Title: PROCESS FOR RECOVERING A BASE METAL FROM A SULFIDE ORE MATERIAL

Abstract: The present invention relates generally to processes for mineral extraction and ore processing and more specifically, to processes for recovering iron as hematite from a sulfide ore material and to processes for recovering base metals from sulfide ore materials. A process for recovering a base metal from a sulfide ore material involves performing a leaching step on the sulfide ore material at atmospheric pressure in the absence of a lixiviant, using a leachant to obtain a solid residue in the nature of an upgraded base metal concentrate and a leachate containing dissolved iron therein. The lixiviant includes an acid and a chloride. The acid may be selected from the group consisting of an organic acid, sulfuric acid, sulfuric acid and hydrochloric acid. The chloride may be selected from the group consisting of magnesium chloride, calcium chloride, sodium chloride, potassium chloride, ferrous chloride and lithium chloride. Hydrogen sulfide gas formed during the leaching step is removed and at least a portion of the hydrogen sulfide gas formed is reacted with molten copper to form cuprous sulfide and hydrogen gas. The hydrogen gas formed may be collected for use as an energy source. The leachate is separated from the upgraded base metal concentrate and the upgraded base metal concentrate is subjected to a series of base metal recovery steps. The leachate may be further treated to recover therefrom at least some of the iron as a hematite product of high purity, as well as the constituent acid and chloride of the lixiviant for reuse in the leach circuit.
For two-letter codes and other abbreviations, refer to the "Guidance Notes on Codes and Abbreviations" appearing at the beginning of each regular issue of the PCT Gazette.
PROCESS FOR RECOVERING A BASE METAL FROM 
A SULFIDE ORE MATERIAL

FIELD OF THE INVENTION

0001 The present invention relates generally to processes for mineral extraction and ore processing and more specifically, to processes for recovering iron as hematite from a sulfide ore material and to processes for recovering base metals from sulfide ore materials.

BACKGROUND OF THE INVENTION

0002 Various hydrometallurgical techniques have been developed for recovering base metals, such as zinc, nickel, copper and cobalt, from base metal sulfide and oxide ores. One such technique involves leaching the base metal sulfide ore with a lixiviant that promotes dissolution of one or more base metals in the leaching solution. The base metals may then be recovered by using known base metal separation techniques. Over the years, various compounds have been used individually as leaching agents in the lixiviant, for instance, sulfuric acid, hydrochloric acid, ferric chloride, ferric sulphate, cupric chloride and magnesium chloride. Of late, much work has been done in the area of chloride-based leaching processes.

0003 One such chloride-based leaching process is described in United States Patent Publication No. 2005/0118081 of Harris et al. The process involves leaching a base metal sulfide ore at atmospheric pressure with a lixiviant containing a relatively low concentration of hydrochloric acid and a high chloride concentration. More specifically, the lixiviant used in this process comprises hydrochloric acid, a chloride and an oxidant. The chloride may be an alkali metal chloride, magnesium chloride, calcium chloride and mixtures thereof. The oxidant may be an alkali metal peroxide, an alkali metal perchlorate, magnesium perchlorate, alkali metal chlorate, earth metal perchlorate, chlorine, alkali metal hypochlorite, hydrogen peroxide and peroxysulfuric acid, and mixtures thereof. The leaching step yields a solid residue and a base metal-rich leachate which contains dissolved iron therein. The sulfur from the leachate and solid residue may be removed by forming and stripping hydrogen sulfide during the leaching step. Some or all of the hydrogen sulfide may then be converted to elemental sulfur. A by-product of this conversion reaction is energy. The base-metal rich leachate may be subjected to base metal recovery steps (which may include an additional
leaching step) and/or value metal recovery steps to extract the nickel, copper, zinc and cobalt
and any dissolved platinum group metals, gold and silver.

0004 In this process, iron is first removed from the pregnant leachate by precipitating an
iron oxide (hematite or spinel) using a magnesium oxide additive and effecting a solid/liquid
separation step. The magnesium oxide is obtained from a pyrohydrolysis reaction of spent
magnesium chloride, which also recovers hydrochloric acid for recycle. The process only
contemplates the recovery of hydrochloric acid formed as a by-product of the pyrohydrolysis,
for reuse in the leach circuit.

0005 Iron is and has always been considered a major problem in hydrometallurgical
processes. In atmospheric processes, the iron is usually precipitated as an oxy-hydroxide, and
in higher temperature autoclave processes, as an impure hematite.

0006 United States Patent No. 3,682,592 issued to Kovacs describes a process for
recovering HCl gas and ferric oxide from from waste hydrochloric acid steel mill pickle
liquors (commonly referred to as "WPL"). WPL typically contains water, 18 to 25% weight
of ferrous chloride (FeCl₂), less than 1% weight ferric chloride (FeCl₃), small amounts of free
hydrochloric acid and small amounts of organic inhibitors. The process of Kovacs includes
two steps namely, a first oxidation step and a second thermal decomposition step. During the
first oxidation step, the ferrous chloride in the WPL is oxidized using free oxygen to obtain
ferric oxide and an aqueous solution containing ferric chloride. No hydrochloric acid is
liberated at this stage. The first oxidation step is carried out under pressure (preferably, 100
p.s.i.g.) and at an elevated temperature (preferably, 149°C).

0007 During the second step, the resultant ferric chloride solution is thermally decomposed
to obtain ferric oxide and HCl gas, which is recovered as hydrochloric acid. More
specifically, the resultant solution is heated up to 175-180°C at atmospheric pressure. The
HCl is stripped off at a concentration of 30% with >99% recovery and good quality hematite
is produced. While recovery of hydrochloric acid and hematite may be achieved using this
process, its application tends to be limited to liquors containing only ferrous/ferric chlorides.
When other chlorides are present in the solution, for instance, magnesium chloride, the
activity of the chloride ions and protons tends to be too high to permit any reaction to take
place simply by heating the solution to the desired temperature indicated by Kovacs.
Accordingly, this process tends not to be well adapted for use in leaching processes involving chlorides other than ferric chloride.

0008 J.E. Dutrizac and P.A. Riveros in a technical paper entitled "Precipitation of Hematite from Ferric Chloride Media", Hydrometallurgy 46 (1997), pages 85-104, showed that it was possible to precipitate a pure form of hematite from dilute ferric chloride systems, but that only a small part of the total iron could be precipitated. The presence of additional chlorides further suppressed hematite precipitation, as did hydrochloric acid above 0.6M in concentration. The system used was closed (in an autoclave) and no attempt was made to recover the hydrochloric acid. J.E. Dutrizac, in an earlier paper, "Jarosite Formation in Chloride Media", Proceedings of the Aus. IMM, No. 278, June, 1981, pages 23-32, similarly showed that jarosites could be precipitated from chloride solutions containing sulfate ions. But, as with hematite, the presence of a very small amount of hydrochloric acid, this time less than 0.1M suppressed the reaction. Only alkali and lead jarosites and not hydronium jarosite could be formed. To date, no methods to recover substantial amounts of iron in a useful form have been developed for application in chloride-based leaching processes.

