INTEGRATED HYDROMETALLURGICAL AND PYROMETALLURGICAL PROCESSING OF BASE-METAL SULPHIDES

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The present invention relates to the recovery of base metals, in particular but not exclusively copper, via integrated hydrometallurgical and pyrometallurgical processing of base-metal sulphides, in particular but not exclusively iron-containing base-metal sulphides.
INTEGRATED HYDROMETALLURGICAL AND PYROMETALLURGICAL PROCESSING OF BASE-METAL SULPHIDES

FIELD OF THE INVENTION

[0001] The present invention relates to the recovery of base metals, in particular but not exclusively copper, via integrated hydrometallurgical and pyrometallurgical processing of base-metal sulphides, in particular but not exclusively iron-containing base-metal sulphides.

[0002] The present invention relates more particularly but not exclusively to the recovery of copper from iron-containing copper sulphide concentrates in which the copper is present as chalcopyrite and/or bornite, by first processing the initial iron-bearing copper sulphide concentrate feedstock into three separate fractions by means of froth flotation or other beneficiation processes, the fractions being a high-grade concentrate fraction, a low-grade concentrate fraction, and tailings fraction. Typically the high- and low-grade concentrates will have copper contents greater than about 25%, and less that about 25%, respectively. The three separate fractions thereby recovered are individually and collectively treated in an integrated hydrometallurgical and pyrometallurgical flowsheet to recover copper in the metallic state or another suitable form in such a manner that the overall copper recovery is greater than that which would be obtained by direct smelting or roasting of the single iron-bearing copper sulphide concentrate.

[0003] The present invention relates more particularly but not exclusively to the recovery of copper from iron-containing copper sulphide concentrates whereby the high-grade copper concentrate recovered by froth flotation or beneficiation of the initial iron-bearing copper sulphide concentrate feedstock is further upgraded to yield a product containing about 50% copper or higher by reaction with a copper sulphate solution.

[0004] The present invention relates more particularly but not exclusively to the recovery of copper from iron-containing copper sulphide concentrates whereby the sulphate solution used to further upgrade the high-grade concentrate recovered by froth flotation or beneficiation is produced by leaching the low-grade copper concentrate recovered from the froth flotation circuit.

[0005] The present invention also relates more particularly but not exclusively to the combined recovery of copper from iron-containing copper sulphide concentrates and from pyrometallurgical smelter slags, dusts and/or fumes by means of an integrated hydrometallurgical and pyrometallurgical process.

[0006] The present invention also relates more particularly but not exclusively to the separation of any uranium or other potential byproducts such as cobalt, nickel, and rare earths present in the iron-containing copper sulphide concentrate feedstock or in the flotation tailings.

[0007] The present invention also relates more particularly but not exclusively to the recovery of precious metals, in particular gold and silver, from iron-containing copper sulphide concentrates by means of an integrated hydrometallurgical and pyrometallurgical process.

[0008] The present invention also relates more particularly but not exclusively to the rejection of environmentally stable arsenic-containing compounds from iron-containing copper sulphide concentrates by means of an integrated hydrometallurgical and pyrometallurgical process.

BACKGROUND OF THE INVENTION

[0009] The present invention also relates more particularly but not exclusively to additional upgrade of high grade concentrate after reaction with copper sulphate solution by means of froth flotation, beneficiation or further treatment of a portion of the upgraded concentrate to make copper sulphate solution.

[0010] In this specification, where a document, act or item of knowledge is referred to or discussed, this reference or discussion is not an admission that the document, act or item of knowledge or any combination thereof was, at the priority date:

[0011] (i) part of common general knowledge; or

[0012] (ii) known to be relevant to attempt to solve any problem with which this specification is concerned.

[0013] Copper can be recovered from a range of copper-containing ores and concentrates by both hydrometallurgical and pyrometallurgical routes. Where practical the copper-containing ore of mine ore is concentrated by means of various physical beneficiation techniques, especially froth flotation. Some copper ore treatment plants sacrifice overall recovery in order to maximise the copper grade of the flotation concentrate.

