery could be made by subsequent "flotation treatment" or re-treatment by wet concentration, would have to be made; nevertheless the re-handling and the capital tied up in the unextracted metal in the large accumulations of tailings, in some cases extending over a number of years back, amounts to an immense sum of money. At the present writing it is not even very certain that "oil flotation" will successfully re-treat tailing in every case nor even raw ore when first handled in every case, as it is just announced at present writing that "flotation" experiments at the Burro Mountain Copper Co. at Tyrone, have been unsatisfactory but that before discarding the process, another modification will be experimented with, so that the success of oil flotation in the case of copper ores is not so pronounced as with that of zinc ores.

With the above losses in mind, other expenses connected with the refining and selling of the copper must also be considered.

In the beneficiating of zinc ores, that step involved in wet concentration entails a loss of 30-35% of the original zinc contents of the raw ore, in reducing the concentrates by smelting to metallic zinc a further loss of 10-15% must be sustained, so that as noted, an average 50% of the original zinc values of the ore is lost; "flotation" has decreased the wet concentration losses by increasing the recovery from 60-65% to 90% in some cases, still, the smelting losses remain, along with the heavy freight charges incurred in hauling the concentrates to the zinc smelter; for example, the concentrates of the Butte Superior Mill in Butte City, Montana are hauled to Oklahoma for treatment; a distance of several hundred miles, likewise various zinc ores from Utah and Nevada, are hauled a distance of 700 to 1,000 miles to Colorado, to be smelted.

For the purpose of more clearly setting forth the losses that the zinc miners and millmen have to suffer in addition to the milling losses above noted, may be mentioned the deductions in the form of freight and smelting charges, handling, smelter deductions and losses, selling charges and other forms of arbitrary deductions which are not specifically mentioned in the above but nevertheless included; we may take for our example the concentrates of one of the large Montana Producers as it is quite proper to assume that a large producer whose output exceed 6,000 tons of concentrates a month, will get the best possible rates, the ore being not out of the ordinary for smelting purposes. Here again may be found another item of loss, in the case of the small producer who because he is a small producer finds himself obliged to either accept a heavier discount than the large producer, or, to stop production. The concentrates above mentioned assayed as follows:

Zinc	50%
Lead	2.4
Copper	.5
Iron	1.3
Manganese	2.7
Insoluble	11.
Silver	26.3 ozs. per ton.
Gold	.05 " " "

At the current price at that time, the metallic contents of the concentrates were worth in the neighborhood of $80.00 gross per ton, however, the settlement price received F.O.B. cars at the point of delivery was $23.80 per ton, a further loss of $56.20, or in round numbers of 70% of the value of the concentrates.

In summarizing, we are enabled to tabulate the following losses:

Where "flotation" is used		10.%
Transportation and smelter deductions		63.%
Final Recovery of the merchantable mineral in the crude ore by the miner and millman		27.%
	Total,	100.%

In ordinary wet concentration of crude ore 30-35% say an average of		32.5%
Transportation and smelter deductions		47.25%
Final Recovery by miner		20.25%
	Total,	100.%

In averaging the recoveries of both processes we find that the zinc operator recovers but 23.63% of the original values of the ore, obviously, when studying over the losses in both the copper and zinc industry, it is seen at once, that there is great room for methods that will reduce this loss, whether due to unavoidable natural causes such as may be encountered in concentration and in smelting the concentrates to metal, or to artificial ones, such as exorbitant freight charges, manipulation, arbitrary charges and deductions on the part of the smelters, and other well concealed ways, whereby, under one pretext or another, in the one case the middleman succeeds in taking 63% of the gross value of the ore, and in the other 47.25%. The losses may be divided into two divisions:

1. Those occurring in concentration of the crude ore.
2. Those occurring in the reduction of the concentrates to metal.

Diagramatically expressing these losses we have under

A. By wet concentration - loss 30-35%.

1. Concentration.	B. By oil flotation - loss	10%.
2. Reduction to metal.	C. Freight and hauling	47.25% to 63.%
	D. Smelting losses and charges and handling of the metal	
	E. Manipulation and arbitrary deductions	

Analysing subdivision A under division 1, we find that the loss of 30 to 35% of the value of the crude ore in wet concentration does not give promise of being materially bettered in the near future, apparently in this particular, the maximum of efficiency has just about been reached, in as much as no new line of thought or application of mechanical contrivances has been opened up, that would indicate marked improvement over the present practice whereby the heavy losses would be materially cut down.

Under subdivision B of division 1, oil flotation when used in connection with preliminary wet concentration of the coarser particles or raw ore, has in all possibilities, effected, where it can be used, the much needed improvement in concentration, and the saving of 90% of the contents of the raw ore as now exemplified by the performance of the Butte and Superior Mill at Butte City, Montana, bears ample testimony of the high efficiency and valuable improvement given to the zinc operators, by the use of this process wherever it can be applied to the ores.

Reduction to Metal.

In subdivision C, freight and hauling such as described from points in Montana to Oklahoma or from Utah and Nevada to Colorado, don't admit of much chance in effecting a saving, at least no great one, nor one that lies within the power of the operator to overcome.

Under subdivision D, Smelting Losses, charges and handling of the metal are so blended in with that of the following subdivision E, namely, manipulation and arbitrary deductions, that it is practically useless to attempt an analysis. The facts remain, that under the heading of <u>Reduction to Metal</u>, that the zinc producer loses from %47.25 to %63. of his ore values, and, that in as much as the difficulty is practically beyond his power to overcome, and apparently purposely shrouded in a maze of technical arguments that serve well the purpose of giving no satisfactory explanation for this unwarranted loss nor a tangible problem even, that the producer might undertake to solve. Having reviewed in a general way the losses both in copper and zinc production, together with a more specific discussion of those large losses of metal that are apparently unavoidable, and the other heavy losses that occur under the heading of <u>Reduction to Metal</u>, and for which no good reason seems to exist together with the fact that the point of attacking the problem, is beyond reach of the aforenamed producers; it can easily be understood why, any process that has any merits along the lines above indicated, is readily given much serious consideration.

Hydrometallurgy.

Hydrometallurgy with its peculiar fascination, due, in a great measure to the ease with which laboratory tests and experiments can be made, whereby processes and their necessary modification can be, with reasonable despatch evolved, that apparently seem to assure immediate success on the completion of a plant when built in accordance with the results obtained in the laboratory tests.

Bearing in mind the heavy losses in wet concentration that as yet have not been completely overcome, together with the other apparently useless and heavy deductions in converting into cash, the products of the concentrating mills, as explained above, a further excellent reason can be seen for the great interest, as evidenced by the immense amount of experimental work and money that is at present being expended, in the development of this branch of metallic extraction. The advantages can be enumerated as follows:

1. <u>Completeness of Process</u>.

It is quite possible to extract the ore and produce the marketable metal on the spot, which in all probabilities is the great goal which all metal producers are striving to attain. The beauty, completeness and efficiency of the cyanide process, acting as a model, like which, chemists and metallurgists are striving to find some application of wet extraction that will successfully treat zinc and copper ores.

2. Elasticity of Products.

Though the metal in such a state of purity as to be able to be sold directly to the consumer without any further refining, is without doubt, the ultimate object of the present day hydrometallurgist, the possibilities of this method i$ such, that various products can be produced in lieu of the metal, should it be desirable for any reason that such be done.

3. Freedom of Operations.

Where producers can extract and produce the metal on the grounds or at least when they can do it themselves, we may look to the industry developing to the fullest possible extent, with the benefits that are to be derived from an industry that is thriving strictly on its own merits, whereby every person that is entitled to reward by virtue of their connection with the existence of said industry, receives the full amount of such reward as they are entitled to, as against the unjust and detrimental preferential distribution of the fruits of such labor, due to artificial conditions that have been brought about, by which, the middleman manage to absorb from 47.25% to 63% of the ore values. The exact meaning can be very simply illustrated by citing the case of the gold miner, who after paying all the natural and necessary expenses in connection with producing his gold, can sell same to the mints for what it is worth, less expressage and possibly a very small coinage charge, certainly a strange contrast to the condition that now exists in the zinc and copper metal markets.

4. Development of the Particular Industry.

As renewed activity has been noticed in a great many zinc properties, especially in the Western Section of the United States and even up in British Columbia, due principally to the introduction of "oil flotation" whereby zinc losses in concentration have been reduced from an average of 32.5% to 10%, thereby enabling the mines and mills to re-enter the list of active producers, so will a reduction of the loss of from 47.25% to 63% to the middleman, stimulate a still greater activity in opening up and operation of mines and mills that cannot be profitably operated, due to such excessive deductions on the part of the middleman.

The disadvantages of Hydrometallurgical Processes can be summed up as follows:

1. Mechanical Difficulties.

The handling of large amount of acid solutions, together with the necessary large amount of wash water and in some cases the evolution of noxious gases, are difficulties that are often overlooked in the laboratory. Pollution of streams by foul solution being permitted to escape is a source of annoyance that eventually has to be remedied.

Though these troubles beset a certain plant here in Utah, operating along the lines of leaching, and ~~all though~~ although the objectionable features were quite easily eliminated, nevertheless the little thought or consideration given to these features of the mechanical operations of hydrometallurgy dan be judged from the fact, that the plant in question only caused the nuisance to be abated when several lawsuits had been filed in the district court, and other suits were in preparation.

The handling of comparatively large amounts of dissolved metals and salts is no small or insignificant problem.

2. Chemical Difficulties.

In the cyanide process the metals sought are precious, being worth in the case of silver about $60 per oz. and gold $20 per oz., and the total output at the end of the month being expressed in terms of so many hundred or thousand ozs. of gold or silver bullion, in giving this statement some little consideration, it will be readily seen that gold and silver alone are extracted almost to the complete exclusion of the other metals, not so with the general leaching processes for the recovery of baser metals, such as copper, zinc, etc.; for example the output of The Morro Velho Mine yielded for the year 1913, 97,208 ozs. of bullion worth $2,057,540, which resulted from the treatment of 174,000 tons of ore, whereas the output of the Butte Duluth Copper Leaching Plant near Butte City, Montana, was represented during one month by the production of 27½ tons fine copper and 20 tons of high grade cement copper from the treatment of 110 tons per day, the cement copper produced in various places assays from 50 to 92% pure;

so that, even the large amount of impurities that have to be reckoned with in the refining of this cement copper, besides those taken up by the solution from which the copper is recovered by electrolysis, are an index of the different dissolving powers of the leaching mediums employed in hydrometallurgy; with this fact in mind it may be readily seen how the solvent power of the leaching medium used, may be similarly almost as great in the case of dissolving the sought for metal as in that of undesirable ones.

4. Consideration of Solvents.

The choice of a solvent must receive careful attention, secondary reactions that in the laboratory may not be thought to be a menace, may on a commercial scale prove to be one of the main causes that will lead eventually to the discarding of the process. This may well be exemplified by the Sulphite Process for leaching oxidized or sufficiently roasted ~~[illegible]~~ sulphide ores.

In the laboratory tests made before the process was installed on a working scale, apparently little or no great importance was attached to the formation of the red cuproso cupric sulphite precipitate; in a weak sulphur dioxide solution, especially with the application of heat this precipitate readily forms, and when once formed it is practically impossible to dissolve it. Of the several reasons given for the failure of the sulphite process, when tried on a working scale in Northern Arizona, to the formation of this precipitate may be attributed the chief cause of the failure of the process and finally the abandonment of the enterprise. Other difficulties may be encountered in using leaching mediums in which calcium or other alkaline earth metals are the base, in the presence of iron a precipitate is apt to form that may be a source of much annoyance, so that in both cases either a solvent should be chosen that fails to produce a secondary reaction or the solvent should be so modified as to avoid this danger.

5. Natural Resources.

Hydrometallurgical operations are such, that in their application to metallic extraction, notice must necessarily be taken of the natural surrounding in the center of which it is proposed to carry on such operations, as the natural resources in one section of the country that would tend were tried to make a wet extraction process successful at that point, those same resources found at some other place, would require a modification of the process to be employed in order to preclude the total failure of the same general process. For example in a smelting center where smelter fumes are in abundance and a nuisance, the manufacture of sulphuric acid would involve a one step higher degree of oxidization of the original sulphur contained in the ore, at an enormous expenditure of new capital, and also heavy operating charges in addition to the capital outlay in building the plant, for the production of the acid, whereas utilizing the sulphurous fumes as they are evolved from the smelter stack, with such expenditure as might be necessary for the required degree of purification of such fumes, would be, the proper method of utilizing natural or artificial resources as found in such a district. As sulphur deposits are to be found in various parts of the West, sulphur dioxide gas could be readily produced by simply burning the sulphur and so manipulating the chemistry of the process as to recover, with nominal losses, the sulphur dioxide gas for re-use in extracting more metals from other ore. This possibly might be found to be better practice than evolving SO_2 from useless pyrites, and would be absolutely necessary in some parts of the country. The necessary gas could even be evolved from pyrites of almost any grade if the same pyrites contained sufficient values, to make their use a financial object.

Deposits of high grade lime rock to be used for neutralization and precipitation should be considered, likewise a ready prepared leaching medium afforded by salt water or ~~mashers~~ marshes, such as are to be found in various parts of the West notably the Great Salt Lake in Utah. Special mention of the advantages to be derived from the use of such waters will be seen later under copper leaching.

Metallurgy.

Accepting the broad definition that Metallurgy is the Art of Extracting Metals from their Ores, it may be stated that possibly the most commonly and universally accepted understanding of the term is that it means:

The Application of all necessary chemical and mechanical means whereby metals and their compounds are prepared on a commercial basis, from their ores. In the commercial pursuit of mining and metallurgical operations, an ore means a mineral which will at least pay the cost of extraction and reduction, but most generally, it is understood that it must be "Pay Ore" and that means, a "profit" besides the cost of mining and other expense involved in the final production and marketing of the metallic content sought.