0009 Other chloride-based circuits have also been proposed for the recovery of nickel and cobalt amongst other metals. One such circuit is disclosed in International PCT Publication Number No. WO 02/053788 of Lalancette. This published patent application describes a method for the recovery of base and precious metals using various extractive chloridation techniques. Lalancette teaches effecting chloridation using chlorine in the presence of a source of chloride ions, such as sodium chloride, potassium chloride and calcium chloride. This chloridation is followed by a lixiviation step using hydrochloric acid. This type of chloridation may be carried out at low temperatures (40-50°C) or high temperatures (500-600°C). Additionally, Lalancette describes performing chloridation by leaching a base metal mixture containing substantial amounts of iron with hydrochloric acid at a temperature of 100°C, in the presence of an oxidant. This chloridation transforms part of the iron into ferric chloride which, in sulfide ores, can only be achieved in the presence of an oxidant. The ferric chloride solution is then heated to evaporation while in the presence of moisture. Thereafter, the resultant ferric chloride is subjected to hydrolysis to thereby transform it into ferric oxide and hydrochloric acid. According to Lalancette, the metal chlorides remaining in the final solid mixture are not affected by this mild hydrolysis and can be separated from the ferric oxide solids by leaching with water. However, experiments conducted by the applicant have
shown that this is not the case. More specifically, it was found that the metal chlorides would hydrolyse to a greater or lesser degree depending on the temperature and thus would be lost in the iron residue.

0010 In light of the foregoing, it would be advantageous to be able to separate and selectively recover substantial amounts of iron from a base metal sulfide ore in a generally useful and relatively pure form, while avoiding the need for an energy-intensive and expensive step such as pyrohydrolysis. It would be further desirable to have the recovery of iron in this fashion occur within a process which tends to yield generally high rates of recovery for base metals, permits recovery of elemental sulfur and the acid constituent in the lixiviant, as well as generates some energy.

0011 In United States Patent Publication No. 2005/0118081 of Harris et al. the production of hydrogen sulfide gas is undertaken with an aim to producing elemental sulfur. Harris et al. does not contemplate splitting the hydrogen sulfide into its constituent elements. Several studies have been conducted in this field, a summary of which may be found in a report prepared by E.A. Luinstra entitled "Hydrogen from H2S: Technologies and Economics", submitted to Alberta Energy - Alberta Hydrogen Research Program; Hydrogen Industry Council; CANMET; PanCanadian Petroleum Limited; Shell Canada, Limited; Suncor Inc., 011 Sands Group; Syncrude Canada Limited; Industrial Hydrogen Chair, University of Calgary; May 26, 1995. In this report, the use of molten copper to split the hydrogen sulfide was only briefly mentioned, and not recommended as an approach.

0012 In another report prepared by G.N. Krishnan and D.L. Hildenbrand, entitled "Recovery of Hydrogen from Hydrogen Sulfide: Bench Scale Studies of the Reaction of Hydrogen Sulfide with Liquid Copper, Final Report", Brookhaven National Laboratory, Report BNL 51624, May 1982, studied the reaction between hydrogen sulfide and molten copper. It was concluded that although the reaction was feasible, it was only economical if there was a market for by-product sulfuric acid, and further work was discontinued. None of these studies describe or teach commercially viable methods for utilizing hydrogen sulfide to generate hydrogen. Accordingly, there is a need for a base metal recovery process that can incorporate a hydrogen sulfide-splitting step. Advantageously, such a process would yield fuel for generating on-site electric power, for instance, for use within the ore processing operations.
SUMMARY OF THE INVENTION

0013 In accordance with a broad aspect of the present invention, a process for recovering a base metal from a sulfide ore material is provided. The process includes providing the sulfide ore material and providing a lixiviant comprising an acid and a chloride. The acid may be selected from the group consisting of an organic acid, sulfurous acid, sulfuric acid and hydrochloric acid. The chloride may be selected from the group consisting of magnesium chloride, calcium chloride, sodium chloride, potassium chloride, ferrous chloride and lithium chloride. The process further includes performing a leaching step on the sulfide ore material at atmospheric pressure in the absence of an oxidant, using the lixiviant to obtain a solid residue and a leachate containing dissolved iron therein, and separating the leachate from the solid residue. The solid residue is an upgraded base metal concentrate. Hydrogen sulfide gas formed during the leaching step is removed and at least a portion of the hydrogen sulfide gas formed is reacted with molten copper to form cuprous sulfide and hydrogen gas. The hydrogen gas formed is collected for use as an energy source. The process further includes separating the leachate from the solid residue and treating the leachate to recover therefrom at least some of the iron as a hematite product of high purity. The solid residue is subjected to a series of base metal recovery steps.

0014 In an additional feature, the sulfide ore material is selected from the group consisting of an ore, a concentrate and an intermediate material. In another feature, the sulfide ore material contains at least one base metal selected from the group consisting of nickel, copper, zinc and cobalt. Optionally, the sulfide ore material contains at least one of gold, silver and a platinum group metal. The sulfide ore material may be selected from the group consisting of pyrrhotite, pentlandite, chalcopyrite, pyrite, arsenopyrite, galena and sphalerite.

0015 In a further feature, the concentration of the chloride in the lixiviant is adjusted to obtain a solubility of between about 75% and about 95% of its saturation point. In yet another feature, magnesium chloride is selected as the chloride.

0016 In still another feature, the acid in the lixiviant is an organic acid. The organic acid may be selected from the group consisting of acetic acid, tartaric acid and citric acid.

0017 In another feature, the acid is hydrochloric acid and the chloride is magnesium chloride. The concentration of magnesium chloride in the lixiviant is adjusted to at least
about 300g/L. In an additional feature, the concentration of magnesium chloride in the lixiviant is adjusted to between about 340g/L and 420 g/L.

0018 In a further feature, the leaching step is performed at a temperature between about 20°C and about the boiling point of the lixiviant. In still another feature, the leaching step is performed at a temperature between about 105°C and about 110°C.

0019 In an additional feature, the step of treating the leachate to recover therefrom at least some of the iron as a hematite product of high purity, includes heating the leachate to distill the hydrochloric acid therefrom, and simultaneously subjecting the leachate to a precipitation step to promote the formation of the hematite product. Preferably, the leachate is heated to a temperature of between about 190°C and about 250°C and most preferably, between about 220°C and about 250°C. In another feature, the leachate is heated to a temperature of at least about 180°C and the precipitation step includes one of adding water and adding steam to the leachate. In a further feature, the precipitation step includes one of adding water and adding steam to the leachate. Additionally, the process includes separating the precipitate from the remaining leachate and drying the precipitate.

0020 In another feature, the process further includes recovering the chloride from the remaining leachate.

0021 In yet another feature, the process further includes removing hydrogen sulfide gas formed during the leaching step. Optionally, the hydrogen sulfide is stripped in a continuous manner. In one feature, at least a portion of the hydrogen sulfide gas formed is reacted with sulfur dioxide gas in a Claus reaction to obtain elemental sulfur and steam. In another feature, the cuprous sulfide formed is exposed to oxygen to obtain molten copper and sulfur dioxide gas and the sulfur dioxide gas is converted to sulfuric acid.