[0014] The dominant copper-containing mineral in most copper sulphide deposits is chalcopyrite. Minor copper-containing minerals include chalcocite, bornite, covellite and enargite. Pyrite and to a lesser extent pyrhotite are typically present as gangue sulphide minerals.

[0015] High-grade copper sulphide-containing concentrates (typically greater than about 25% Cu) are commonly treated by pyrometallurgical routes, whereas hydrometallurgical routes are generally applied to lower-grade copper concentrates. The economically and technically most favourable processing route is also influenced by the presence and concentration of minor metals such as zinc and lead, valuable metals such as silver, gold and platinum, as well as deleterious metals such as arsenic, present in the feed material. Hydrometallurgical processing routes are generally characterised as being complex with many unit steps and large circulating loads of copper.

[0016] The three dominant pyrometallurgical routes for high-grade copper sulphide concentrates are smelting to a matte followed by converting to blister copper, direct to blister smelting, and roasting. The efficiency of the smelting technology is determined, amongst other things, the Cu/S ratio and the concentration of slag forming components, especially iron, magnesium and silica. Conventional smelting processes are generally not applicable to lower grade copper concentrates. Not all of the copper content of the original feed is recovered as blister copper, the remaining copper reporting to the slag and to the smelter dusts or fumes recovered from the smelter off-gases.

[0017] Roasting involves conversion of the copper content to a water-soluble form, which is recovered from the roasted calcine by leaching, solvent extraction and electrowinning. Roasting is often inefficient because copper-containing insoluble ferrite phases can form during the roasting stage.

[0018] Many hydrometallurgical processes have been proposed for treating copper-containing concentrates, especially chalcopyrite-containing concentrates. This is a reflection of the so-called refractory nature of chalcopyrite. The more significant of these processes include:
(a) Oxidative leaching under acidic conditions at elevated temperatures and pressures;

(b) Oxidative leaching using bacteria;

(c) Oxidative leaching with ferric sulphate at ambient pressure;

(d) Oxidative leaching under alkaline (ammoniacal) conditions at elevated temperatures and pressures; and

(e) Chloride or chlorine leaching.

Few of the proposed processes have attained full-scale commercial development for a number of reasons, including the need for ultrafine grinding, extended retention times, problems with the generation and neutralisation/precipitation of high ferric iron liquors, difficulties in recovery of any precious metals in the original feedstock, and the generation of relatively dilute copper solutions. All of these factors increase both the capital and operating costs, especially when applied to relatively low-grade copper (chalcopyrite) concentrates. The overall copper recoveries from many of the proposed processes is often less than economically acceptable because of incomplete reaction and/or losses to leach residues via precipitation and/or sorption processes.

The recovery of copper from chalcopyrite-containing copper concentrate that also contains an appreciable uranium content is substantially more complex as it is necessary to effect an efficient copper-uranium separation ahead of the recovery of electrowon/electrorefined copper. Under most hydrometallurgical processing conditions it is difficult to achieve selective leaching of copper over uranium or uranium over copper, so that most hydrometallurgical circuits yield a pregnant copper-uranium solution. Separation of the soluble copper from the uranium requires the installation and operation of a complex solvent extraction circuit that results in increased capital and operating costs. Processing of the copper-uranium ore produced at the Olympic Dam project is a typical example of such a complex flowsheet.

As an alternative to direct hydrometallurgical treatment of a copper concentrate, various means of chemically upgrading the copper content of the feed have been proposed. The upgraded copper-containing intermediate product may be then further processed by known pyrometallurgical and hydrometallurgical technologies. One such approach involves the so-called metathesis process in which the chalcopyrite component of the concentrate is reacted with a copper sulphate solution to produce low-iron copper sulphide (e.g. digenite) and an acidic ferrous sulphate solution.

A similar reaction also occurs for any bornite present in the copper concentrate.

Similar reactions occur representing the recovery of arsenic and iron from emargite, as well as cobalt and nickel from a variety of minerals that may coexist with the predominating copper minerals, for example cobaltite and carrollite, etc.