It might be well to point out, that the definition of Metallurgy given above is well adapted, and is the best definition that it is possible to give the art up to and including the early part of the nineteenth century; the overthrow of the phlogistan theory, and the birth of modern chemistry together with its establishment firmly on its correct principles, by the Master Mind Lavoissier, may in all candor be said to be the close of the period during which Metallurgical Operations may strictly be regarded as an art. It must be admitted nevertheless, that it took several years before the new light in which Lavoissier made chemistry appear, was to be thoroughly understood and accepted, so that the actual applidation of this modern chemistry, to metallurgy, likewise, took additional time, and we will find that such applications date back into the early part of the nineteenth century.

From this time, dates the present time commonly accepted definition of Metallurgy in its restricted sense, wherein chemistry now evolved and placed on its correct basis as a science, is applied to and assists the Art of Metallurgy.

From this time on we find chemistry entering more and more into the domain of Metallurgy, until now the New Metallurgy of Copper presently called Hydrometallurgy, which is slowly dawning, as evidenced by the greatest activity the world over, in wet methods of metallic extraction, will usher in the next greatest Metallurgical Period that the world has ever seen, wherein the profitable extraction of immense low grade ore deposits, the utilization of the acidity of Smelter Fumes without previous oxidization, together with other local natural resources, problems in fact that have heretofore baffled the skill of chemist and metallurgist the world over and have come to be regarded as unsolvable, will elevate, the present day Metallurgy to the ranking title of "Metallogy" the Science of Metals.

Metallurgy may be divided into three parts:

1. General Metallurgy.
    - A. Historical.
    - B. Physical Properties.
    - C. Chemical "
    - D. Processes.
    - E. Mechanical Appliances.
    - F. Fuel.
    - G. Refractory Material.

2. Ferrous Metallurgy.

Wherein iron and steel alone are considered.

3. Non-Ferrous Metallurgy.

Embraces that of all the metals and metalloids both common and rare excepting iron and steel.

Metallurgical Operations are carried on by

1st. Dry method.

2nd. Wet "

The dry way contemplates chiefly various smelting operations such as:

A. Reverberatory or English Process (neutral).

B. Blast Furnace or Swedish " (reducing).

C. Pyritic or American " (oxidizing).

D. Converter Smelting (Experimental as yet) (reducing).

E. Pyro-electric " " " (neutral).

The earliest authentic knowledge that we have concerning metallurgical operations dates back to the first century A. D. and in all possibilities the credit for such knowledge is due to dry methods.

Wet Method.

The first knowledge of some form of process under this heading appears to have been known to Vitruvius, B. C. 13, who was aware of the fact that gold was taken up or amalgamated by Mercury; and gold was thus extracted in the 11th Century and silver in the 16th Century. Natural cementation was practiced in the 14th Century. The birth of the real practical wet methods of metallic extraction, took place in 1835-1836, when Becquerel in France, evolved a process for the treatment of complex ores containing copper, silver and lead, whereby the chloride and sulphate produced were soluble in a solution of salt, and used various metallic couples such as, lead, iron copper and zinc, to recover the silver, copper and lead.

Becquerel published later, plans of a one ton plant for the treatment of ore. An account is also given of the successful treatment of 20 tons of ore in Paris; Becquerel was 36 years in advance of his time, the cheap means of producing electricity, the dynamo, had not been invented, so that the recovery of the metals from their solution in salt, by means of electricity was not practically used.

In the Becquerel Process, we have all the elements of the Augustin Process for the leaching of silver ores, which was introduced in 1840 at Mansfeld, Prussia, to supersede amalgamation for the extraction of silver from matte. In the Augustin Process the silver is recovered by means of copper, no use of electricity being attempted.

The Ziervogel Process was introduced in 1841 at Hettstadt and displaced the Augustin Process at Mansfeld in 1844.

In the year 1858 the Von Patera Process was introduced, and in 1860 the Kiss Process.

Various modifications and improvements have been proposed and introduced, notably in California in 1874 by Küstel; in Mexico, the Patera Process was introduced as early as 1868. At the present time in the United States there are no lixiviating plants in operation, using Patera Process, the plant at Cortez, Nevada, using a modification known as the Clark Process, having shut down some few years ago; the tailings at present are being treated by the cyanide process.

Lixiviation is defined as "the abstraction by water of the soluble alkaline or saline matter present in earthly admixtures; for example that of quicklime and potash to make potash lye, from that of effloresced alum schist to make aluminous liquors".

To Lixiviate means to "impregnate with lixivium or lye, alkaline". Lixivium means "water impregnated with alkaline salts extracted by lixiviation of wood ashes". Leaching means "to be subject to percolating water, to take away from, out, by percolation; to lixiviate." Apparently, lixiviation has been applied to such processes as the Patera, Kiss, Augustin, Ziervogel, Roszner, Patera, Russel, and other modifications; pioneer hydrometallurgists, such as Küstel and Hofman, have used the terms in such a way that lixiviation practically means the equivalent of leaching. Küstel says "To lixiviate, or to leach a soluble metal out of ore — that is, to filter a liquid through the ore, so that it dissolves and carries out the metal in a clear solution — is the process called lixiviation, or leaching process". Nevertheless, the use of the term "lixiviation" as above mentioned in reference to the Patera, Kiss, and other processes, is to be noted, together with the further fact that the cyanide process and the extraction of copper ores, are spoken of as leaching processes; the rejuvenated Augustin Process at Park City, Utah, is spoken of as "Chloridizing Leaching".

The fact cannot be denied, that at the present time very few amalgamating or lixiviating plants are in operation; chlorination, proposed by Percy as a result of some experiments performed by him in 1846, and introduced into California by Deetken in 1857, made some little headway both in the United States, Canada, and other parts of the world.

The Bromine Method of extraction gave promising results for a time at the Nellie Bly Mill in Colorado, U.S., nevertheless all these processes seemed to have their limits, whether due to excessive costs or mechanical difficulties, the facts remain, that they did not give to metallurgists at large the latitude of metallic extraction that was apparently needed, and the more intense application of Chemical Science to Metallurgical Operations in the form of the cyanide method of gold and silver extraction, patented by John Rae and others in the United States, and put into practical operation by the Mac Arthur Bros. of Scotland, may well be stated as the commencement of that intense period of chemical application to the art of metallurgy, desired, both on account of the apparent exhaustion of valuable workable deposits by the known wet methods, and the discoveries of immense deposits of low grade ores that could not be worked by such processes.

So that, soon, we find these new efforts bearing fruit, and lo!

like the hand of magic the new process not only displaces practically all other known wet methods, but even is applied to the extraction of values from bodies of ore, thought to be impossible of metallurgical treatment owing to their small valuable tenor. Well may this indicate a new metallurgical period, and well may it be termed Metallogy, the Science of Metals.

The Wet Method of Metallic Extraction is divided as follows:

A. Amalgamation.
  1. Mortar and Plate Process.
  2. Barrel Process.
  3. Washoe Process.
  4. Arrastra "
  5. Patio "
  6. Hydro-electro-amalgamation.

1. Mortar amalgamation in hewed stones was the primary way in which gold was first won in the earliest known machines, and with the additions of plates and other improvements used in connection with concentration and cyaniding, is treating the bulk of all ores that are amalgamated today.

2. Barrel amalgamation was devised in Europe in 1790 and used there for many years; at present it is nowhere in use.

3. The Washoe or American Process, excepting in possibly one or two places in foreign countries, is not in use any more; the rich silver deposits having been apparently mostly found and exhausted, so that at present the bulk of the silver is obtained in connection with the recovery of other metals, chiefly lead.

4. The Arrastra process, chiefly for extracting gold, is still used in Nevada and other places where small mines are thus worked, by prospectors and miners.

5. The Patio Process, in Mexico and South America, is being used to some extent, principally for silver ores.

6. Hydro-electro-amalgamation processes have not thus far come into practical and constant use. At DeLamar, Idaho and in Colorado experiments with same were discontinued.

B. Lixiviation and Leaching.

The annexed table gives the divisions and subdivisions of this class.

The most important processes only will be discussed, either on account of their present use or seemingly importance, and also because of their past importance and possibility of their re-use — particularly in the case of the Augustin Process which is the oldest and father socalled of lixiviating methods, and which has been recently revived and again put to use on low grade ores, that, in the early days of the Ontario Mine at Park City, Utah, were considered too low grade in value to be treated at the mill by amalgamation, and were used as stope filling in the mine.

The following classification has been adopted because it is thought to be the most scientifically correct one, and even so, combinations of various processes are being used to meet the requirements exacted for the separation of various metals, that are beginning to play an important role in the metallurgy of the second period or "conservation of natural resources", that is shown dawning on our Natural Existence; to the effect, that it is well nigh impossible in some cases to strictly place a certain process under a certain heading, in which case, when possible, the first part of the name of the prodess that tends to chemically classify it will be used as an index.

Hydro-electro processes have been given a place on the list, both on account of their redent great strides in metallic beneficiation and to the fact, that, though they are combinations mostly of leaching methods, and recovery of metal by electricity, common consent, among mining men and metallurgists has designated them by such names as "Electro" — "Recovery by Electricity" — "Wet Electric", and "Hydro-Electric", so that to avoid confusion, and to simplify the classification as much as possible, and not overlooking the fact, that the possibilities of the modifications of this class will most assuredly win for themselves a most deserved place on the list of processes; has induced the writer to depart from the strict chemical classification, wherein such a process would be classified as a sulphuric process wherein the metal was recovered in case of copper, by cementation or by electricity.

-:DIAGRAMMATICAL ILLUSTRATION OF ORE TREATMENT:-

-:NON-FERRONS METALLURGY:-

- DRY
  - 1.Smelting
  - 2.Refining
  - 3.Dry Deposition
- AMALGAMATION.
  - 1. Mortar and Plate Process
  - 2. Barrel "
  - 3. Washoe "
  - 4. Arrastre "
  - 5. Patio "
  - 6. Hydro-Electro Amalgamation Process.
- WATER SOLVENTS
  - 1.Ore In Place Method
  - 2.Dump and Tailings Method
- ACID SOLVENTS.
  - Strong Acid
    - H_2SO_4
      - 1. Using the Acid as the Solvent
      - 2. Using the Sulphates as the Solvent
      - 3. Using a Combination of the Acid and Sulphates as the Solvent
      - 4. Using a Combination of the Acid and other Solvents.
    - HCl
      - 1. Using the Acid as the Solvent
      - 2. Using the Chloride
      - 3. Using a Combination of the Acid and Chloride.
      - 4. Using a Combination of the Acid and other Solvents.
    - HNO_3
      - 1. Using the Acid as the Solvent
      - 2. Using the Nitrates as the Solvent
      - 3.
      - 4.
  - Weak Acid
    - 1-SO_2
      - 1. Using the Gas SO_2 and Water
      - 2. Using the Sulphites
      - 3.
      - 4. A Combination of the Acid and other Solvents.

- WET
  - LIXIVIATION or LEACHING
    - ALKALI SOLVENTS
      - Strong Alkalies
        - 1. Using KCN
        - 2. Using KCN with other Solvents.
      - Weak Alkalies
        - 1. Ammonia Cyanide Process.
        - 2. Ammonia Carbonate Process
    - ELEMENTARY HALOIDAL SOLVENTS
      - 1. Gaseous Chlorine
        - 1. Percy or Plattner Process
        - 2. Barrel "
        - 3. Vat "
      - 2. Liquid Bromine
    - HYDRO-ELECTRO PROCESSES
      - 1. H_2SO_4
      - 2. HCl
      - 3. HNO_3
      - 4. $HC_2H_3O_2$
      - 5. Alkali
      - 6. Haloidal
  - 2 Purification of Solvents.
  - 3 Precipitation of Metals.
  - 4 Recovery of Metals.
  - 5 Refining
  - 6 Electro Deposition.

Items 2–4: Specially Considered under each Process, together with other Considerations pertinent in each case.

Examplification of Processes

WATER LEACHING. LEACHING ORE IN PLACE.

Installation.	Place.	Time.	Metal.	Tonnage.	At Present.	Recovery of Metal.
Medler-Famous Process.	Chase Creek Canyon near Clifton, Arizona, U.S.	1906-1908.	cu.	1 carload of 52.04 % ~~oz~~. cu. produced.	Abandoned.	By iron, etc. and coke iron couple.
LEACHING DUMPS, TAILINGS, ETC.						
Red Metal Plant.			cu.	200,000 tons dump to be leached.	Working.	Scrap iron and tin.
montana ~~Moreland~~ Ore Purchasing Co's dumps. Butte & Boston Plant.	Butte City, Montana, U.S. Meaderville, Montana, U.S.		cu.	90 tons cu. per month.	"	" " " "

~~LEACHING~~.

1. $H2SO_4$. Leaching USING THE ACID AS THE SOLVENT.

shevsky

Installation.	Place.	Time.	Metal.	Tonnage.	At Present.	Recovery of Metal.
The Arizona Copper Co.	Clifton, Arizona, U.S.	1893.	cu.	315		By iron.
Gumeshevsky Mine.	Russia.	1727.			Closed 1871.	By iron.
Southern Tyrol.	Spain.					as $Cu.SO_4$.
Kedabeg.	Russia.					By iron.
Laist Process.	Anaconda, Montana.		cu.ag.	80	Experimental.	By H2S.
Majdanpec Process.	Majdanpec, Servia.		cu.	Working scale.		
Vitriolization "	Freiburg, Saxony.		cu.			" iron.
" "	Oker-Hartz.		cu.	5 tons per day.		As $cu.SO_4$.
" "	Chicago Refinery, Chicago.		cu.	working scale.		" " "
" Hoffman "	Conc.Kansas City Smelting Co.		cu.	"		
	Argentine, Kansas, U.S.				Abandoned.	" " "
Stadtberge "	Stadtberge.		cu.		Abandoned.	
Vitriolization-Pfort Process.	Lautenthal Works, Lautenthal.		zinc.	working scale.		$Zn SO_4$
Vitriolization Process.	Altenau.		cu.ag.etc.	"		$Cu.SO_4$.
Vitriolization "	Hemixem, Antwerp.		"	"		"
	German Gold & Silver Parting Works.		"	"		"
	Frankfurt-on-the-Maine.		" au.			"

2. $H2SO_4$. LEACHING. USING THE SULPHATE AS THE SOLVENT.