0022 In a further feature, the solid residue is an upgraded base metal concentrate. The upgraded base metal concentrate is subjected to a series of base metal recovery steps that includes providing a second lixiviant comprising an acid and a chloride. The acid may be selected from the group consisting of an organic acid, sulfurous acid, sulfuric acid and hydrochloric acid. The chloride may be selected from the group consisting of magnesium chloride, calcium chloride, sodium chloride, potassium chloride, ferrous chloride and lithium chloride. The process further includes performing a second leaching step on the upgraded base metal concentrate at atmospheric pressure, using the second lixiviant to obtain second
solid residue and a second leachate containing at least one base metal dissolved therein. The second leachate is then separated from the second solid residue and treated to recover therefrom the at least one base metal.

0023 In an additional feature, the sulfide ore material contains at least one of gold, silver and a platinum group metal and the second solid residue is an upgraded value metal concentrate. The upgraded value metal concentrate is subjected to at least one value metal recovery step. In another feature, the sulfide ore material contains at least one of gold, silver and a platinum group metal and the second leachate is a value metal-rich leachate. The value metal-rich leachate is subjected to at least one value metal recovery step.

0024 In a further feature, the base metal containing material is a laterite ore material and the leachate is a base metal-rich leachate. The base metal-rich leachate is subjected to a series of base metal recovery steps.

BRIEF DESCRIPTION OF THE DRAWINGS

0025 The embodiments of the present invention shall be more clearly understood with reference to the following detailed description of the embodiments of the invention taken in conjunction with the accompanying drawings, in which:

0026 FIG. 1 is a schematic representation of a process according to an embodiment of the invention involving the removal of iron material from a sulfide ore material using a lixiviant in a leaching step to obtain a solid residue in the nature of an upgraded base metal concentrate and a leachate containing dissolved iron therein; and

0027 FIG. 2 is a schematic representation of a process for recovering base metals and value metals from the upgraded base metal concentrate obtained by the process shown in FIG. 1.

DETAILED DESCRIPTION OF EMBODIMENTS OF THE INVENTION

0028 The description which follows, and the embodiments described therein are provided by way of illustration of an example, or examples of particular embodiments of principles and aspects of the present invention. These examples are provided for the purposes of explanation and not of limitation, of those principles of the invention. In the description that follows, like parts and/or steps are marked throughout the specification and the drawings with the same respective reference numerals.
With regard to nomenclature, the term "ore material" as it is used throughout the specification in connection with the processes described herein, means an ore and any material derived from the processing of an ore, including without limiting the foregoing, any metal processing by-product (i.e. flue dust and furnace baghouse dust), any intermediate material produced during the treatment of an ore (i.e. slags, calcines, impurity residues, dross and anode slimes), any concentrate, any matte such as a converter matte, or any tailings from the processing of an ore.

Referring to FIG. 1, there is shown a schematic representation of a process in accordance with an embodiment of the invention, designated generally with reference numeral 20. Broadly speaking, the process 20 involves the removal of iron material from a base metal containing ore material using a lixiviant during a leaching step. More specifically, the process 20 includes the following steps: providing a base metal containing material in the nature of a sulfide ore material (step 30); providing a lixiviant (step 40); performing a leaching step on the sulfide ore material at atmospheric pressure, using the lixiviant to obtain a solid residue and a leachate containing dissolved iron therein (step 50); separating the leachate from the solid residue (step 60); and treating the leachate to recover therefrom at least some of the iron as a hematite product of high purity (step 65).

In the present embodiment, the sulfide ore material contains relatively substantial amounts of sulfur and iron, as well as one or more base metals, such as nickel, copper, zinc and cobalt. The sulfide ore material may further contain one or more value metals such as gold, silver or a platinum group metal (PGM). The type and quantity of the value metal contained in the sulfide ore material will depend on the source of the ore material. Suitable sulfide ore materials for use in process 20 include pyrrhotite, pentlandite, chalcopyrite, pyrite, arsenopyrite, galena, sphalerite and any aggregates and/or mixtures thereof.

In the present embodiment, the sulfide ore material does not require substantial treatment (i.e. roasting or flotation) prior to being subjected to the process shown in FIG. 1. However, it may be advantageous to perform certain well-known physical conditioning steps on the sulfide ore material. For instance, the sulfide ore material may be subjected to beneficiation or may be crushed and/or ground (at step 70) as shown in FIG. 1. The performance of such conditioning steps tends to improve the overall efficiency of the process by reducing the residence time of the lixiviant in the sulfide ore material and by encouraging
the dissolution of iron in the leachate thereby leading to a reduction in the volume of solid residue obtained from the leaching step 50.

0033 The lixiviant used in the leaching step 50 includes an acid and a chloride. Contrary to other known hydrometallurgical processes applicable to sulfide ore materials, the lixiviant does not contain an oxidant and the leaching step is carried out under predominantly reducing conditions. The acid in the lixiviant may be an organic acid, sulfurous acid, sulfuric acid or hydrochloric acid. The preferred acid is hydrochloric acid. Examples of an organic acid that may be used in performing the leaching step 50, include acetic acid, tartaric acid and citric acid.

0034 The lixiviant employs relatively low concentrations of acid. The amount of acid to be used in the lixiviant tends to depend on the chemical composition of the sulfide ore material. Different sulfide ore materials require different amounts of acid in the lixiviant. However, where hydrochloric acid is selected, it has been found that between about 500 kg to about 1000 kg (100% dry basis) of hydrochloric acid may be needed for each tonne of sulfide ore material to be leached. Preferably, the amount of acid in the lixiviant is substantially stoichiometric. However, in certain instances, it may be desirable to add acid in an amount slightly in excess of the stoichiometric amount. For example, in the case of pyrrhotite, between about 100% and about 110% of the stoichiometric amount of acid may be used. It is believed that the high activity of $\text{H}^+$ ions in the high strength chloride solution may make the use of close to stoichiometric concentrations of acid in the lixiviant possible.

0035 The chloride constituent in the lixiviant may be magnesium chloride, calcium chloride, sodium chloride, potassium chloride, lithium chloride, ferrous chloride or mixtures thereof. The preferred chloride for use in the lixiviant is magnesium chloride as it tends to be more readily recycled than the other specified chlorides. Magnesium chloride is most preferred where the acid employed in the lixiviant is hydrochloric acid. As will be explained in greater detail below, where the lixiviant includes hydrochloric acid, it will be possible to precipitate more of the iron in the leachate as hematite or magnetite during a hydrolytic distillation stage. The hematite or magnetite thus formed is of high purity and may be easily recovered.

0036 Where magnesium chloride is used in the lixiviant with hydrochloric acid, there exists the possibility that magnesium oxychloride may be formed during the hydrolytic distillation
stage, the formation of which would contaminate the hematite and result in a loss of chloride from the system. However, the risk of this occurrence tends to be relatively small. On the other hand, magnesium chloride tends not to suffer from some of the drawbacks associated with some of the other above-identified chlorides. For instance, sodium chloride and potassium chlorides are prone to crystallisation and calcium chloride tends to form a stable sulfide, which on oxidation causes insoluble calcium sulfate (gypsum) to precipitate, resulting in significant scaling problems in the reactors and piping.