One or both of these reactions (1) and (2) constitute one aspect of U.S. Pat. Nos. 2,568,963, 2,662,009, 2,744,172 and 4,024,218, Canadian Patent No. 1 258,181, and WIPO Patent Publication No WO 2004/106561. All of these patents propose to forward the upgraded copper sulphide concentrate, which typically contains about 50% Cu, to either a smelter or further processed by other means. The claimed flowsheets are deficient in a number of aspects in that, for example, they do not disclose how the copper sulphate leachant is generated, involve additional flotation steps, require the addition of a reductant to facilitate the process metallurgy, or achieve complete conversion (metathesis) to ensure that the resultant discarded acidic ferrous sulphate solution has an acceptably low copper content.

The present invention overcomes these deficiencies and is capable of maximising overall copper recovery more particularly but not exclusively from low-grade concentrates and tailings, or smelter dusts, flames or slags, and together where appropriate an efficient copper-uranium separation from the concentrate, while at the same time minimising the capital and operating cost components of the total processing flowsheet.

The objects and advantages of the invention described in the present specification are to be read disjunctively with the object of at least providing the public with a useful alternative.

SUMMARY OF THE INVENTION

In one aspect, the present invention provides a method for the recovery from a metal value bearing concentrate of one or more metal values including at least a primary metal value, the method using an integrated hydrometallurgical and pyrometallurgical process and including the steps of:

- performing a separation grading step by subjecting the metal-bearing concentrate to one or more beneficiation steps to enable the formation of at least three separate metal-bearing fractions including a high-grade metal concentrate fraction, a low-grade metal concentrate fraction, and a tailing fraction;

- subjecting the low-grade metal concentrate fraction to an acidic leaching process to enable formation of an acidic upgrading solution containing a soluble form of the primary metal value;

- subjecting the high-grade metal concentrate fraction to a metal value upgrading step via reaction with at least a portion of the acidic upgrading solution formed in step (ii) to produce an upgraded metal concentrate fraction having a higher w/w concentration of the primary metal value; and

- subjecting the upgraded metal concentrate fraction derived from step (iii) to a smelting process to obtain at least the primary metal value.

The primary metal value present in the metal value-bearing concentrate may include various metals including copper, zinc or nickel. Preferably, the primary metal value present in the metal value-bearing concentrate is copper. Preferably, the acidic upgrading solution in step (ii) is an acidic copper sulphate solution.

The one or more beneficiation steps of step (i) can include milling, gravity, magnetic and/or flotation processes.

Preferably, the tailing fraction obtained in step (i) can be subjected to an acidic leaching process to form an acidic solution for use in the processes of step (ii) or step (iii), and/or to further hydrometallurgical treatment to recover any additional soluble metal values that may be present in the tailing fraction.

In a preferred embodiment, the smelter flue dust or slag derived from the smelting process of step (iv) is further subjected to hydrometallurgical treatment with an acidic solution to form an underflow stream containing insoluble comp-
ponents and an overflow stream containing soluble forms of the primary metal value, wherein the underflow stream is recycled for use in step (iii) or step (iv) and/or the overflow stream is recycled for processing with the low-grade metal concentrate fraction or tailing fraction for facilitating recovery of any contained metal value. Preferably, the further step of treating the smelter flue dust or slag is a one or two stage process, whereby in the second stage the underflow product formed is further treated, prior to use in step (iii) or step (iv), by reaction with a stronger acid solution to facilitate further removal of any ferric iron or other leachable impurities that may be present. Preferably, the underflow stream is also treated through regrinding and/or flotation processes to provide a concentrate feed for use in step (iii) or step (iv) for combination with the high-grade metal concentrate fraction.

[0041] Preferably, the acidic leaching process of step (ii) includes a pressure oxidation circuit wherein an oxidative leach reaction is continued until substantially all of the primary metal value present in the low-grade metal concentrate fraction is dissolved in the acidic upgrading solution and the ferric ion concentration of the acidic upgrading solution is below about 10 g/L. More preferably, any leach residues formed in the pressure oxidation circuit are further processed or recycled with the tailing fraction.

[0042] The metal value-bearing concentrate can also include a precious metal value, typically gold and silver, wherein any leach residues formed in any of the hydrometallurgical and pyrometallurgical steps are further processed for recovering the precious metal value. More preferably, the further process for recovering the precious metal value is via cyanide leaching.