Millberg Process.						Basic sulphate.
Elliott "						By iron.
Cementation.	Rio Tinto, Spain	11th Century.	cu.		working steady.	" "

Continued

Installation.	Place.	Time.	Metal.	Tonnage.	At Present.	Recovery of Metal.
Tharsis.	Tharsis.		cu.		working steady.	By iron.
Van Arsdale Process.			cu.			" SO_2 gas, as metallic cu.
Agordo-Zoppi "	Agordo, Italy.		cu.	working scale.		By iron.
Kedabeg "	Kedabeg, Russia.		cu.	"		" "
Fodal "	Fodal, Norway.		cu.	"	abandoned.	
Wickslow "	Wickslow, Ireland.		cu.	"	"	
Balan "	Seibenbürgen, Transylvania.		cu.	"		
Ziervogel matte "	Gottesbelohnungshütte, Hettstadt.		ag.	"		ag. by cu. cu. by Fe.
" "	Argo, Denver, Colorado.		ag.	"		
" "	Hafod Works, Swansea, Wales.		ag.	"		
3. $H2SO_4$. LEACHING. USING A COMBINATION OF THE ACID AND SULPHATES.						
Cananea Conc. Mining Co.	Cananea, Mexico.	1907.	cu.			By iron.
Nitre Cake Process.		1900.	cu. zn.		experimental.	cu. by Fe. zn. as oxide.
4. $H2SO_4$. LEACHING. USING A COMBINATION OF THE ACID AND OTHER SOLVENTS.						
Snowstorm Mine Process.	Larson, Idaho, U.S.		cu. au.	250		as, cu. by iron.
Hunt and Douglass Modification.	Argentine, Kansas, U.S.		cu.	working scale.		as oxide or metal.
Schemnitz Process.		1897.	Tellurium gold.			Te. pp. by SO_2 or Zn.
Chenhall "		1913.	Tin, Fe. As. W.			Sn. by H2S, cu. by Fe. Fe as oxide. W. as Tungstic acid. As. as oxide.

1. HCl LEACHING. USING THE ACID AS THE SOLVENT.

Installation.	Place.	Time.	Metal.	Tonnage.	At Present.	Recovery of Metal.
Chiquicamata Process.	Chiquicamata, Chile, S.A.	1906.	cu.			As black oxide.
Stadtberge Process.	Stadtberge.				Abandoned a/c sulphides.	
Twiste Process.	Waldeck.		cu.	Practical scale.	Abandoned too much lime.	
Stadtberge "	Westphalia.		cu.	"	Abandoned in favor of H_2SO_4 Process. Account of sulphides.	
2. HCl LEACHING. USING THE CHLORIDE AS THE SOLVENT.						
Doetsch Process.	Rio Tinto, Spain.		cu.	106.	Abandoned in favor of ferric sulphate process.	
Longmaid-Henderson Process.	Bede Metal Works, Hebburn, England.	1865.	cu.	Practical scale.		By iron.
" "	Oker, Germany.			60 tons.		" "
Longmaid-Henderson-Claudit Process.	England.		ag.au.	Practical scale.		" " and iodides.
Longmaid-Henderson Process.	Helsingborg Copper Works, ~~[illegible]~~. Sweden			"		
" "	Penn. Salt Co. Natrona, Pa. U.S.			200 tons.		" "
Vaygourney Experiments.	Tonopah, Nevada, U.S.	1904.				
John A. Greenwalt Experiments.	Denver, Colorado.	1900.				
Augustin Process.	Mensfeld, Prussia.	1840.	ag.	Working scale.	Abandoned	ag. by cu.
Augustin Bittsanszky Process.	Kapnik, Hungary.			3 tons per 24 hours.		By iron.

Continued

Installation.	Place.	Time.	Metal.	Tonnage.	At Present.	Recovery of Metal.
Augustin-Bittsanszky Process.	Tajowa, Hungary.			Working scale.		By iron.
" "	Kosaka, Japan.			"		" "
Augustin-Holt-Dern Process.	Park City, Utah.	1914.	ag.au. cu.Pb.	100 tons.		" " and copper.
Augustin-Christensen Process.	Silver City, Utah.	1914.	"	100 tons per day.		" "
3. HCl LEACHING. USING A COMBINATION OF THE ACID AND THE CHLORIDE.						
Ferrous Chloride Process.	Deva in Liebenburgen.			Working Scale.		By iron.
" "	Rochlitz in the Riesengebiege.				Abandoned.	
Stadtberge "	Westphalia.			Practical scale.	In use.	" "
4. HCl LEACHING. USING A COMBINATION OF THE ACID AND OTHER SOLVENTS.						
Doetsch Process Modification.	Naya, Rio Tinto.		cu.	106.	Abandoned.	
Froelich Process.			cu.			By iron.
Hunt & Douglass Process.	Ore Knob, North Carolina.		cu.	Produced 400,000 lbs. cu.	"	By iron or by CaO as a product and smelted.
" "	Argentine, Kansas, U.S.		cu.	On Working scale, treating cu.matte.	"	Also by electricity. As cuprous oxide and smelted.
" "	Modification by Hoffman.		cu.	"	Abandoned account of making blue vitriol.	
Falum.	Falum, Sweden.			57 tons daily.		Blue vitriol.
Bradley "	Anaconda, Montana, U.S.	1911.	cu.	Experimental.	Abandoned.	as oxide.
Black-Etard Process.	Dunedin, New Zealand.		gold.	Working scale.		
Sulphur Chloride "			gold.	"		by charcoal.

NITRIC ACID LEACHING. USING THE ACID AS THE SOLVENT.

	Installation.	Place.	Time.	Metal.	Tonnage.	At Present.	Recovery of Metal.
1.	Ranklin-Westling Process.		1914.	ag.Bi.As. Fe.cu.co.N. Cd.	Experimental.		As metal or oxides.
2.	USING THE NITRATE AS THE SOLVENT.						
	The Perrino Process.			au.ag. cu.etc.			

SULPHUROUS ACID LEACHING. USING THE GAS SO_2 WITH WATER.

	Installation.	Place.	Time.	Metal.	Tonnage.	At Present.	Recovery of Metal.
1.	Neill Process.	Coconino, Arizona.		cu.	Practical scale.	Abandoned.	as oxide or sulphide.
	Montana Ore Purchasing Co. Neill Process.	Butte City, Montana.		cu.	35 tons experimental run.	"	as a product and by iron.
2.	USING THE SULPHITES AS THE SOLVENT.						
	Hyposulphite of sodium.			copper.			as sulphide.
	Stromeyer Process.			"			by sodium sulphide.
	Patera "	Joachimsthal-Bohemia.	1858.	ag.	working scale.		sodium sulphide.
	Kiss "	Schmollnitz, Hungary.	1860.	"	"		by calcium sulphide.
	Patera Kiss Hoffman Process.	San Francisco del Oro, Mexico.					" " "
	Kiss Process.	Nagybanya.			"		" " "
	Patera.	Melrose, California.	1874.	"	"	Abandoned.	
	"	Bertram Mill, Nevada.	1882.	"	"	"	as a sulphide.
	" clark "	Cortez, Nevada.		"	"	"	" " "
	"	Cerro Gordo, Chile.		"		"	" " "
	"	Cusihuiriachic, Mexico.		"		"	" " "
	"	Yedras Mine, Mexico.		"		"	" " "
	"	Sombrerete, Mexico.		"		"	" " "
	" Russel Process.	Marsoc Mill, Park City, Utah.		Ag.Pb.		"	ag. as sulphide.
	" " "	Holden Mill, Colorado.				"	Pb. as carbonate.
	" "	Sala, Sweden.				"	
	" "	Blue Bird, Butte, Montana.			Experimental.	"	

Sulphurous Acid Leaching - Continued - Using the Sulphites as the Solvent

Installation.	Place.	Time.	Metal.	Tonnage.	At Present.	Recovery of Metal.
Patera Russel Process.	Sombrerete, Mexico.			Experimental.	Abandoned.	
Patera " Process.	Cusihuiriachic, Mexico.			"	"	
Patera " Process.	Broken Hill, N.S.W.			"	"	
Patera Roeszner Process.	Hungary.		ag.au.	working scale.		
4. USING A COMBINATION OF THE ACID AND OTHER SOLVENTS.						
Vadner Process.	Salt Lake City, Utah.	1913.	cu.zn.Fe. ag.au.	Experimental.		cu.as metal by iron or electricity. ag. as metal by iron or electricity. Fe as Fe_2O_3. Zn. as oxide or metal by electricity. au. as residue.

STRONG ALKALI LEACHING. USING K C N AS THE SOLVENT.

Installation.	Place.	Time.	Metals.	Tonnage.	At Present.	Recovery of Metals.
1. Chloride Point.	Chloride Point, Utah, U.S.		ag.	working scale.	abandoned.	by zinc.
Homestake Mining Co.	Dakota, U.S.	1904.	au.ag.	4,000 per day.	in full operation.	" " & amalgamation.
Tonopah Mining Co.	Tonopah, Nevada, U.S.	1913.	au.ag.	500 per day.	"	" " & concentration.
Palmaejo Mine.	Chihuahua, Mexico.	1905.	ag.au.	60 per day.	"	" " " "
Hollinger "	Porcupine, Canada.	1914.	au.		"	" "
McIntyre "	" "	1914.	au.		"	" "
Goldfield Con. Mine.	Goldfield, Nevada.	1913.	au.	1,000 per day.	"	" " (Conc. cyanided).
2. USING K C N WITH OTHER SOLVENTS.						
Cobar Chesney Process.	Cobar Chesney Mine, Cobar, N. S. Wales.			75 tons.		cu. by iron. au. " zinc.
Sulman Teed "	Canada & Australia.			working scale.		"
Dale Process.	Dale, California.			working scale.		" "
Bagdad Mining Co.	Camp Rochester, Arizona.		cu. au.			cu. by zinc dust. au. " " threads.
1. WEAK ALKALI LEACHING. USING AMMONIA AND CYANIDE.						
Mosher-Ludlow-ammonia-cyanide process.			au.ag. copper.			cu. by boiling, as oxide. au. & ag. by zinc dust.
Bertram Hunt.			silver, gold.	Proposed.		by zinc.
2. USING AMMONIA CARBONATE AS THE SOLVENT.						
Ammonia Carbonate.	Lautenthal.		zinc.	working scale	superseded.	as basic carbonate and zinc oxide.
Schnabel Process.	Hoboken, Antwerp.		"	"	"	
Bretherton "	California.		"	experimental.		

0. ELEMENTARY HALOIDAL LEACHING. USING CHLORINE AS THE SOLVENT. PERCY OR PLATTNER PROCESS

	Installation.	Place	Time	Metal	Tonnage	At Present	Recovery of metal
1.	Alaska Treadwell.	Alaska, U. S.	1884	au.			
	Utica Mine.	California U. S.		au.			
	Plymouth Conc. Mining Company	California U. S.		au. pb.	100 tons per month		Pb. as sulphate Au. by $FeSO_4$
	Colvert's method			Au Ag Cu			Cu by Fe. Ag. by Cu. plates Au. by $FeSO_4$
2.	Barrel Process						
	U. S. Smelting & Refining Co.	Colorado Springs, Colo.	1912	gold	400		
	Newbery Vautin Process.	Mt. Morgan, Australia	1887	"	working scale	Replaced by vat process	
	Mears "	Dolora, Canada		"	"		
	Thies "	No. Carolina		" au	"		Au. by $FeSO_4$ Cu by Fe. Au by H_2S
	Golden Reward "	Deadwood, Dakota		"	90 tons per day		
	Doloro "	Canada		"	working scale		Au by H_2S
3.	Vat or Munktell Process						
	Mount Morgan Mine	Mt. Morgan Australia	1898	gold	400 per day		
	Black-Etard Permanganate Process	Bethanga Mill Victorai	1900	"	2000 per day	abandoned	
	ELEMENTARY HALOIDAL LEACHING. USING BROMINE AS THE SOLVENT.						
	Hinnman Cassel Process	Nelly BLY Mine Colorado		gold		abandoned	
	Bromination Mill Process	Rapid City Dakota		"		Abandoned	By H2S

HYDRO-ELECTRO LEACHING. H2 SO_4.METHOD.

Installation.	Place.	Time.	Metals.	Tonnage.	At Present.	Recovery of Metal.
Laszcznski Process.	Medzianka, Poland.		copper & zinc.	225 to 500 lbs. pure cu.per day		Electrically.
Intercolonial Copper Co.	Nova Scotia, Canada.		copper.	1 ton pure cu.per day		"
Siemen's Halske Process.	Southern Tyrol, Spain.		"			Cu.SO_4. Vitriolization.
Tosizza Process.			"			Electricity.
Braden "	Braden, Chile.		"	Proposed.		"
Marchese matte"	Stolberg.		"	Working scale.	Abandoned.	"
" " "	Genoa.		"	"	"	"
Gunther "	Mansfied, Germany.			"		"
Sjostedt-James"			nickel. " cu.	"		"
Hybinette "	Canadian Nickel Co. St.St.Marie, Canada.		" cu.			~~as alloy and~~ electrolysis.
Carmichael "	Dorchester, N.B. Canada.		"	1 ton cu. per day.		Electricity.
Thum "	Balbach Co. Newark, N.J.		"	3 tons nickel.per day.		"
Smith and Gutensohn Process.	Milan, London & France.		tin.	Working scale.	"	"
Siemens Halske Zinc Process.	Berlin, Germany.	1895.	zinc.	Experimental.		"
	Illawarra, N.S.W.	1898.	"	Working scale.	"	"
Cooper Cowles Zinc Process.	Hayle, Cornwall.	1896.	"	Experimental.		"
Salom Lead Process.	Niagara Falls, N.Y.		lead.			"
Letrange Process.	St.Denis, France.		zinc.		"	
	Romily.		"			
Rosings "	Tarnowitz, Silesia.	1886.	"	Working scale.		"
Andre "		1877.	cu.ni.			
Siemens Halske"			tin.	6 tons scraps per week.		"
Gutensohn "			tin.			"
Cito "			Ni.			" anodas alloy

HYDRO-ELECTRO LEACHING. H2SO$_4$ METHOD - continued.