0037 The chloride concentration of the lixiviant prior to leaching is adjusted to obtain a solubility in the range of 75-95% of its saturation. This adjustment will yield different amounts of chloride ion initially in solution given that the solubility of the specified chlorides varies considerably. The chloride concentration may also be adjusted to take into account the concentration of iron in the sulfide ore material. In the preferred embodiment, the chloride concentration will be adjusted so as to yield between about 30g/L and about 50g/L of ferrous chloride at the end of the leaching step 50. It should be appreciated that the concentration of chloride ions and acid in the lixiviant is selected to maximize dissolution of iron in the lixiviant.

0038 Preferably, the concentration of magnesium chloride in the lixiviant should be at least 300 g/L. However, the optimum concentration of magnesium chloride has been determined to be between about 340 g/L and about 420 g/L. The concentration of chloride has been determined to be important in the kinetics of the leaching reactions, and on the vapour pressure of both water and hydrochloric acid above the reaction slurry. Higher total chloride concentrations lead to increased kinetics and lower vapour pressures of water but higher in hydrochloric acid.

0039 During the leaching step 50, the sulfide ore material is contacted and leached with the lixiviant to obtain a solid residue and a leachate containing dissolved iron therein. Following the leaching step 50, a majority of the leachable iron in the source sulfide ore material will be in solution. Preferably, at least 70% of the iron will be dissolved in the leachate and more preferably, the leachate will contain over 90% of the iron. Somewhat lower values may be obtained where the ore material being treated is very pyritic (or refractory).

0040 The leaching step 50 may be conducted in a single reactor, or in a plurality or reactors arranged either in series or in parallel. In the preferred embodiment, the leaching
step 50 is carried out in three or more leaching reactors. These leaching reactors may be pressurized or unpressurized vessels. Preferably, the leaching step 50 is carried out in an unpressurized vessel (i.e. at atmospheric or ambient pressure). The use of an unpressurized vessel tends to be less cost intensive. In contrast to certain known prior art processes which require elevated pressures and temperatures to obtain reaction kinetics sufficiently rapid to enable a viable commercial process, the leaching step 50 tends to achieve satisfactory reaction kinetics at atmospheric pressure.

0041 The leaching step 50 may be conducted as a continuous process or a batch process. If conducted as a continuous process, the leaching may be performed co-currently, countercurrently, or in any other suitable manner known in the art. The leaching step 50 may also be conducted in a pachuca.

0042 The leaching step 50 may carried out at a temperature that lies in the range of about 20°C to about the boiling point of the lixiviant at ambient pressure (which is about 120°C). The filterability of the leach residue has been found to be greatly enhanced at temperatures greater than about 105°C due to the dehydrating effects of the chloride lixiviant. In view of the foregoing, the temperature at which the leaching step 50 may be conducted, is preferably between 105°C and about 110°C.

0043 It should be noted that whilst the pH is usually an important parameter in hydrometallurgical processes, the pH tends not to have an important role in the leaching step 50. It may however be used as an indicator of the reaction progress or as a control mechanism. Similarly, the redox (or oxidation-reduction) potential (Eh) tends not be a significant parameter of the leaching step 50. Preferably, the leaching conditions are maintained at the natural oxidation-reduction potential of the system, which tends to be reducing. However, in contrast to most leach circuits, there is no requirement in the present embodiment to control the redox potential.

0044 In this embodiment, leaching of the sulfide ore material is controlled to permit the dissolution of iron only - no substantial amounts of base metals or value metals are dissolved within the leachate. The base metals and any value metals in the sulfide ore material remain in the solid residue. As will be explained below in greater detail, in this embodiment, the solid residue obtained from the leaching step 50 is an upgraded base metal concentrate upon
which may be further processed to recover the base metals and any value metals contained therein.

0045 Hydrogen sulfide gas is formed during the leaching step 50 as the sulfur in the sulfide ore material is converted to hydrogen sulfide under reducing conditions. The hydrogen sulfide thus formed is removed from the lixiviant at step 70. Preferably, the hydrogen sulfide gas is stripped from the lixiviant in a continuous manner to ensure the concentration of hydrogen sulfide in the lixiviant is kept relatively low. An inert (non-oxidizing) carrier gas (for instance, argon or nitrogen) may be added to the lixiviant solution to aid in the stripping of the hydrogen sulfide. Alternatively, the leaching step 50 may be conducted under a relatively small negative pressure (vacuum) to remove the hydrogen sulfide gas formed.

0046 It is believed that the high proton activity achievable at the low acid concentrations herein permits the process to be operated under conditions that cause the formation of hydrogen sulfide rather than elemental sulfur or sulfate ion, so-called reductive leaching conditions at low redox potential.

0047 In the preferred embodiment, at least one portion of the hydrogen sulfide gas is reacted (at step 80) with sulfur dioxide gas, in a Claus reaction to recover an elemental sulfur product and steam, according to the following chemical reaction:

\[ 2\text{H}_2\text{S} + \text{SO}_2 \rightarrow 3\text{S} + 2\text{H}_2\text{O} \]

0048 The Claus reaction may be carried out in one or more stages, using one or more catalysts. High rates of recovery in the order of 94% to 97% of elemental sulfur may be achieved. It will thus be appreciated that reacting the sulfur dioxide gas in a Claus reaction allows for the recovery of the intrinsic energy in the sulfide ore material. Advantageously, the high-pressure steam produced by this reaction may be used to supply and/or sustain the energy requirements of the processing and refining operations, thereby potentially resulting in substantial savings in operating costs.

0049 The remaining portion of the hydrogen sulfide gas (and preferably, at least one third of the hydrogen sulfide gas formed) is reacted with molten copper (at step 90) at a temperature between about 1200°C and about 1500°C. Thus reacted, the hydrogen sulfide is absorbed rapidly and directly into the molten copper to thereby form cuprous sulfide, \( \text{Cu}_2\text{S} \),
and liberate pure hydrogen gas (approximately greater than 99.5% pure), according to the following chemical equation:

\[ \text{H}_2\text{S} + 2\text{Cu} \rightarrow \text{Cu}_2\text{S} + \text{H}_2 \]

**0050** The hydrogen gas so liberated may be collected for use as a clean energy source. For instance, it could be employed as clean fuel to power the oxy-fuel burners of a pyrohydrolysis reactor used for further refining the upgraded base metal concentrate obtained. Preferably though, most of the hydrogen gas will be used in a fuel cell (i.e. a solid oxide fuel cell) or in a co-generation plant to produce electric power and high-pressure steam. The electric power thus generated could be used to supply some or all of the energy requirements of the plant and associated township. As an illustrative example, a typical sulfide ore containing about 30% sulfur at a treatment rate of 5000 to 10,000 tonnes ore/day would generate at least 10 MW of electric power using existing solid oxide fuel cell technology. Clearly, recovering the useful energy from a sulfide ore in this manner alleviates the requirement for the generation of on-site power and heat energy from the burning of fossil fuels such as coal, oil or natural gas, and furthermore, eliminates the production of the greenhouse gases associated with the burning of these fuels. Moreover, substantial savings in energy costs may be achieved.

**0051** Preferably, the cuprous sulfide formed at step 90 is exposed to oxygen (i.e. "blown") at step 100 as in any typical copper converter to re-form the elemental molten copper and liberate sulfur dioxide gas. Since both of the reactions with the molten copper (steps 90 and 100) are kinetically fast, only a relatively small amount of copper need be employed to perform step 90. As a result, the reactor may be run on a continuous basis. Alternatively, the cuprous sulfide may be sold to a copper smelter if desired and fresh copper may be supplied to carry out step 90.

**0052** A portion of the sulfur dioxide gas formed at step 90 may be used as an oxidant in a lixiviant in an additional leaching step carried out to further refine the upgraded base metal concentrate (as further explained below). The remaining portion may be employed in the Claus reaction carried out at step 80. Alternatively, the sulfur dioxide may be converted to sulfuric acid, or collected as liquid sulfur dioxide for sale and re-use.

**0053** It should be appreciated that while it is contemplated that a portion of the hydrogen sulfide stripped from the lixiviant will be diverted directly toward a Claus reactor for use in step 80 such that less than the entire amount of hydrogen sulfide will be reacted with molten
copper, it should be appreciated that this need not be the case in every application. In alternative applications, all the hydrogen sulfide gas could be used in step 90.

Moreover, while in the preferred embodiment, the hydrogen sulfide stripped from the lixiviant is reacted to molten copper at step 80 to obtain cuprous sulfide and hydrogen gas, it should be appreciated that other uses may be made of the hydrogen sulfide gas. In an alternative embodiment, the hydrogen sulfide may be contacted with a solution of a metal or metalloid to thereby form a sulfide, in particular a solution of a copper salt or one containing arsenic - which sulfide may be used in further base metal refining steps. In still another embodiment, the hydrogen sulfide could be burned directly in air to generate steam and a high-strength stream of sulfur dioxide which may be recovered separately as liquid sulfur dioxide.

A solid residue in the nature of an upgraded base metal concentrate and a leachate containing ferrous chloride and the initial chloride salt (in this case magnesium chloride) is obtained following the leaching step 50. The solid residue and leachate are subjected to a solid/liquid separation step 110, following which the upgraded base metal concentrate is subjected to a series of base metal recovery steps and the leachate is treated for the recovery of an iron product and recycled hydrochloric acid.

In these very high chloride concentration brines, it tends not to be necessary to add a base to precipitate the iron, nor to invoke pyrohydrolysis (which is a high temperature and energy intensive reaction at ~700°C to 950°C) to recover the acid. It has been found that by heating the leachate to a temperature of at least 180°C in the presence of additional moisture/water (step 120) and simultaneously subjecting the remaining leachate to a precipitation (step 130), then at least some of the iron could be precipitated as a hematite product of high purity and the hydrochloric acid could be distilled from the leachate. Where no additional moisture/water is supplied, it may be necessary to heat the leachate to at least about 190°C to precipitate the iron as hematite. Preferably, the leachate will be heated to a temperature between about 190°C and about 250°C, and most preferably, between about 220°C and about 250°C.

In the present embodiment, the precipitation step 130 involves air/oxygen sparging (although, other suitable oxidants may be used) and adding sufficient moisture/water to hydrolyse the iron to thereby promote the formation of a hematite precipitate. The precipitate
thus obtained is a high-purity, easily filterable, crystalline hematite product. The water
required for the reaction may be most conveniently added as steam, the latter also providing
the heat necessary for distilling the hydrochloric acid from the leachate.

0058 The iron may be oxidized by air and hydrolysed by water to form hematite according
to following chemical reactions:

\[
\begin{align*}
4\text{FeCl}_2 + \text{O}_2 + 4\text{H}_2\text{O} & \rightarrow 2\text{Fe}_2\text{O}_3 + 8\text{HCl} \\
2\text{FeCl}_3 + 2\text{H}_2\text{O} & \rightarrow \text{Fe}_3\text{O}_4 + 6\text{HCl}
\end{align*}
\]

0059 Where the leachate contains appreciable amounts of sulfate ion, then after the ferrous
iron has been oxidized, a mixture of hematite and jarosite may be formed with jarosite being
formed according to the following chemical reaction:

\[
5\text{FeCl}_3 + 2\text{Fe}_2(\text{SO}_4)_3 + 21\text{H}_2\text{O} \rightarrow 3(\text{H}_3\text{O})\text{Fe}_3(\text{SO}_4)_2(\text{OH})_6 + 15\text{HCl}
\]

0060 This jarosite reaction also liberates hydrochloric acid, and importantly acts as a
control for any sulfate in the system. Both the hematite and jarosite formed are highly
crystalline in these high chloride systems, and consequently settle and filter very easily,
unlike in corresponding sulfate or dilute chloride systems.

0061 Alternatively, if the chloride used in the lixiviant is magnesium chloride, or if
appreciable quantities of magnesium have been dissolved in the leaching step, then
magnesium sulfate, which has a much lower solubility than magnesium chloride, will
precipitate and will form instead of jarosite.

0062 While in this embodiment, the iron is precipitated out of the remaining leachate as
hematite, this need not be the case in all applications. In alternative embodiments, other iron
product precipitates could be obtained. More specifically, if only a limited quantity of air is
added during the precipitation step, magnetite will be formed according to following
chemical reactions:

\[
6\text{FeCl}_2 + 6\text{H}_2\text{O} + \text{O}_2 \rightarrow 2\text{Fe}_3\text{O}_4 + 12\text{HCl}
\]

0063 Alternatively, if air or oxygen is completely absent during the precipitation step, the
iron will precipitate as ferrous oxide according to the following chemical reaction:
Following the precipitation step 130, the remaining leachate (now an iron-depleted, magnesium chloride liquor) and the hematite product are then subjected to a solid/liquid separation step 140. The hematite product thus recovered may be dried and sold, or simply disposed of. In the preferred embodiment, the magnesium chloride liquor is regenerated and recycled for use in the process 20. As shown in FIG. 1, the recycled magnesium chloride liquor and the recycled hydrochloric acid may be combined to form some of the lixiviant used in leaching step 50. It should be appreciated that the recycled magnesium chloride liquor, the recycled hydrochloric acid and the sulfide ore material may be combined in any particular order prior to being introduced into the reactor(s) in which leaching step 50 is conducted. However, it is preferred that sulfide ore material and the recycled magnesium chloride liquor be combined prior to the recycled hydrochloric acid being added.

Turning now to the upgraded base metal concentrate, following the solid/liquid separation step 110, the upgraded base metal concentrate is subjected to a series of base metal recovery steps that include performing a second leaching step 150. This, however, need not be the case in every application. In an alternative embodiment, the upgraded base metal concentrate may be subjected to flotation to obtain separate base metal concentrates and a precious metal concentrate.

Except as otherwise set forth below, the second leaching step 150 is carried out substantially as set out in United States Patent Application Publication No. 2005/01 18081 of Harris et al., the disclosure of which is hereby incorporated by reference. To facilitate understanding, the second leaching step 50 is briefly described below.

With reference to FIG. 2, during the second leaching step 150 the upgraded base metal concentrate is contacted and leached with a second lixiviant to obtain a solid residue in the nature of a value metal concentrate and a leachate containing predominantly all of the base metals and any iron remaining from the primary leach circuit, in solution. The value metal concentrate and the base metal rich-leachate are subjected to a solid/liquid separation step 160, following which the value metal concentrate is treated to extract one or more value metals therefrom and the base metal rich-leachate is subjected to a series of base metal recovery steps as explained in greater detail below. The solid/liquid separation step 160 may
employ any known technique for effecting such separation including pressure or vacuum filtration, countercurrent decantation or centrifuge.