Installation.	Place.	Time.	Metals.	Tonnage.	At Present.	Recovery of Metal.
Bohne's Process.	Robertson & Bense Tostedt, Hamburg		tin.		Abandoned.	Electricity.
Salom "	Electrical Lead Co. Niagara Falls, N.Y., U.S.	1902.	lead.			"
Marchese Process matte.	Ponte San Martino, Piedmont, Casarza, Sestri, Ponente, Genoa.	1882.	copper.	2 tons cu. per day.	"	"
Marchese "	Actien Gesellschaft, von Stolberg und Westphalia, Prussia.			1200 lbs. cu. per day.		"
Butte & Duluth	Butte City, Montana.	1914.	copper.	500 tons per day.		By electricity and iron.
HYDRO-ELECTRO LEACHING. H2SO$_4$ METHOD. USING THE SULPHATES.						
Siemens Halske Process.			copper.			Electricity.
Siemens " "			zinc.	Experimental.		"
Ramen "		1909.	"			"
Farmer "		1888.	nickel.			"
Rickets "		1894.	"			Electricity and precipitation.
Bischoff & Tiemann Process.		1895.	" & co.			Electricity.
Kugel Process.			"			"
E. Wohlwill Experiments.			" & cu.			"
Borcher's "			" & Fe.			" & precipitation.
Lange & Kosman.			zinc.			"
Brandenburg & Weyland.		1901.	tin.			"
Quintaine.		1900.	tin.			"
Placet-Bonnet Process.		1890.	chromium.			"
Dechaud & Gaultier.		1846.	copper.			By iron, copper couple
Cobley Process.		1880.	"	Experimental.		Electricity.
Deligny "		1881.	"			"

HYDRO-ELECTRO LEACHING. H_2SO_4 METHOD. USING THE SULPHATES. - continued.

Installation.	Place.	Time.	Metals.	Tonnage.	At Present.	Recovery of Metal.
Elkington mette Process.		1871.	copper,			Electricity.
Blas & Miest "		1882.	"			"
Barnard Mohr "		1899.	zinc.			"
Nahnsen "	Silesian Co. Lipine.		"	Experi-mental.	Abandoned.	"
Keith "	Arlington, New Jersey, U.S.	1902.	copper.	125 tons.		"
Siemens Halske			"	Experi-mental.	Not in use.	"
" " M. deKay ~~Kay~~.	Thompson Experiments.	1904.			"	"
Ray Mines.	Ray, Arizona. U.S.			"	"	"
Ashcroft Zinc Process.	Grays - Essex.	1895.	zinc.			
	New Castle, N. S.W.	1898.	"	Working scale.	Abandoned.	"
Titus Ulke "	Constable's Hook, New Jersey.	1897.	nickel.			"
Strap "		1893.	"			"
LeVerrier "		1898.	"			"
Classen "		1882.	" & copper.	Experi-ments.		"
Förster's "		1897.	nickel & copper.	"		"
Becquerel "		1835.	Lead & silver.	1 ton plant, experi-mental.		"
Cowper Cowles "			cu.zn. ag.au.			"
Möller & Street"		1898.	chromium,			"

HYDRO-ELECTRO LEACHING. HYDROCHLORIC ACID METHODS. USING THE ACID.

Installation.	Place.	Time.	Metals.	Tonnage.	At Present.	Recovery of Metals.
1. Lambotte-Doucet Process.	Bleyburg Mine.		zinc.			Electricity.
Meyer.	Aix-la-Chappelle.		tin.			"
USING THE CHLORIDE SOLUTION.						
2. Body Process.		1883.	cu. au. ag.			Electricity.
Hoepfner "	Schwartzenberger-Hutte, Saxony.	1891-1892.	copper.	Working scale.	Abandoned.	"
" "	Giessen-an Weidenau,	1888.	copper.	"		
" "	Siegen,				"	"
" "	Brunner Mond & Co. Winnington, England.		zinc.	6 tons per day of zinc and 18 tons of bleach.	Working steadily.	"
" "	Huesshan, Austria.		"		"	"
" "	Papenburg, Germany.		nickel.	3,000 lbs. nickel per day.		
Brown "	Cleveland, Ohio, U.S.	1902.	"	1,000 lbs. nickel daily.	Purchased by trust and shut down.	"
Canadian Copper Co.		1908.	"			
Gunther-Frank-Process.			"	Experimental.		"
Oxford Copper Co.	Bayonne, New Jersey, U.S. America.		~~tin~~. "	Working scale.		"
Brown-Neil Tin Process.			tin.	"		"
Bergsoe "	Copenhagen, Denmark.		tin.	"		"
Vortmann "		1894.	nickel, cotoll.			as oxide and hydroxide.
Sanderson "		1890.	Antimony.			Electricity.
Kopp			"			"
Siemens ~ Siemens Halske "			"	"		~~precipitation and~~ electricity.
Swinbourne-Ashcroft anhydrous chlorination zinc process.			zinc, cu.			" + precipitation

HYDRO-ELECTRO LEACHING. HYDROCHLORIC ACID METHODS. USING THE CHLORIDE - continued.

Installation.	Place.	Time.	Metals.	Tonnage.	At Present.	Recovery of Metals.
Feree's Experiments.		1901.	Chromium.			Electricity.
Glaser's "		1901.	"			"
Moissan Process.			Uranium.			"
Bunsen's Experiments.		1854.	Manganese.			"
Ashcroft Process.	Cockle Creek, N.S.W.	1897.	zinc.	Working scale.	not in use.	"
Siemen's Halske Process.		1895.	zinc.			"
4.USING A COMBINATION OF THE ACID AND OTHER SOLVENTS.						
Greenawalt Process.		1910.	cu.	10 tons.	experimental.	Precipitation and electricity.
F. Getsthorpe "		1910.	tin.	60 tons in all.	abandoned.	"
Garcia "	London,England.	1891.	tin.	working scale.	"	"
Frasch "	Canadian Works, Canada.	1900.	cu.	"		"

HYDRO-ELECTRO LEACHING. NITRIC ACID METHODS.

Installation.	Place.	Time.	Metals.	Tonnage.	At Present.	Recovery of Metals.
1.						
2. USING THE NITRATE SOLUTION.						
Blas and Meist Process.	Louvain, Belgium.	1882.	Lead, etc. and sulphur.	Laboratory experiments.	Not put into use.	Electricity.
USING A COMBINATION OF THE ACID AND NITRATE.						
3. Dietzel.			au. ag. cu. etc.			

HYDRO-ELECTRO LEACHING. ALKALI METHODS.

Installation.	Place.	Time.	Metal.	Tonnage.	At Present.	Recovery of Metal.
Tin Alkali Electric Recovery. Th. Goldschmidt & Co.	Essen, Germany.		Tin.	Treating 13,000 tons scrap tin per annum.		Electricity.
Claus Process.	London, England. Bootle, "	1896 1900.	Tin.	100 tons in all.	Abandoned.	"
Borcher's "			Antimony.			"
LeLande's "	Birmingham, England.	1886.	Zinc.	Working scale.		"
C.A. Burghardt.		1880.	Zinc.			
Tin Alkali Electric Recovery.	Electrochemische Fabrik, Kempen-on-the-Rhine.		Tin.	10,000 tons scrap tin per year.		"
Tin Alkali Electric Recovery.	H.W. von der Linde-Crefeld.		Tin.			"
Ludwig, Process		1880.	Antimony.			"
Luckow, "		"	"			"
Classen, "			"			"
USING A COMBINATION OF SOLVENTS.						
M.E. Hermit Process.	Rouen.		Nickel.	Practical.		"
Vortmann.			Mercury.			"
Engelhart & Nettel.		1896.	"			"
Bratson.		1892.	Tin.			"
Price & Coleman.		1884.	"			"
Cruikshank.		1899.	"			"
Watt's Process.		1887.	Zinc.			"
Basse Selve "		1891.	Nickel.			"
Santa Rosa "	San Sebastian Mine, Santa Rosa, Salvador.		Gold,cu.			"

HYDRO-ELECTRO LEACHING. HALOIDAL METHODS.

Installation.	Place.	Time.	Metal.	Tonnage.	At Present.	Recovery of Metal.
H.R. Cassel Process.	Glasgow, Scotland.	1883- 1885.	gold,etc.	working scale.		Electricity.
Stolp's "		1886.	gold.			"
Luckow		1880.	zinc.			"
Blas & Miest	Broken Hills.		zinc and sulphur.	ton.		"
Baker-Burwell		1907.	zinc.			" and precipitates.

In summing up the present hydro-metallurgical processes in use, we find that the various modifications of the cyanide process have displaced all forms of silver and gold recovery under this heading, with the exception of a possible few chlorinating plants and miners' arrastres in the United States and Canada, and possibly of a very few plants coming under this general heading in other parts of the world, mortar and plate amalgamation being excepted. A recent exception may be noted in the revival of the Augustin Process at Park City, and in the Tintic Mining District, Utah, where a 75 and a 100 ton plant, respectively have been installed; in the case of the former, to treat stope filling from the Ontario Mine, and in the case of the latter, regular ore from the mines of that district.

A second mill is being constructed at Park City at the American Flag Mine. At present writing none of the mills are in operation. At Park City new forms of roasters are to be installed, and in the Tintic District the mechanical features of the plant are to be remodeled.

The Compania Minera de Oruro at Oruro, Bolivia, South America is contemplating installing this process for the treatment of their ores which carry

Tin	3.%
Silver	35 oz.
Copper	.5%

The present method in use in treating the 100 tons per diem which the mine produces, results in a heavy loss of silver through dust and volatilization.

In the recovery of base metals, copper and zinc are the only ones that need to be considered. Only one wet zinc extraction process is in operation at present writing, and that is the Hoepfner Hydro-Electrolytic Process at Brunner Mond & Co. at Winnington in

England and Hruschau, Austria.

It is a notable fact that the oldest wet copper extraction processes still survive, namely at Rio Tinto, Spain, where an immense tonnage is worked; at Natrona, Pennsylvania and in England the Longmaid Henderson Process is still in operation on a scale of 100 or more tons per day.

The newer hydro-electric practice of solution in sulphuric acid and recovery by means of electricity, though not absolutely new, has not in the past been absolutely successful, though heralded as an accomplished fact in various parts of the world, the fact remains that it has fallen in disuse wherever installed, for various reasons or other. An exception to this statement can be made in the case of the Arizona Copper Co. at Clifton, Arizona, where a 300 ton plant has been in successful operation for a few years past. In Montana, two hydro-electric recovery units have been erected, the Bullwhacker Plant of 60 and the Butte Duluth of 110 tons capacity. Sulphuric acid was used as the solvent in both cases.

At the Bullwhacker Plant the expected results were not realized and same was shut down about July 1914. The Butte Duluth plant has been somewhat successful, so that it is proposed to increase the capacity up to a 500 tons or more per day. The ore assays 2.01% Cu. The output in January 1914 being 55,000 lbs of electrolytic and 40,000 of high grade cement copper.

The moral effect of the preliminary successes, especially in Montana, has been of an electrifying nature. A 2,000 ton plant with the necessary sulphuric acid generating apparatus in connection therewith is being erected at Anaconda. A 10,000 ton plant is being contemplated by the Utah Copper Co. to be erected for their Bingham Canyon Mines, near Salt Lake, Utah, while the world's greatest leaching plant with a capacity of 20,000 tons daily is being rapidly completed at Chuquicamata, Chile.

CAUSE OF SUCCESS

If we enquire into the reasons for the success of the successful plants we find that at Rio Tinto, Spain, that it is principally due to taking advantage of natural advantages and resources, and that these conditions have either not been found or developed elsewhere.

The operations of the Hoepfner Process, for a long time thought to be inoperative, are due to the utilization of the chlorine gas in making "bleach" and also to the utilization of a waste product, calcium chloride, of the ammonia-soda process.

In Montana, success will doubtlessly come from utilization of the sulphur in the ore for the production of the necessary sulphuric acid, chaap hydro-electric power at Great Falls and large scale operations.

In Chile the ore has been found to be brochantite, $CuSO_4 . 3Cu(OH)_2$ with a small amount of atacamite $Cu(OH)Cl, Cu(OH)_2$. The presence of sulphates yields, in subsequent operations the necessary acid; the metallic values being in the friable portion of the ore, is broken sufficiently fine for satisfactory extraction, by coarse crushing. At the same time the interfering metals chiefly iron, etc. are left in this coarse portion, and are but slightly acted upon by the solvent.

The absence of gold, silver, arsenic, antimony and bismuth should also be noted.

These advantages, together with an immense plant and over 300,000,000 tons of ore in sight (by 1915) should reasonably insure success. The fact, that utilization of natural resources and advantages has so apparently been the cause of success in the above cited cases, has been the object of thought and experimentation on the part of the writer for a long time, being especially directed toward the utilization of the waste product and nuisance, sulphur dioxide gas.

Smelting centers the world over have felt the influence of the nuisance in the form of injunction and damage suits, to their great expense. Attempts made to utilize these fumes have not resulted in any thing but failure thus far in the United States, where metallic extraction was the object sought to be attained.

The principal attempt was made at Coconino in Northern Arizona where a 100 ton plant was erected to treat oxidized ores found in that district. The venture was a failure, the reason given for such being that:

1st. The attempt to saturate the water by simply blowing the gas through it.

2d. The poor agitation obtained and the consequent length of time required to leach a comparitively small amount of ore.

3d. The dilute solution obtained, 1% being the maximum amount of copper which can be held in solution by an excess of SO_2.