0068 The second leaching step 150 may be carried out at a temperature that lies in the range of about 20°C to about the boiling point of the lixiviant at ambient pressure, (which is about 120°C). The leaching conditions, in particular, the lixiviant, redox potential (Eh) and pH, may be controlled to dissolve predominantly all of the base metals and remaining iron from the upgraded base metal concentrate. The value metals, such as, the platinum group metals (PGMs), gold and silver are not substantially leached - these remain in the solid residue to be recovered by any means known in the art. For instance, the value metal concentrate may be subjected to a leaching step to dissolve the PGMs, gold and silver. The lixiviant used for this leaching step could be the same as that employed in the first and second leach circuits, except that the redox potential for the system would be increased preferably to greater than about 700 mV and, more preferably, to greater than about 800 mV. Known PGM separation steps could then be performed to recover the PGMs, gold and silver present.

0069 While not preferred, in the event, some of the PGMs gold and silver came to be present in the second leachate, then some or all of the leachate could be subjected to one or more PGM separation steps to recover the value metals. For instance, value metal recovery could be effected by cementation with metallic copper, zinc or organic or inorganic reductants.

0070 In this embodiment, the lixiviant used in the second leaching step 150 includes an acid, an oxidant, and a chloride. Preferably, the lixiviant includes the remaining leachate from the precipitation step 130 (that is, a solution of ferric chloride and magnesium chloride). By recycling the remaining leachate for use in the second leaching step 150, the addition of external reagents may be avoided.

0071 It will however be appreciated that a lixiviant similar or substantially similar to that used in the primary leach (that is, a lixiviant comprising an acid and a chloride, but no oxidant) could also be employed successfully in leaching certain base metals (i.e. nickel and cobalt). In one embodiment, the leachate (containing ferrous chloride and magnesium chloride) obtained following the first leaching step 50, could be employed in the second leaching step 150.
In alternative embodiments, other combinations of acid, chloride and oxidant may be used to constitute the lixiviant. In this regard and as in the primary leach circuit, the acid may be an organic acid, sulfurous acid, sulfuric acid or hydrochloric acid - with hydrochloric acid being the preferred acid. The oxidant may be a mixture of sulfur dioxide and oxygen and/or air. The sulfur dioxide formed during the "blowing" of cuprous sulfide at step 100 could be used as a constituent of the second lixiviant. However, it will be appreciated that other oxidants, taken alone or in combination, may be used to similar advantage. For instance, any of the following compounds could be suitable oxidants for the second lixiviant: alkali metal peroxides, alkaline earth metal peroxides, alkali metal perchlorates, alkaline earth metal perchlorates, ammonium perchlorate, magnesium perchlorate, alkali metal chlorates, alkaline earth metal chlorates, alkali metal hypochlorites, alkaline earth metal hypochlorite, chlorine, hydrogen peroxide and peroxysulfuric acid.

The chloride may be magnesium chloride, calcium chloride, sodium chloride, potassium chloride, lithium chloride, ferrous chloride or mixtures thereof and moreover, may be the same chloride as the one used in the primary leach circuit. The use of magnesium chloride in the second lixiviant is particularly favoured, since it is amenable to a pyrohydrolysis step (if necessary), as further described below.

As in the primary leach circuit, hydrogen sulfide gas may be formed during the leaching step 150 as the remaining sulfur in the upgraded base metal concentrate material is converted to hydrogen sulfide under reducing conditions. The hydrogen sulfide thus formed may be stripped from the lixiviant in a continuous manner. The stripping of hydrogen sulfide may be facilitated by adding an inert (non-oxidizing) carrier gas to the lixiviant solution.

While the formation of hydrogen sulfide in the secondary leach circuit is preferred, in contrast to the process described in United States Patent Application Publication No. 2005/0118081 of Harris et al., this is not strictly required because most of the sulfur in the sulfide ore material was removed as hydrogen sulfide in the primary leaching step 50. The formation of sulfates (most obviously from the use of the preferred oxidant i.e. the mixture of sulfur dioxide and oxygen) in the secondary leach circuit can be tolerated, since the sulfate ion concentration may be easily controlled by the formation of jarosite and/or magnesium sulfate.
0076 Removal of any remaining iron in solution occurs at step 170. Accordingly, where hydrogen sulfide is formed during the leaching step 150, it will be possible to distil the hydrochloric acid and precipitate hematite from the leachate by heating the leachate to a temperature of at least 180°C in the presence of additional moisture/water and simultaneously subjecting the remaining leachate to a precipitation step, as in the primary leach circuit. As mentioned above, where no additional moisture/water is supplied, it may be necessary to heat the leachate to at least about 190°C to precipitate the iron as hematite. Preferably, the leachate will be heated to a temperature between about 190°C and about 250°C, and most preferably, between about 220°C and about 250°C.

0077 Where sulfates are present in the leachate, hydrolytic distillation may be carried out to precipitate jarosite. If there is insufficient iron present for this purpose, then some may be added from the primary leach circuit. The jarosite thus obtained tends to be very pure and may be easily filtered. A solid/liquid separation step 175 is then carried out to separate the hematite or jarosite precipitate from the leachate.

0078 Once the remaining iron has been removed, the base metal-rich leachate may be subjected to a series of base metal recovery steps 180. These steps may be carried out in accordance with any known base metal extraction process including ion exchange, solvent extraction, electrowinning or sulfide precipitation. The use of electrowinning may be particularly advantageous, since power generated from process 20 either in the form high-pressure steam or hydrogen gas may be used. In accordance with the foregoing process high rates of base metal extraction may be obtained. For instance, using this process, recovery rates of greater than 95% have been achieved for nickel and cobalt, and greater than 85% for copper.

0079 Hydrochloric acid may be recovered and recycled after each metal removal step in a manner analogous to that carried out in the primary leach circuit, thereby obviating the need to neutralize the acid with a base. The value metal-depleted leachates may also be treated to regenerate the chloride and acid constituents of the lixiviant.

0080 Furthermore, magnesium oxide (magnesia) and hydrochloric acid may be obtained by subjecting the base metal-depleted leachate to a pyrohydrolysis step. As previously mentioned, the oxy-fuel burners of the pyrohydrolysis reactor could be fuelled by the hydrogen gas generated at step 90. The magnesium oxide thus produced may be used in the
recovery of base metals - more specifically, to effect neutralization and precipitation of certain metal products (e.g. cobalt and nickel oxide). The use of magnesium oxide for neutralization and precipitation is advantageous because the required amount of magnesium oxide may be produced by the system. Moreover, the addition of magnesium oxide does not add any further ions in the leachate, which would otherwise require the use of additional treatment steps.

0081 The principles of the present invention are illustrated by the following examples:

Example 1

0082 A sample of a polysulfide ore, analyzing 45.3% Fe, 1.46% Cu, 0.78% and 32.2% S, was crushed to 100% passing 100 mesh and leached in 360 g/L MgCl₂ solution at 105°C. Concentrated hydrochloric acid was added in increments of 200 kg/tonne and allowed to react for one hour. Finally, 1200 kg/tonne of sodium hypochlorite as oxidant was added with 1000 kg/tonne of hydrochloric acid to the solids residue. The oxidant addition was not optimised. The results achieved are shown in Table 1 below.

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<th>Acid Addition (kg/tonne)</th>
<th>Extraction (%)</th>
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<tr>
<td></td>
<td>Fe</td>
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<tr>
<td>200</td>
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<tr>
<td>400</td>
<td>51.7</td>
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<tr>
<td>600</td>
<td>82.8</td>
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<td>800</td>
<td>92.8</td>
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<td>92.1</td>
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<tr>
<td>1000 + NaOCl</td>
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0083 The results showed no reaction until after the addition of 200 kg/tonne, with the optimum being between 600 and 800 kg/tonne. Predominantly all of the iron was leached and negligible quantities of base metals. The addition of oxidant resulted in significant leaching of the base metals.
Example 2

0084 The same sample of polysulfide ore was leached under identical conditions, in a series of separate tests, except that the acid was added all at once. The results achieved are shown in Table 2 below.

Table 2

<table>
<thead>
<tr>
<th>Acid Addition (kg/tonne)</th>
<th>Time (min.)</th>
<th>Extraction (%)</th>
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<td></td>
<td>Fe</td>
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<td>600</td>
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<td></td>
<td>60</td>
<td>92.8</td>
</tr>
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</table>

0085 The results showed that the reaction was essentially instantaneous, with only a relatively small increase in iron extraction with time. Both iron and sulfur extraction
increased with increasing acid addition. At higher acid addition, the extraction of nickel became significant, whereas copper was harder to leach.

Example 3

0086 The filtrate from the above tests, analyzing 37.5 g/L iron, all as ferrous, was heated up to close to its boiling point with air sparging. The iron was seen to oxidize as the solution colour changed from green to red, and as water was removed as condensate. However, when the volume in the reactor was maintained approximately constant by the addition of water and the temperature maintained at around 180°C, then red solids analyzing 69.4% Fe (99.2% hematite), which were subsequently shown to be entirely hematite by X-Ray diffraction analysis, were formed. The condensate generated was collected. When the off-gas temperature was maintained at greater than 109°C, the hydrochloric acid content of the condensate was in the range 0.5-3.0 g/L. When the temperature of the vapour phase was allowed to increase to greater than 109°C, then the concentration of the hydrochloric acid was as high as 3.0-3.5M.

0087 The results of this test show that in the presence of magnesium chloride, ferrous iron could easily be oxidised without recourse to a pressure vessel (unlike in the process of Kovacs described earlier). Additionally, the test demonstrates that by adding water at a high temperature, hydrolysis to hematite was effected and HCl was recovered. The recovered HCl was, however, relatively dilute under these conditions.

Example 4

0088 500g of pure copper metal was heated up to 1200°C, and hydrogen sulfide gas was passed into it at a rate of 1 L/min. for a period of 1.5 hours. Hydrogen gas of greater than 99.5% purity was obtained, with a conversion rate of greater than or equal 99% for the hydrogen sulfide. Thereafter, air was blown into the cuprous sulfide so formed at a rate of 2 L/min. for a period of 3.5 hours to achieve regeneration of the molten copper.

Example 5

0089 A sample of primary leach residue, analyzing 20.0% Fe, 2.12% Ni, 4.56% Cu and 0.35% Co was leached in a solution of 400 g/L magnesium chloride and sufficient ferric chloride to account for 150% of the stoichiometric requirement for all of the above metals.
The leach was carried out for 6 hours at a temperature of 118°C, after which time, 83% Fe, 99.5% Ni, 99.8% Cu and 68.1% Co extraction was achieved.

0090 This example demonstrates the magnesium chloride/ferric chloride solution generated during the oxidation of the primary leach liquor is a very effective lixiviant for the first stage leach residue, resulting in very high extractions of copper (from chalcopyrite) and nickel (from pentlandite).

**Example 6**

0091 A solution containing 360 g/L magnesium chloride and 43.0 g/L ferric iron was heated under a slight vacuum to its boiling point, and 50 g/L of hematite seed added. Water was pumped slowly into the reactor, and the distillate collected. The experiment was continued until the acidity of the distillate dropped below 5 g/L HCl. The total solids were then collected, washed and dried, and submitted for X-Ray diffraction analysis, which showed the solids to contain only hematite, and chemical analysis showed a purity of greater than 99%. Acid and iron balances showed that 73.5% of the iron in the original solution had been precipitated. The temperature of the solution, always under a slight vacuum, rose to as high as 182°C, and the distillate collected was 1.1 M in HCl. The reaction was continued for a period of 20 hours.

This example shows that it is possible to precipitate and recover soluble iron with a high recovery from high chloride concentration solutions by a simple hydrolytic distillation method, without the need to add any neutralizing agent. The recovered solids are a highly pure form of hematite. The example also demonstrates that hydrochloric acid can also be effectively and simultaneously recovered without recourse to high temperature methods such as spray roasting or pyrohydrolysis. However, the kinetics of the reaction carried out in this manner are slow.

**Example 7**

0092 A solution containing 360 g/L MgCl₂ and 48 g/L Fe (III) was heated up to 155°C. A total of 1 g/L oxalic acid was added. The final iron concentration was 1.6 g/L, representing a recovery of 96.7% of the contained iron, compared to only 73.5% as described in example 6. The X-Ray diffraction analysis of the solids showed only the presence of hematite. When the
experiment was repeated, but using a vertical condenser, the concentration of the distilled hydrochloric acid was 3.7M.

Example 8

0093 A solution containing 360 g/L MgCl₂ and 58 g/L Fe (III) was heated up to a temperature of between 230-240°C and water was added to maintain the volume constant. In 12 hours, over 96% of the iron was precipitated as hematite and over 96% of the HCl was recovered. The maximum HCl concentration attained was 1.6M.

Example 9

0094 The above experiment was repeated except that instead of water, additional feed solution was injected, as might be the case in a continuous circuit, and the temperature allowed to rise to 245°C. In four hours, 45% of the total iron added was precipitated as hematite and the equivalent amount of HCl recovered. At 245°C, the concentration of the recovered HCl was 6.3M, equivalent to the HCl azeotrope.

0095 Examples 8 and 9 demonstrate that as fresh solution is added, iron recovery kinetics are increased and higher strength hydrochloric acid are recovered.

0096 Although the foregoing description and accompanying drawings relate to specific preferred embodiments of the present invention and specific processes for the removal of iron from ore materials as presently contemplated by the inventors, it will be understood that various changes, modifications and adaptations, may be made without departing from the spirit of the invention.
CLAIMS

1. A process for recovering a base metal from a sulfide ore material, the process comprising:

   providing the sulfide material;

   providing a lixiviant comprising an acid and a chloride; the acid being selected from the group consisting of an organic acid, sulfurous acid, sulfuric acid and hydrochloric acid; the chloride being selected from the group consisting of magnesium chloride, calcium chloride, sodium chloride, potassium chloride, ferrous chloride and lithium chloride;

   performing a leaching step on the sulfide ore material at atmospheric pressure in the absence of an oxidant, using the lixiviant to obtain a solid residue and a leachate containing dissolved iron therein; the solid residue being an upgraded base metal concentrate;

   removing hydrogen sulfide gas formed during the leaching step;

   reacting at least a portion of the hydrogen sulfide gas formed with molten copper to form cuprous sulfide and hydrogen gas;

   collecting the hydrogen gas formed for use as an energy source;

   separating the leachate from the solid residue;

   treating the leachate to recover therefrom at least some of the iron as a hematite product of high purity; and

   subjecting the solid residue to a series of base metal recovery steps.