4th. The ease with which copper separates from these solutions, both in the leaching tank, the pressure tank and the filter tank, forming the cupro-cupric precipitate throughout the mass of ore and which it was impossible to redissolve with sulphuric acid.

5th. The difficulty of treating the remaining 40% of copper in solution as sulphate after the 60% had been precipitated. Scrap iron was not cheaply available.

As sulphur cannot be oxidized directly to sulphuric acid but must be first oxidized to SO_2, and that this step of oxidization is a cheap one, and in fact where sulphide ores are being treated, SO_2 gas is a nuisance, and is to be found as a natural resource for he that can utalize it, as against the higher step of oxidization of SO_2 to sulphuric acid; a 500 ton acid plant at the Tennessee Copper Company at Ducktown costing $1,689,925.00, it becomes quite evident that a great saving in capital invested and operating expenses can be effected, if a process can be evolved that will utilize the raw SO_2 gas. The matter of doing away with the sulphurous gases nuisance, will also be a measure of advantage.

With these several points in view the experimenter undertook to discover a method of effecting the solution of Copper Ores by means of SO_2 gas and moisture, wherein the solution of the copper so obtained would be comparatively stable. Low grade copper ores toward which the beneficiation of the metallic contents thereof these efforts are especially directed, generally contain some little values in gold and silver, which at the present state of the hoped for advancement of the art of metallurgy to that of the science of metalogy could not be overlooked; likewise the recovery of the iron contents or at least a sufficient percentage of same for use in precipitating the copper, or for use as a flux for some nearby smelter that is purchasing its iron from afar, or for use in the arts etc. as may be.

After performing a large number of experiments, it was found that if oxidized or properly roasted sulphide ore of copper was sprayed down a tower, up which a current of sulphurous gases was made to pass, in the presence of moisture containing from 10 to 20% of a soluble chloride or of a mixture of same, that the copper from such a solution was held in solution with sufficient stability as to enable the operator to manipulate said solution almost at will, and in any event without danger of the red cuprose-cupric precipitate of copper forming.

For the purpose of testing the efficacy of soluble chlorides in preventing the formation of the above referred to precipitate, a one lb. sample of oxidized copper ore was used; upon careful analysis the sample was found to contain as per the following annexed results. After a number of tests it was found that crushing the ore to pass a 100 mesh screen gave the best results, treating the pulp for an hour.

1 lb. sample Silicious Oxidized Ore

Heads - assayed

cu	2.3	%
CaO	none	"
Fe	2.2	"
Au	.02 - 40¢ a ton.	
Ag.	trace	

Experiment #1.

After 1 hour treatment 1 lb. lot

Tails assayed

Cu	.08	% -	Extraction	96.5
Fe	1.00	"	"	54.77
Ag	---			
Au	.01	oz. -	"	50%

Experiment #2. - In grinding sample to 150 mesh

Tails assayed - Cu. -	trace -	absolute	extraction
Au. -	"	"	"

Experiment #3 - 1 lb. sample - 1 hour treatment

Head --- <u>sampled as fore-mentioned</u>

Tails - Cu.	%.15	=	Extraction	%	93.48
Au.	oz. .015	=	30¢ "	"	25.00
Fe	% 1.00	=		"	54.77
Ag	----				----

Experiment #4 - 1 lb. sample.

Head -- <u>sampled as forementioned</u>

Tails - Cu.	% .07	=	Extraction	= %	96.06
Au.	oz. 012	=	24¢ "	%	40.00
Ag	------				

Average Extraction -

Cu	%	96.48%
Au		53 7
Ag		total
Fe		54.77

The results having been proven successful beyond expectation, the possibilities of applying this solvent to copper ores on a large scale, became apparent, and further confirmatory any comparative tests were undertaken with results as below recorded.

The Utah Copper Carbonate capping ore is an oxidized copper bearing ore carrying low values in Gold and Silver. The gangue consists principally of silicious material, interstratified and forming a matrix with small amounts of bases such as lime, magnesia, alumina, and oxides of iron. The copper exists mostly as carbonates with minor amounts of the silicate and sulphide of copper. Taken as a whole the ore is a thoroughly oxidized material.

In view of the fact that the Utah Copper capping ore is practically free from sulphides, it has been selected for experimentation to determine whether the Vadner process possesses merit for treating oxidized copper ores for their contained metals of value. The ore, however, has been claimed to be perfectly amenable to the process.

The experiments conducted on the ore were more in the nature of preliminary ones, that is, to determine whether sulphur-dioxide in the presence of a saline solution has a solvent action on the copper, gold and silver as occurring in such oxidized material.

The experiments were conducted by leaching the ground ore with a salt solution of sulphur dioxide, the solution containing 18% sodium chloride and 3% sulphur dioxide. The ore and solution were agitated from time to time, then filtered and washed with an 18% salt solution. The extraction of the metals were calculated from the heads and tails matetial.

For the purpose of comparison, duplicate leaches were made with a saline solution of sulphuric acid containing 18% salt and 3% sulphuric acid and an equivalent of 1 lb. chloride of lime per ton of solution for dissolving the gold. The sulphuric acid leaches were made to note the relative effects of the two acids on treating the material for the contained metals of value.

The material experimented on showed on assay and analysis, the following:

Au	0.016 oz. per ton.
Ag.	0.21 do.
Cu.	1.03%
Pb.	traces
SiO_2	89.7%
Fe.	2.2
Zn	0.1
S.	0.2
CaO	0.7
Al_2O_3	2.3
MgO	0.2

Experiment #1 - With Sulphur Dioxide

Ore ground to pass four mesh.

Time leached 6 hrs.

Composition of leaching solution - 18% NaCl and 3% SO_2

Analysis tails

Au.	0.015 Oz. per ton.
Ag.	0.10 " " "
Cu	0.35%

Extractions

Au.	6.25%
Ag.	52.38%
Cu.	66.%

Exp. #1-A - with sulphuric acid.

Ore ground and leached same as in Exp. #1.

Composition of solution 18% NaCl - 3% H_2SO_4 and $CaOCl_2$

Analysis of Tails.

Au.	0.012 oz per ton.
Ag.	0.16 do.
Cu.	0.34%

Extractions

Au.	25%
Ag.	25%
Cu.	67%

Exp. #3 - with Sulphur Dioxide.

Ore ground to pass 20 mesh.

time leached 6 hrs.

Composition of solution - same as in Exp. #1.

Analysis Tails.

~~A.~~

Au.	0.01 oz per ton.
Ag.	0.085 do
Cu.	0.12%

Extractions.

Au.	37.50%
Ag.	60.00%
Cu.	88.35%

Exp. #2 - A with sulphuric acid.

Ore ground and leached same as in Exp. #2.

Composition of solution - 18% NaCl and 3% H_2SO_4 and $CaOCl_2$.

Analysis tails.

Au.	0.006 oz. per ton.
Ag.	0.11 Do.
Cu.	0.11%

Extractions.

Au.	62.50%
Ag.	47.62
Cu.	89.32

Exp. #3 - with sulphur dioxide.

Ore ground to pass 100 mesh.

Time leached - 6 hrs.

Composition of solution - same as in Exp. #1.

Analysis of Tails

Au.	0.006 oz. per ton.
Ag.	0.08 do
Cu.	0.09%

Extractions

Au.	62.5%
Ag.	61.9%
Cu.	91.26

Exp. #3-A with sulphuric acid.

Ore ground and leached same as in Exp. #3.

Composition of solution 18% NaCl and 3% H_2SO_4 and $CaOCl_2$.

Analysis of tails.

Au.	0.006 oz. per ton.
Ag.	0.07 Do.
Cu.	0.10%

Extractions

Au.	62.5%
Ag.	66.66%
Cu.	90.30%

All of the above experiments are outlined in a tabulated form in the last sheet of this report. The results show, for example, on grinding the material to pass a 20 mesh screen, that the extraction of the copper is approximately the same, viz. 88.35% ext. by SO2 and 89.32% Ext. by H2SO4, while the extraction of gold is greater for the H2SO4 solution but a lesser extraction of silver results with the H2SO4. For the 100 mesh material the extractions for all the three metals are almost identical, the results being over 90% for the copper and over 60% for both the Gold and Silver.

As stated before, the ore contains a few per cent of bases. From this, the result would be that appreciable amounts of sulphuric acid would be consumed by those bases. For example, on the solution resulting from the treatment on the 20 mesh material the following consumption of acid was found:

Pounds H_2SO_4 consumed by	Fe.	18.0	pounds per ton.
Do	Al_2O_3	14.0	Do
	CaO	3.0	
	MgO	1.0	
Total acid consumed by bases		36.0	"
Quantity of acid actually used		65.0	"
Leaving acid to combine with Cu.		29.0	"

On using sulphur dioxide, for example from smelter waste gases, the consumption of the gas would not be an item of account, and in the measure would be superior to the use of sulphuric acid.

Since sulphur dioxide is fully as efficient as sulphuric acid for the treatment of oxidized ores, it would follow that the Vadner process is one of considerable merit on such oxidized material.

Comparison of Sulphur Dioxide and Sulphuric Acid Leaches on Utah Copper

Capping ore

Exp. No.	Mesh Ground to pass	Solution used	Ratio Sol. to Ore	Extraction Cu.	Extraction au.	ag. Extraction
#1	4 mesh	18% NaCl 3% SO_2	1 of ore to 1 sol.	66%	6.25%	52.38%
#1 A	"	18% NaCl 3% H_2SO_4 1# $CaOCl_2$ per ton	do.	67%	25.0%	25.0%
#2	20 mesh	18% NaCl 3% SO_2	"	88.35%	37.5%	60.0%
2 A	do	18% NaCl 3% H_2SO_4 1# $CaOCl_2$ per ton	"	89.32%	62.5%	47.62%
#3	100 mesh	18% NaCl 3% SO_2	"	81.26%	62.5%	61.90%
3 A	do	18% NaCl 1# $CaOCl_2$ per ton 3% H_2SO_4	"	90.30%	62.5%	66.66%

56 1/2

It was found that while the proportion of solution to that of pulp could be as 1 : 1, it was also determined that a proportion of 1½ to 2 parts of solution to 1 part of pulp gave the best results, and was to be recommended for use on a working scale.

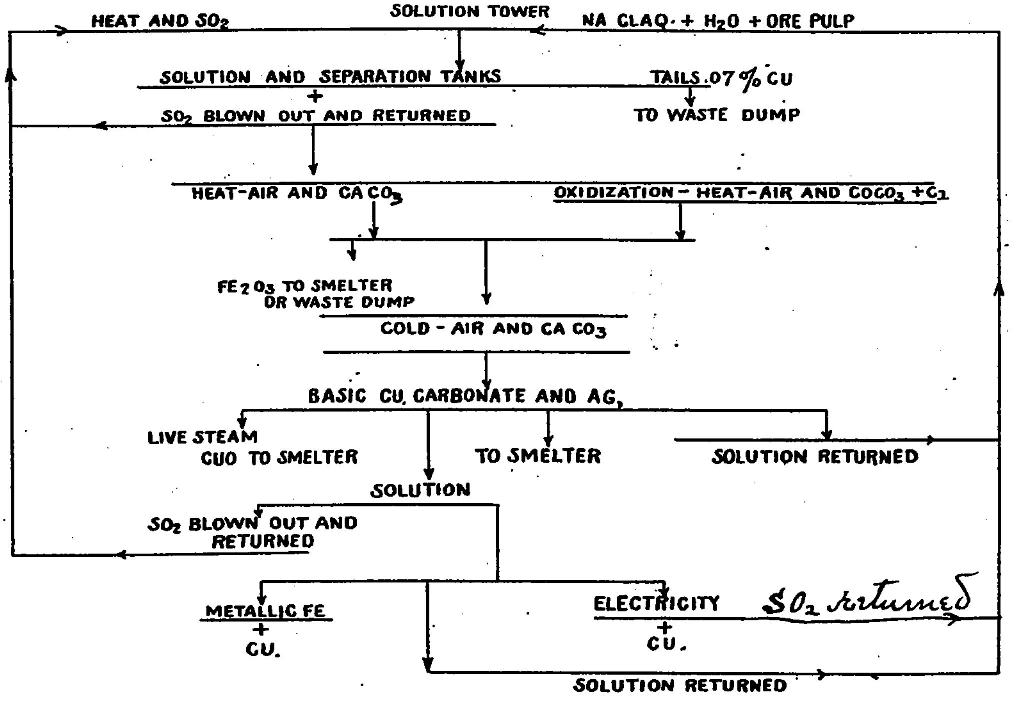
The results of the confirmatory tests so plainly demonstrate the commercial possibilities of this solvent, that further laborious experiments were carried on for the purpose of merging same into a working method whereby copper ores, the value of which was so low as to preclude the recovery of their metallic values by smelting methods, could be treated by utilizing waste and harmful gases from smelters, for the purpose of extracting and recovering said values. Wherefore the general outlines of such a process was evolved and merged into a United States Patent, a copy of which is hereto annexed.

C. S. VADNER.

PROCESS FOR THE TREATMENT OF COPPER ORES AND THE RECOVERY OF THEIR VALUES.

APPLICATION FILED DEC. 9, 1913.

1,111,874. Patented Sept. 29, 1914.



Witnesses:
J. Hansen.
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UNITED STATES PATENT OFFICE.

CHARLES S. VADNER, OF SALT LAKE CITY, UTAH.

PROCESS FOR THE TREATMENT OF COPPER ORES AND THE RECOVERY OF THEIR VALUES.

1,111,874. Specification of Letters Patent. **Patented Sept. 29, 1914.**

Application filed December 9, 1913. Serial No. 805,540.

To all whom it may concern:

Be it known that I, Charles S. Vadner, a citizen of the United States, a resident of Salt Lake City and Salt Lake county, State of Utah, have discovered a new and useful Process for the Treatment of Copper Ores and the Recovery of Their Values.

It is a well known fact that the recovery of copper and other values from ores by hydro-metallurgical methods presents serious difficulties with the final result that the ultimate cost of the copper and other values recovered is very high, except in one or two cases where the method of leaching with sulfuric acids and the subsequent recovery of the copper by precipitating with iron and by electrolysis is followed.

By means of my discovery, low grade ores that have heretofore been considered as carrying too little values to be successfully reduced can be treated at a small cost.

In my specifications where I refer to ores of copper I intend to include oxids, carbonates and sulfids. However the treatment of heavy sulfids would be so slow that it would be necessary to roast them sufficiently before applying my process for the abstraction of their values.

In my process which is diagrammatically represented in the accompanying drawing, naturally oxidized or sufficiently roasted ores (which may also contain other values such as gold, silver, and iron) are crushed to the necessary fineness either dry or in a solution of sodium chlorid. The strength of the solution depends on the amount of copper that the ore contains. The crushing in a salt solution is necessarily applied only in the final stage of the fine crushing. This can be done in a tube mill or other similar device wherein the use of iron is excluded thereby avoiding the danger of precipitating any copper dissolved by the solvent action of the salt solution in the crushing of the ore. By crushing the ore in a salt solution, the solvent action of the same materially reduces the time of exposure of the ores to the final solvent, thereby effecting a saving of time and expense of treatment without materially increasing the cost of crushing. Where no advantage can be derived by crushing as described in the salt solution, or where ores have to be roasted, the ores are first crushed dry and then made into a flowing pulp with a solution of sodium chlorid or any soluble chlorid. This solution can be of from 10 to 20 per cent. and is placed in any suitable tank. The liquid pulp is then showered or sprayed down a tower up which a current of sulfurous gas is caused to pass. The operation is repeated if necessary until the extraction of the copper and other values contained in the ores is completed. The pulp and the solution fall into a sump at the bottom of the tower where it is tested and if the extraction is not satisfactory the pulp is returned to the top of the tower, or, in case that more than one tower is used and the force of gravity can be utilized, it is led by gravity to the top of a second tower. In a case where the extraction is satisfactory, the undissolved pulp and solution is led to a settling tank and allowed to settle. The clear solution is then decanted and permitted to drain. The bottom of the treatment tank may either be made with a filter bottom or the pulp may be filter pressed. The insoluble residue from which the values have been extracted as far as it is found profitable to so do, is rejected. The solution of copper ores by means of sulfurous gas and water is by no means new, but where said treatment has been attempted on a large scale without the use of salt, has been a miserable failure for the reason that the solvent action of the solution is hastened by the use of heat; the resulting solution readily decomposes and precipitates the red cuproso cupric sulfite $(Cu_2SO_3\text{—}Cu\ .\ .\ 2H_2O)$ even in the mass of pulp and when said precipitate is once formed it is practically impossible to redissolve same by a solution of sulfur dioxid in water. Again the solvent action of such a solution on the gold and silver values contained in various ores is not sufficiently completed.

I make use of heat in my process to hasten it. The presence of an excess of salt or other soluble chlorid in the solution precludes the danger of the above red precipitate forming and should a slight precipitation form from the chilling of the solution a little live steam will readily clear it up.

Copper ores are not especially soluble in salt solution but a salt solution impregnated with sulfur dioxid gas becomes a powerful solvent of copper ores and the values contained therein.

1. Leaching solution.

$$XNaCl+XH_2O+XSO_2=X(NaCl\ aq)+H_2SO_3$$

2. Dissolving various ores.

Green copper carbonate:

A. Malachite (as orthocarbonate.)—

$$Cu_4CO_4aq+\underset{aq.}{(XNaCl)}2H_2SO_3=2Cu_2SO_3(XNaCl\ aq)+2H_2O+CO_2$$

B. Malachite (as metacarbonate.)—

$$Cu_2CO_32CuOH+\underset{aq.}{(XNaCl)}2H_2SO_3=\overset{2}{\wedge}Cu_2SO_3\underset{aq.}{(XNaCl)}+3H_2O+CO_2$$

Blue copper carbonate:

C. Lasurite (as orthocarbonate.)—

$$2\,Cu_3HCO_4+\underset{aq.}{(XNaCl)}3H_2SO_3=3Cu_2SO_3\underset{aq.}{(XNaCl)}+2CO_2+4H_2O$$

D. Red oxid copper.—

$$Cu_2O+\underset{aq.}{(XNaCl)}H_2SO_3=Cu_2SO_3\underset{aq.}{(XNaCl)}+H_2O$$

E. Spathic iron ore.—

$$\overset{\text{Ferrous carbonate}}{FeCO_3}+\underset{aq.}{(XNaCl)}+H_2SO_3+2H_2O=Fe_2SO_33H_2O\underset{aq.}{(XNaCl)}+CO_2$$

(F) Ferric oxid.—

$$Fe_2O_3+\underset{aq.}{(XNaCl)}+3H_2SO_3=\overset{\text{Red ferric sulfite}}{Fe_2(SO_3)_3}\underset{aq.}{(XNaCl)}+3H_2O$$

and red ferric sulfite decomposes as follows:

$$Fe_2(SO_3)_3=\underset{\text{Ferrous sulfite}}{2FeSO_3}+SO_2\ +O$$

3. Precipitation of ferric oxid.

$$\overset{\text{Ferrous sulfite}}{2FeSO_3}+(XNaCl\ aq)+CaCO_3\overset{+O}{=}Fe_2O_3(XNaCl\ aq)+SO_2+CO_2\ +CaSO_3$$

4. Precipitation of basic copper carbonate.

$$\underset{\text{Cuprous sulfite}}{2Cu_2SO_3}+(XNaCl\ aq)+2CaCO_3=Cu_4CO_4aq+2CaSO_3+CO_2$$

5. Formation of black oxid of copper.

$$\underset{\text{Basic ortho copper carbonate.}}{Cu_4CO_4aq}=CuCO_3+Cu(OH_2)+H_2O+heat\overset{+2O}{=}\underset{\text{Black oxid of copper}}{4CuO}+CO_2+2H_2O$$

6. Recovery of copper by electricity.

$$Cu_2SO_3+H_2O+electricity=2Cu+H_2SO_3+O$$

and

$$H_2SO_3=SO_2+H_2O$$

7. Recovery of copper by iron.

$$Cu_2SO_3+Fe+3H_2O=2Cu+FeSO_33H_2O$$

The advantages of using sulfur dioxid gas in connection with the salt solution are as follows:

(a) The gas in a dry state has little or no acid reaction, but owing to the presence of water in the salt solution, it imparts to the same a strong acid and quick solvent action.

(b) The ease with which sulfur dioxid is driven out by heat or a current of compressed air from a solution of same in salt enables the useless acidity due to the presence of excess of dissolved sulfur dioxid gas, to be easily and cheaply gotten rid of and recovered for the subsequent treatment of other ores.

(c) The cheapness of salt in general and also the sulfur dioxid gas which can be readily obtained by burning common sulfur or from the roasting of ores by any suitable means.

(d) The fact that in smelting sulfur dioxid gas is a nuisance and its useful application to low grade ores for the extraction of its values is certainly the practical application of scientific chemical metallurgy.

(e) The slight solubility of sulfur dioxid gas as above used amounting from one to two per cent. whereby the cost of heat or compressed air used later on to rid the solution of the excess of said gas is proportionately small.

The clear solution decanted in the treatment tank as above mentioned is treated as follows: If it is desired to separate any iron the same may contain, from the copper the solution is treated while hot, with finely pulverized calcium carbonate whereby the iron is eventually precipitated as (Fe_2O_3) ferric oxid. By not completely neutralizing the solution with calcium carbonate, the amount of iron precipitated can be regulated to suit the purpose of the application of the process. Before precipitating the calcium carbonate, the excess of sulfur dioxid can be gotten rid of either by heat or by blowing compressed air through the solution. The ferric oxid so precipitated is granular and readily settles to the bottom of the tank and is easily separated by decantation from the copper solution the iron can be entirely precipitated by adding finely pulverized calcium carbonate until the solution is neutral and then heating with live steam. Any arsenic present will also be precipitated along with the iron as ferric arsenite. It is not necessary before using this process to oxidize the iron from the ferrous to the ferric condition. By using heat the complete precipitation of the iron can be readily obtained. In case it is found desirable to previously oxidize the iron it can be readily done by compressed air or a small current of chlorin gas. The oxid of iron thus obtained can be used as flux for the smelters or for the production of metallic iron. The resulting solution is allowed to cool and upon addition of more pulverized calcium carbonate together with the use of a current of compressed air blown through the solution, the copper in the said solution is readily and completely thrown down as a green basic carbonate which can be readily separated by any suitable means from the remaining solution. Any silver contained in the solution is precipitated at the same time. If it is desired to recover the copper from the above green precipitate by smelting then this precipitate is washed with water and live steam, whereby the said green precipitate is reduced to the black oxid of copper—CuO—which is of comparatively small bulk and of high values in copper and also contains the precipitated silver. The mother liquor from the precipitation of the copper can be allowed to stand, whereby any excess of calcium sulfate in solution formerly present as calcium sulfite and oxidized to calcium sulfate as above mentioned either in the process of oxidizing the iron from ferrosum to ferricum, or in the precipitation of the copper is allowed to settle out of the solution. The now cool solution can be again used in connection with sulfur dioxid gas as a solvent for more ore. If it is desired the iron and the copper can be precipitated as above described and recovered together. In either case the silver and the gold contained in the ore will be precipitated with the copper and recovered.

If it is desired for any reason to precipitate the copper before the iron, same can be accomplished by allowing the solution to cool and precipitating the copper as above described whereby little or none of the iron will be precipitated. I also provide in my process for the recovery of the copper by the use of iron or electricity where cheap iron or cheap electrical power can be had. Where it is not desirable to smelt the copper precipitated as above described, the same is made into a pulp with water and exposed as before mentioned to sulfur dioxid gas; the resulting solution is freed of the excess of sulfur dioxid as before mentioned and the copper can be recovered where cheap iron is available by ridding the original solution of sulfur dioxid by air or heat and passing the solution over iron in any suitable manner. By this treatment the iron displaces the copper yielding ferrous chlorid and metallic copper which is recovered and cast into pigs. The ferrous chlorid can be used in connection with sulfur dioxid for the further solution of copper ores. The absence of free acids and ferric salts would minimize the amount of iron consumed to the very lowest point and at the same time regenerate a solvent ferrous chlorid which when used by itself would hardly be considered efficient as a solvent of copper ores, but which when saturated with sulfur dioxid is very efficient. The recovery of the copper by electricity where electricity is cheap is accomplished as follows: The copper precipitate is dissolved in a solution of sulfur dioxid in any suitable manner and the copper is recovered by electrolyzing the said solution in any ordinary way at a low voltage. By this means little or no oxidization of the sulfur dioxid gas occurs, a greater part of it being driven off in a free condition especially if the solution is heated. What is driven off can be used for the subsequent solution of more ore. If the slight oxidation that does take place is confined to the calcium sulfite, contained in the solution being oxidized to calcium sulfate, it being quite insoluble, it is precipitated and sinks to the bottom of the electrolyzing tank.

In witness whereof I have hereunto set my hand in the presence of two subscribing witnesses.

CHARLES S. VADNER.

Witnesses:
EVVA OSBORN,
SAMUEL CARLISLE.

RECOVERY OF ZINC

For reasons mentioned on Pages 6,7,12,13 it is evident that the treatment of various zinc ores and the recovery of the metallic values therefrom is a matter of no little importance, both on account of economy and the loss of industrial pursuits to the people of the commonwealth wherein such producing mines are situated.

In testing various zinc ores by the method described under copper, a product called Bag House Mud" was used, This represents the mechanical impurities that are found in smelter fumes, which have had their sulphur trioxide neutralized by oxide of lime or zinc before the fumes containing such and other impurities are passed through the bag house and voided into the atmosphere.

The Bag House Mud analyzed:

Zinc	25.5 %
Silver	4.33 ozs per ton
Gold	0.35 "
Lead	10.00%
Copper	1.06%
Iron	9.1
Lime	0.55
Silica	5.6
Arsenic	6.9

Reduced to pass 100 mesh and treated as mentioned below an extraction of 80% of the zinc contents was readily obtained, Further treatment with the solvent failed to abstract anything but insignificant amounts of zinc; an inspection of the pulp gave rise to the belief that possibly undecomposed sulphides were present, so the pulp was roasted and further treatment with the solvent yielded an extraction of 93.8% of the zinc.

The danger of forming difficult soluble zinc ferrites etc. was not in evidence in these experimental roasting and extraction tests. Roasting of zinc ores followed by satisfactory hydro-metallurgical extraction of their contents has been recorded in California as follows:

Flue dust without any preparation, yielded by the ammonia carbon dioxide process, 72 to 78% of the zinc. In properly roasting sulphide ores from the mine, it was found possible to obtain (by the same method) an extraction of from 80 to 94%. These results agree close enough with those of the author to indicate the ready solubility of zinc ores in other than strong acids, and further that in view of such results, the danger attending the roasting of zinc ores offers no serious problems.

Experimental work tending to utilize sulphurous gases for the extraction of zinc etc. from ores, is described and embodied in the form of a United States Patent and hereto annexed.

The use of calcium carbonate presents no difficulty as the delivery of such an article running 97½ to 98% in purity is nothing uncommon at Smelting Plants both in Utah and California, at a cost of 75 to 92 cents per ton.