2. The process of claim 1 wherein the sulfide ore material selected contains at least one base metal selected from the group consisting of nickel, copper, zinc and cobalt.

3. The process of claim 2 wherein the sulfide ore material selected contains at least one of gold, silver and a platinum group metal.

4. The process of claim 1 the sulfide ore material selected is chosen from the group consisting of pyrrhotite, pentlandite, chalcopyrite, pyrite, arsenopyrite, galena and sphalerite.
5. The process of claim 1 wherein providing the sulfide ore material includes crushing the sulfide material.

6. The process of claim 1 wherein providing a lixiviant further includes adjusting the concentration of the chloride to obtain a solubility of between about 75% and about 95% of its saturation point.

7. The process of claim 1 wherein providing a lixiviant includes selecting magnesium chloride as the chloride.

8. The process of claim 1 wherein providing a lixiviant includes selecting hydrochloric acid as the acid.

9. The process of claim 8 wherein providing a lixiviant includes selecting magnesium chloride as the chloride.

10. The process of claim 8 wherein providing a lixiviant further includes adjusting the concentration of magnesium chloride therein to at least about 300g/L.

11. The process of claim 10 wherein the concentration of magnesium chloride in the lixiviant is adjusted to between about 340 g/L and 420 g/L.

12. The process of claim 1 wherein the leaching step is performed at a temperature between about 20°C and about the boiling point of the lixiviant.

13. The process of claim 12 wherein the leaching step is performed at a temperature between about 105°C and about 110°C.

14. The process of claim 9 wherein treating the leachate to recover therefrom at least some of the iron as a hematite product of high purity, includes:

   heating the leachate to distill the acid therefrom; and

   simultaneously subjecting the remaining leachate to a precipitation step to promote the formation of a hematite precipitate.

15. The process of claim 14 wherein the leachate is heated to a temperature of between about 190°C and about 250°C.
16. The process of claim 15 wherein the leachate is heated to a temperature of between about 220°C and about 250°C.

17. The process of claim 14 wherein:

   the leachate is heated to a temperature of at least about 180°C; and

   the precipitation step includes one of adding water and adding steam to the leachate.

18. The process of claim 17 wherein the precipitation step includes one of adding water and adding steam to the leachate.

19. The process of claim 14 wherein the precipitation step includes adding water to the leachate.

20. The process of claim 14 wherein the precipitation step includes further adding a catalyst the leachate.

21. The process of claim 20 wherein the catalyst is oxalic acid.

22. The process of claim 14 further comprising separating the hematite product from the leachate.

23. The process of claim 14 further comprising recovering the chloride from the leachate.

24. The process of claim 1 further comprising reacting at least a portion of the hydrogen sulfide gas formed with sulfur dioxide gas in a Claus reaction to obtain elemental sulfur and steam.

25. The process of claim 1 further comprising exposing the cuprous sulfide formed to oxygen to obtain molten copper and sulfur dioxide gas.

26. The process of claim 25 further comprising converting the sulfur dioxide gas to sulfuric acid.

27. The process of claim 25 further comprising reacting the sulfur dioxide gas with hydrogen sulfide gas in a Claus reaction to obtain elemental sulfur and steam.
28. The process of claim 1 wherein:

- the lixiviant is a first lixiviant;
- the leaching step is a first leaching step;
- the leachate is a first leachate;
- the solid residue is a first solid residue; and

the step of subjecting the solid residue to a series of base metal recovery steps, includes:

- providing a second lixiviant comprising an acid and a chloride; the acid being selected from the group consisting of an organic acid, sulfurous acid, sulfuric acid and hydrochloric acid; the chloride being selected from the group consisting of magnesium chloride, calcium chloride, sodium chloride, potassium chloride and lithium chloride;

- performing a second leaching step on the first solid residue at atmospheric pressure, using the second lixiviant to obtain a second solid residue and a second leachate containing at least one base metal dissolved therein; and

- separating the second leachate from the second solid residue; and

- recovering from the second leachate the at least one base metal.

29. The process of claim 28 wherein the second lixiviant further includes an oxidant.

30. The process of claim 28 wherein:

- the base metal containing material contains at least one of gold, silver and a platinum group metal;

- the second solid residue is an upgraded value metal concentrate; and

- the process further includes subjecting the second solid residue to at least one value metal recovery step.

31. The process of claim 30 wherein:
the base metal containing material contains at least one of gold, silver and a platinum group metal;

the second leachate is a value metal-rich leachate; and

the process further includes subjecting the value metal-rich leachate to at least one value metal recovery step.

32. The process of claim 1 wherein:

the lixiviant is a first lixiviant;

the leaching step is a first leaching step;

the leachate is a first leachate;

the solid residue is a first solid residue; and

the step of subjecting the solid residue to a series of base metal recovery steps, includes:

- providing a second lixiviant comprising ferric chloride and magnesium chloride;

- performing a second leaching step on the first solid residue at atmospheric pressure, using the second lixiviant to obtain a second solid residue and a second leachate containing at least one base metal dissolved therein; and

- separating the second leachate from the second solid residue; and

- recovering from the second leachate the at least one base metal.
Figure 2
INTERNATIONAL SEARCH REPORT

A. CLASSIFICATION OF SUBJECT MATTER
IPC: C22B 3/06 (2006.01) , C22B 3/08 (2006.01) , C22B 3/10 (2006.01) , C22B 3/16 (2006.01) , C22B 3/20 (2006.01) , C22B 3/44 (2006.01)

According to International Patent Classification (IPC) or to both national classification and IPC

B. FIELDS SEARCHED
Minimum documentation searched (classification system followed by classification symbols)
USPC: 75, 423, 75/743, 423/140
IPC8: C22B-3/06, 3/08, 3/10, 3/16, 3/44
CPC: 53/287,306,327,371

Documentation searched other than minimum documentation to the extent that such documents are included in the fields searched

Electronic database(s) consulted during the international search (name of database(s) and, where practicable, search terms used)
CAPlus, Scopus, Espacenet, WEST, CPD

C. DOCUMENTS CONSIDERED TO BE RELEVANT

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<th>Category*</th>
<th>Citation of document, with indication, where appropriate, of the relevant passages</th>
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[X ] See patent family annex.

[ ] Further documents are listed in the continuation of Box C.

Date of the actual completion of the international search
1 March 2007 (01-03-2007)

Date of mailing of the international search report
10 April 2007 (10-04-2007)

Name and mailing address of the ISA/CA
Canadian Intellectual Property Office
Place du Portage I, C1 14 - 1st Floor, Box PCT
50 Victoria Street
Gatineau, Quebec K1A 0C9
Facsimile No.: 001-819-953-2476

Authorized officer
Jay Fothergill 819-997-4842

Form PCT/ISA/210 (second sheet) (April 2005)
## INTERNATIONAL SEARCH REPORT
### Information on patent family members

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