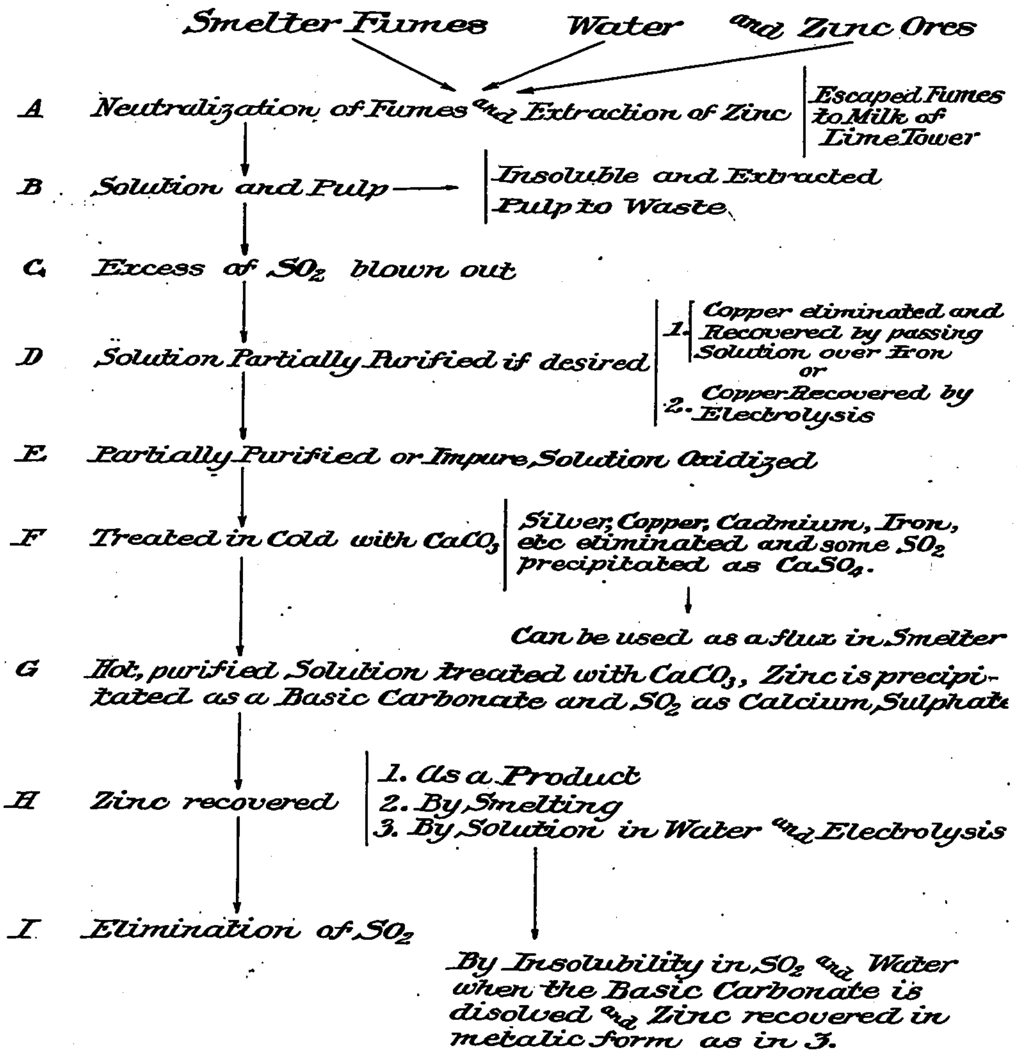
C. S. VADNER.

PROCESS FOR ARRESTING SULFUROUS GASES AND FUMES AND UTILIZING THE HEAT AND GASES CONTAINED THEREIN.

APPLICATION FILED OCT. 29, 1913.

1,110,660.

Patented Sept. 15, 1914.



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THE NORRIS PETERS CO., PHOTO-LITHO., WASHINGTON, D. C.

UNITED STATES PATENT OFFICE.

CHARLES S. VADNER, OF SALT LAKE CITY, UTAH.

PROCESS FOR ARRESTING SULFUROUS GASES AND FUMES AND UTILIZING THE HEAT AND GASES CONTAINED THEREIN.

1,110,660. Specification of Letters Patent. Patented Sept. 15, 1914.

No Drawing. Application filed October 29, 1913. Serial No. 797,949.

To all whom it may concern:

Be it known that I, CHARLES S. VADNER, a citizen of the United States, and a resident of Salt Lake City, Salt Lake county, Utah, have invented a new and useful Process for Arresting Sulfurous Gases and Fumes and Utilizing Heat and Gases Contained Therein.

It is well known that in various smelting operations, in roasting ores and in various other operations that produce sulfurous gases that a large amount of obnoxious fumes are emitted into the air and that these fumes and gases are a source of annoyance and injury to animal and vegetable life. While the mechanical impurities contained in these fumes can be fairly well arrested by the baghouse system and by the use of long flues and baffles, the sulfurous fumes and gases are but partially arrested or rendered innocuous by them.

The object of my invention, then is to utilize these sulfurous fumes and the contained heat and to render them harmless to all forms of life.

Another object of my invention is to provide an effective and inexpensive method of recovering zinc from ores containing it.

To accomplish these objects I make use of the heat and sulfurous gases contained in fumes, subjecting oxids, carbonates and sufficiently roasted ores containing copper, iron, zinc, lead, arsenic, antimony, cadmium lime and bismuth, in the presence of a sufficient quantity of water, to the action of these sulfurous gases. By this means these metals are extracted and recovered from the ores and the fumes arrested and rendered harmless.

It will be understood that if desirable in cases where smelter fumes are not available, sulfur can be burnt in order to provide the necessary sulfurous gases for the reduction of ore.

Oxids, carbonates or sufficiently roasted ores are ground to the proper size and showered or sprayed in water down through the first of a series of purification towers. These towers are of suitable size and height according to the quantity of ore to be treated and the length of time the ore requires to be exposed. The sulfurous fumes are led into these purification towers and mingled with the sprayed ore pulp. By means of this spraying, every particle of ore is subjected to the action of the sulfurous gases, which, in the presence of water, effects the solution of the copper and zinc, some of the iron and arsenic and other metals contained as aforesaid in the ore. This solution falls to the sump at the bottom of the tower while the unused gases pass on to the next tower. These towers are interposed between the points in the flues where the sulfurous gases are all united and the point from which they are finally voided. The sulfurous fumes are drawn either by natural or artificial draft from the main flue in the base of the first purification tower and by a suitable flue led down into the base of a second tower, thence up through the tower and down again to the base of the third tower and so on throughout the series of towers and finally out of the stack. In each tower the fumes come in contact with the sprayed ore pulp so that the sulfurous gases may have a chance to unite with the metals, and fall to the sump at the bottom. As many towers can be interposed in the path of the sulfurous fumes as may be found necessary in order to expose sufficient ore requisitely long to take from the fumes all the sulfurous gases and render them harmless enough to be voided into the air. Where the smelter stack is built on an elevation with the flues leading thereto and with towers interposed in succession in the path of the flues, the sprayed ore pulp can be fed again to the successive towers by gravity, otherwise it can be drawn up by pumps. As a precaution against voiding fumes containing appreciable amounts of sulfurous gases an absorption tower containing boulders of lime rock over which trickles a solution of milk of lime is interposed between the last purification tower and the stack through which the fumes are finally voided into the atmosphere. The solution of milk of lime is returned to the top of the absorption tower by any suitable means as long as it is found efficient in neutralizing the traces of sulfurous acid that may escape from the last purification tower. The resultant calcium sulfite on exposure to the air is quickly oxidized to calcium sulfate which can be dried or dried and calcined and used in the arts. The purification towers are built over sumps into which the solution of ore and dissolved sulfurous vapors and undissolved ore falls. If the settled and undissolved portion of the ore is found to contain sufficient values it is removed from the sump and sent to the smelter, where on account of the lead, iron, gold, and silver it is very desirable. If the solution is found to contain copper, it is now passed in any suitable manner over metallic iron and the copper in the solution is precipitated and settles to the bottom, when it may be withdrawn and purified and so become a marketable product. Air under pressure is forced through the solution in this tank and the excess of sulfurous gas is thereby blown out of the solution. When the ore coming from the solution has only a faint odor, or none at all of sulfurous gas, further air treatment is stopped. If the solution is found to contain copper it is now passed in any suitable manner over metallic iron. The iron displaces copper in the solution and the copper settles to the bottom whence it may be withdrawn and purified and so become a marketable product. Instead of the use of the metallic iron the sealed cover may be removed and the copper recovered by any suitable electrolytic method. After the copper is removed the solution is then oxidized with oxygen, chlorin gas or any other suitable oxidizing agent until all of the iron contained in the solution is in the ferric condition. The oxidized solution is now cooled and treated in the cold. Finely ground calcium carbonate is now added to the solution until further addition of the same causes no effervescence. By this means the whole of the iron is precipitated as ferric hydrate which will react with any arsenic present to precipitate the same along with any copper, cadmium and bismuth that may have escaped previous precipitation. The solution is now allowed to settle and the sediment drawn off, washed and filter pressed and if found to be of any use for its iron contents is sent to the smelter, or worked up if desired, for its arsenic contents, or if valueless thrown away. The clear solution is now treated with live steam and finely ground calcium carbonate until the solution becomes neutral and the zinc in the course of two to four hours treatment is all precipitated in the form of carbonates together with some lime. The solution is settled and decanted. The precipitate is washed and decanted and drawn off on to a draining floor and dried, when the zinc can be recovered by any suitable smelting method. By effecting a solution of said carbonate of zinc by exposing same to the action of sulfurous gases in any of the purification towers, the precipitated carbonate of zinc may be dissolved and the solution drawn off in to any suitable treatment tank and the excess of sulfurous gases blown out by air. The solution may be then electrolyzed by any suitable means and the zinc recovered as metallic zinc.

Having described my process what I claim is new, and wish to secure by Letters Patent is;—

1. The process of recovering zinc and other metals from naturally oxidized and sufficiently roasted sulfid ores containing zinc and other metals which consists in spraying the finely ground pulp of said ores into sulfurous gases, eliminating the copper contained therein by passing the solution over iron, oxidizing the solution, treating same in the cold with finely pulverized calcium carbonate, whereby the ferric hydrate reacts with arsenic and eliminates same from the solution as ferric arsenite, reheating the solution and precipitating the zinc as a basic carbonate by means of additional finely pulverized calcium carbonate.

2. The process which consists in recovering zinc and other metals from naturally oxidized or sufficiently roasted sulfid ores containing zinc and other metals by spraying the finely ground pulp of said ores into smelter fumes containing sulfurous gases, eliminating the copper contained therein by passing the solution over metallic iron, oxidizing and treating the solution in the cold with finely pulverized calcium carbonate whereby the ferric hydrate formed reacts with the arsenic and eliminates same from the solution as ferric arsenite.

3. The process which consists in recovering zinc and other metals from naturally oxidized or sufficiently roasted sulfid ores containing zinc and other metals by spraying the finely ground pulp of said ores into smelter fumes containing sulfurous gases, eliminating the copper contained therein by passing the solution over metallic iron, oxidizing the solution and eliminating the other metals except zinc by precipitating same in the cold by means of a sufficient quantity of finely pulverized calcium carbonate, reheating the solution and precipitating the zinc as a basic carbonate by means of additional finely pulverized calcium carbonate.

4. The process which consists in recovering zinc and other metals from naturally oxidized or sufficiently roasted sulfid ores containing zinc and other metals except zinc by spraying the finely ground pulp of said ores into smelter fumes containing sulfurous gases, eliminating the copper therein by passing the solution over metallic iron, oxidizing the solution and eliminating the other metals by precipitating same in the cold by means of a sufficient quantity of finely pulverized calcium carbonate and reheating the solution and precipitating the zinc as a basic carbonate by means of finely pulverized calcium carbonate, and dissolv-

ing the basic zinc carbonate in sulfur dioxid and water and electrolyzing the solution whereby the zinc is recovered in metallic form.

5. The process which consists in recovering zinc and other metals from naturally oxidized or sufficiently roasted sulfid ores containing zinc and other metals by spraying the finely ground pulp of said ores into smelter fumes containing sulfurous gases, oxidizing the solution and ridding the solution of the other metals contained therein by precipitating the same in the cold by means of a sufficient quantity of finely pulverized calcium carbonate.

6. The process which consists in recovering zinc and other metals from naturally oxidized or sufficiently roasted sulfid ores containing zinc and other metals by spraying the finely ground pulp of said ores into smelter fumes containing sulfurous gases, oxidizing the solution, and ridding the solution of the other metals contained therein except the zinc by precipitating the same in the cold by means of a sufficient quantity of finely pulverized calcium carbonate, heating the solution and precipitating the zinc as a basic carbonate by precipitating the same from the hot solution by means of finely pulverized calcium carbonate.

7. The process which consists in recovering zinc and other metals from naturally oxidized or sufficiently roasted sulfid ores containing zinc and other metals by spraying the finely ground pulp of said ores into smelter fumes containing sulfurous gases, oxidizing the solution, and ridding the solution of the other metals contained therein except the zinc by precipitating the same in the cold by means of a sufficient quantity of finely pulverized calcium carbonate, reheating the solution and precipitating the zinc as a basic carbonate from the hot solution by means of finely pulverized calcium carbonate and dissolving the basic carbonate in sulfur dioxid and water and electrolyzing the solution whereby the zinc is recovered in metallic form.

8. The process of recovering zinc and other metals from naturally oxidized and sufficiently roasted sulfid ores containing zinc and other metals which consists in spraying the finely ground pulp of said ores into sulfurous gases, eliminating the copper contained therein by passing the solution over iron, oxidizing the solution, treating same in the cold with finely pulverized calcium carbonate, whereby the ferric hydrate reacts with arsenic and eliminates same from the solution as ferric arsenite, reheating the solution and precipitating the zinc as a basic carbonate by means of additional finely pulverized calcium carbonate, dissolving the zinc carbonate in sulfur dioxid and water and electrolyzing the solution whereby the zinc is recovered.

9. The process of recovering zinc and other metals from naturally oxidized and sufficiently roasted sulfid ores containing zinc and other metals which consists in spraying the finely ground pulp of said ores into sulfurous gases, oxidizing the solution, treating same in the cold with finely pulverized calcium carbonate, whereby the ferric hydrate reacts with arsenic and eliminates same from the solution as ferric arsenite, reheating the solution and precipitating the zinc as a basic carbonate by means of additional finely-pulverized calcium carbonate.

10. The process of recovering zinc and other metals from naturally oxidized and sufficiently roasted sulfid ores containing zinc and other metals which consists in spraying the finely ground pulp of said ores into sulfurous gases, oxidizing the solution, treating same in the cold with finely pulverized calcium carbonate, whereby the ferric hydrate reacts with arsenic and eliminates same from the solution as ferric arsenite, reheating the solution and precipitating the zinc as a basic carbonate by means of additional finely precipitated calcium carbonate, reheating the solution, precipitating the zinc as a basic carbonate by means of additional finely pulverized calcium carbonate, dissolving the zinc carbonate in sulfur dioxid and water and electrolyzing the solution, whereby the zinc is recovered.

In witness whereof I have hereunto set my hand in the presence of two subscribing witnesses.

CHARLES S. VADNER.

Witnesses:
Marius J. Jacobsen,
Evva Osborn.

RECOVERY OF IRON

The recovery of iron from ores and various products that are finally rejected as slag or otherwise in various smelting operations, and the possibility of working large deposits of low grade copper ores containing more or less iron, together with little or no silver and gold, provided some physical assistance or financial reimbursement could in some way, finally be derived from such a recovery, would certainly be a welcomed addition to the metallurgical practice of the country.

It is well known that a large tonnage of iron is daily used as a flux at the various smelting concerns throughout the United States, and that such iron is finally thrown away in the form of a slag, containing, besides the necessary iron purchased either in the form of "scrap" or as high grade iron ore, some little copper, silver, zinc, possibly a very little gold and in case of lead smelters a small percentage of lead; other metals such as arsenic, cadmium, bismuth, etc., may also be found, so that any percentage of recovery of the above named metals that could be made at a profit either from slag dump, as above mentioned, or from low grade ores containing copper and iron and possibly some other values, would necessarily be worthy of metallurgical attention.

The fact that it is proposed to treat slag dumps and low grade copper or copper iron deposits - material, in fact that thus far has been rejected or thought impossible to utilize - necessarily means, that ways and means heretofore unknown or unused must be applied or such ways and means must be used in combination with the reclamation or utilization of waste products that have heretofore been permitted to go to waste; whether such waste products were merely a source of waste, that no method was known whereby a saving of same could be made at a financial profit and no further damage was sustained by its loss; or whether said waste, besides being a loss in itself, inflicted serious inconvenience and damage upon the person and property of unwilling victims. Special reference is made to the emission of sulphurous gases from various smelting operations and thefinancial losses that are being sustained both by the smelters in question, who are permitting these noxious gases to escape, and by the unwilling victims in attempting to defend themselves and their property.

The extraction of other values contained in a slag of the following composition:

Iron	26.	%
Lime	23	"
Silica	37	"
Zinc	5.7	"
Lead	.7-1½	"
Copper	.12	"
Silver	.5-1	oz. per ton.

becomes, within the possibilities of such a process, as below described while receovering iron from slags. At the X Smelting Plant in one of the western states of the United States, the purchase of iron flux costs in round numbers in the neighborhood of $6,000.00 per month; other operations for the supression of solids in smelter fumes before voiding same in in the air, cost $4,000.00 per month, making a total of $10,000.00 per month or $120,000.00 per annum; the SO_2 gas in the fumes is not however eliminated.

Wherefore, I propose utilizing the sulphurous gases emitted from smelting works or any industry that produces sulphurous gases or even the production of sulphurous gases by burning sulphur or otherwise, for the reclamation of iron from slag dumps or other ores, as may be. I do not merely wish to limit myself to the recovery of iron, as other metals are found associated more or less with iron such as gold, silver, copper, lead, bismuth, cadmium, arsenic, antimony, zinc, lime, in fact all the present known metals that can be recovered by the proposed process and the recovery of which would be a source of profit in addition to the objective ends of the application of the process.

Various suitably pulverized smelter slags, and ores containing gold, silver, lead, bismuth, cadmium and antimony, are exposed in the presence of heat, which is either artificially generated or abstracted from the slag by using same while in a heated condition, to hot sulphurous gases either from smelters or artificially generated.

The suitably pulverized slags and ores are preferably showered or sprayed down towers up which the said gases are made to pass, or otherwise exposed to the solvent action of said sulphurous gases in the presence of heat and moisture a sufficiently long period of time for the purpose of effecting the solution of the metal and metals that it is desired to extract.

This operation can be performed over a sump, so that the solvent action can be prolonged until the desired results are obtained, any method of carrying on this operation as will best suit in each particular case will necessarily be installed.

In certain instances, particularly where copper, zinc, silver and gold are sought to be extracted, the addition of soluble chlorides, one or more, the presence of which favors the solution of the above enumerated metals in the sulphurous solution, either by their own solvent powers or due to their presence in preventing secondary reactions from taking place that would tend to render the operation of the process more expensive, or in some cases, prohibitive, is desirable, and when found to be necessary, is made. I would not desire to limit myself to the above enumerated metals, but would use a solution of the soluble chlorides containing at least one of the common chlorides capable of reacting with the metals, such as sodium chloride, magnesium chloride, calcium chloride, ferrous and ferric chloride and cuprous and cupic chloride in operating the process either in the commencement of same or at any state where the use of said chloride and chlorides could be advantageously used in the extraction of any and all of the aforesaid metals from their ores.

I intend to use any one of the soluble chlorides, either singly or in combination, that are capable of reacting with the metals for the purpose of preventing a secondary reaction and a precipitation of metal.

The solution of the ore as may be, having been satisfactorily accomplished by means of sulphurous gas as afore mentioned either with or without the use of one or more of the soluble chlorides mentioned above, is separated from the insoluble residue by decantation or filtration into suitable containers and the excess of sulphurous gases are gotten rid of, either by heat, steam or air, and the surplus acidity of the solution due to acid salts, free acid or otherwise, neutralized, partially or wholly as may be necessary, with the carbonates, oxides or hydrates of calcium, magnesium, sodium, potassium or with a mixture of same as may be possible.

When iron is present, the solution is first partially neutralized and oxidized or not, as desired, with air, chlorine or any other suitable oxidizing agent. The solution, hot or cold, is then exposed in any suitable manner to the action of air or oxygen; the iron contained in said solution can thus be partially or wholly precipitated as a brownish red pulverulent powder Fe_2O_3 (ferric oxide) at will by the operator, simply by regulating:

1st.......The acidity of the solution:

2nd.......The length of time of the treatment of the solution with air or oxygen;

3rd.......The degree of strength of the solution of the soluble chlorides, especially while using sodium chloride, calcium chloride or a mixture of the soluble chloride;

4th.......The temperature of the solution;

5th.......The condition of the iron after it has been leached out of the ore and while in the leaching medium.

Thus, $2\ FeSO_3 + SO_2 + O = Fe_2\ (SO_3)_3$
Ferrous Sulphite = Ferric Sulphite.

$$Fe_2\ (SO_3)_3 + Fe_2O_3 + 3\ SO_2$$

The neutralizing agent added takes no part in the reaction, merely reducing the acidity of the solution thus:

$$SO_2 + H_2O + CaCO_3 = CaSO_3 + CO_2 + H_2O$$

Freshly precipitated ferric oxide is soluble in moderately strong acid solution, hence the necessity of reducing the acidity in order to obtain the precipitate.

PHENOMENON OF HYDROLYSIS

Hydrolysis ≠ ὕδωρ = water
λύσις = act of loosening= decomposition
by means of water.

Here, applying one of the rules laid down in the fore part of this thesis, (that natural resources must be made use of etc.) comes the problem of cheaply recovering the iron from solution. Having gotten the iron in solution by means of a cheap solvent, SO_2 gas and water, and admitting that water by virtue of its hydrogen ion concentration may be regarded as able to play the part of an acid, and that dilution is not a factor of the degree of hydrolysis of the combination of a weak base and a weak acid, and that SO_2 is volatile and can easily be eliminated from its solution, and, that ferrosum can readily be oxidized to ferricum in a sulphite solution without oxidizing the sulphite to a sulphate in appreciable quantities;

HYPOTHESIS

The hypothesis is advanced that between a weak acid like $(SO_2 + H_2O)$ and a weak base like Fe_2O_3 (that we hope to produce) that there will be practically no combination, excepting that of the solution of nascent Fe_2O_3 in SO_2 and water, but what can be easily and readily distupted by water and the iron recovered in the form of an oxide, on elimination of excess of the volatile radical SO_2

PROOF

By simply blowing air through the solution of ferrous sulphate (Fe_2O_3) as above obtained, the excess SO_2 gas is quickly eliminated. The iron is quickly oxidized to the ferric condition (the time of the reaction as well as the character of the precipitate can be regulated by the addition of a neutralizing agent, such as calcium,carbonate and the like) and ferrous sulphite becomes ferric sulphite, thus:

$$2\ FeSO_{3\ aq.} + SO_2 + O = Fe_2\ (SO_3)_{3\ aq.}$$

and the ferric sulphite thus formed is as quickly decomposed into ferric oxide and SO_2 gas. Thus:

$$Fe_2\ (SO_3)_3\ aq. = Fe_2O_3\ aq. + 3\ SO_2$$

The precipitate oxide of iron settles quickly and easily and the SO_2 gas is eliminated.

DEDUCTION

That the correctness of the above advanced hypothesis is borne out by the results of experimental data recorded.

CONCLUSION

That the practical application of this method, for the recovery of iron from ores and solutions, is feasible, and when applied with due regard to economic conditions should become a valuable process in metallurgy, and another step that is slowly but surely raising hydro-metallurgy to the ranking title of Metallogy, the Science of Metals.

In case that the ores used contain arsenic and it is desired to eliminate and recover the arsenic, a different form of precipitating the iron is used. The solution of the ore is effected and the excess of SO_2 gas is gotten rid of as above mentioned and the solution is oxidized or not, as may be desired, by a suitable oxidizing agent and the iron is precipitated as ferric hydrate $Fe_2(OH)_6$ by the addition of a sufficient amount of neutralizing agent or agents (or a combination of same) such as the oxides, carbonates or hydrates of barium, calcium, strontium, magnesium, sodium, potassium or a mixture of same can also be used. When the iron is precipitated in this form, the ferric hydrate $Fe_2 (OH)_6$ formed has the power of uniting with the arsenic present and forming either ferric arsenite or ferric arsenate as may be, which is precipitated and removed by filtration and other suitable ways from the solution.

Antimony, silver, gold, lead, bismuth, cadmium and other known metals that would be found dissolved in such a solution, depending of course on the contents of the ore and ores being treated, can be precipitated as above described and recovered, if found to be desirable. This method of eliminating and purifying the solution of any and all undesirable metals as aforementioned, can be used, whether the iron is eliminated and recovered by the first or second mentioned method, and also when a combination of the two methods is necessary to effect an elimination of the above mentioned metals for the purpose of purifying the solution. It is well understood that any and all steps in this process can be carried out in the presence of heat or cold, as may be most advantageous. Copper can also be eliminated by passing the solution, hot or cold, in any suitable manner over metallic iron. The condition of the solution as to the amount of free acid and state of oxidization can be regulated to suit the best purpose of the application of the process. The copper and silver can also be precipitated as a basic carbonate by the suitably pulverized carbonates of the alkalies, alkaline earth or a mixture of same with or without the use of air and in the cold.

The precipitate is separated by filtration and other suitable means, and the copper, silver and other contained metals as may be, are recovered by electrolyzing the hot or cold solution in suitable containers. The excess acidity of the solution, due to the decomposition of the sulphites, is gotten rid of both by heat and also by the carbonates of the metals, alkalies, alkaline earths or a mixture of same used in effecting the precipitation of the copper. Sulphites are formed which may precipitate out of the solution and which can be separated, and SO_2 gas and their valuable basic contents recovered therefrom. A current of air will also assist in reducing the acidity of the solution. The formation of calcium sulphate due to oxidation is effective in reducing said acidity, and any such sulphate formed can be separated and recovered for use.

It is my intention of affecting the functioning of this process for the purpose of cheapness of operation by:

1st.....Using SO_2 gas and water;

2d .....Using SO_2 gas, water and heat;

3d .....Using SO_2 gas, water and a soluble chloride; or a mixutre of soluble chlorides as aforementioned, along with heat when necessary; or using the above means in part, together or in combination, also using the soluble chlorides mentioned in part, together or in combination for the purpose of extracting the metals aforementioned.

The strength of the solution in soluble chlorides will vary from 5% to 20% as may be.

The solution being freed from the aforementioned metals as may be and in the manner described for the best purpose of the application of the process, the zinc is precipitated from cold solution by the means of the suitably pulverized neutralizing agent, such as oxides, carbonates, hydrates of barium, calcium, strontium, magnesium, sodium, potassium. A mixture of same can be used, and the basis precipitate of zinc is separated by filtration or otherwise and calcined to the oxide, smelted or dissolved in any suitable manner in a solution of SO_2 and water with or without heat and electrolyzed in any suitable manner, whereby the zinc is recovered in metallic form.

The precautions for theremoval of the acidity of the solution and recoveries of the by-products, as in the elimination of copper, should be duly observed.

Copper, arsenic, antimony, bismuth, lead, cadmium, mercury, tin, gold and silver can also be precipitated from an acid solution, as a sulphide, by a suitable precipitant, such as H_2S and the like. Copper and silver can also be precipitated by passage over iron, the precipitated metals being eliminated, the solution containing the iron, zinc, nickel, cobalt, manganese, etc. that are not precipitated as sulphides from acid solutions, is treated as aforementioned.

Whatever method is employed, the precipitated metals are eliminated, the solution is neutralized as before-mentioned, and by the aid of heat and air the zinc can be precipitated as a basic carbonate to be recovered as mentioned under zinc recovery; it being possible to precipitate copper as a basic carbonate by means of air from a cold solution containing one of the common chlorides. The SO_2 gas is eliminated as noted, and re-used along with water or with barren solution in regenerating the solvent for the leaching of more ore, and repeating the cycle of solution, precipitation and regeneration until the metallic contents of the ore is sufficiently exhausted; it being well understood that the application of the methods herein described, being applied in each individual case, guided by the necessary experimental work, previously carefully made.

Respectfully submitted,