[0043] Preferably, the metal value-bearing concentrate is an ore slurry containing at least one of the minerals selected from the group consisting of covellite, chalcocite, chalcopyrite, bornite and enargite.

[0044] In one preferred embodiment, the high-grade metal concentrate fraction includes greater than 20-35% w/w of the primary metal value and the low-grade metal concentrate fraction includes less than 20-35% w/w of the primary metal value.

[0045] The separated high-grade metal concentrate fraction and/or low-grade metal concentrate fraction from step (i) can also be subjected to further processing including grinding, flotation, filtering, dilution, thickening, washing and/or other cleaning processes.

[0046] Preferably, prior to step (iii) or step (iv), the difference between the concentration of the primary metal value in the high-grade metal concentrate fraction compared to that in the low-grade metal concentrate fraction is increased through further processing to facilitate the recovery of the primary metal value.

[0047] In one embodiment of the invention, the hydrometallurgical upgrading step (iii) includes the use of a single or multi-compartmented autoclave or digester to produce an upgraded metal concentrate fraction having an increased primary metal value concentration.

[0048] A source of sulphur dioxide can also be obtained from off-gases of the smelting process of step (iv) and used in the hydrometallurgical upgrading step (iii) to facilitate reduction of any ferric ions that may be present.

[0049] In another preferred embodiment, the primary metal value concentration in the upgraded metal concentrate fraction is increased to a concentration of greater than about 45-50% w/w.

[0050] Prior to the smelting process step (iv), the upgraded metal concentrate fraction obtained in step (iii) can be further processed by at least one of washing, flotation, dewatering, or drying and mixing with fluxes, and/or subjected to further separation grading to form a further upgraded feed for the smelting process step (iv) and a secondary recycle feed for step (iii).

[0051] In another embodiment of the invention, secondary or precious metal values such as uranium, cobalt, nickel, gold and silver, are recovered, typically during the smelting process, when present in the metal value bearing concentrate or any one of the tailings fraction, low grade metal concentrate fraction or high grade metal concentrate fraction.

[0052] Furthermore, any arsenic present in the metal value bearing concentrate can be transformed into an environmentally stable form acceptable for further storage.

[0053] For convenience, the following summary of the invention as well as the detailed description of the invention is specifically directed to the recovery of copper from chalcopyrite, bornite and/or enargite containing concentrates through the application of the integrated hydrometallurgical and pyrometallurgical process. Those skilled in the art will appreciate that the same processing principles can be applied to other base- and precious-metal containing feedstocks and as such the inventors include within the scope of the present invention the application of such feedstocks.

[0054] Those skilled in the art will also appreciate that efficient operation of the integrated hydrometallurgical and pyrometallurgical flowsheet depends on strict control of water, mass and heat balances. As such, where process streams are split ahead of forwarding to both downstream and upstream unit stages, the proportions of such splits or fractions is determined by meeting these balance requirements and will be determined, in part, by the chemical, physical and mineralogical properties of each process stream.

[0055] Furthermore, in one particular aspect the present invention provides a novel flowsheet first involving treatment of one slurry, such as typical run of mine ore slurry, after any necessary and appropriate comminuting stage or stages, to a conventional froth flotation step to form three streams or fractions: a high-grade copper concentrate fraction (typically greater than about 25% Cu), a low-grade copper concentrate fraction (typically less than about 25% Cu), and a tailings fraction. Each of these streams or fractions is then subjected to further processing by means of the integrated hydrometallurgical and pyrometallurgical flowsheet.

[0056] If appropriate, the separate high- and low-grade copper concentrates produced from the original copper-containing feedstock can be reground and refloated, filtered, washed or cleaned by other known methods prior to treatment by the integrated hydrometallurgical and pyrometallurgical processing flowsheet if the mineral separation is improved by a finer size distribution or deleterious water-soluble chemical species, eg., halides, can be conveniently separated from the concentrate.

[0057] The mass and copper split between the high- and low-grade copper concentrate fractions can be adjusted to optimise recovery of copper for upgrading and smelting, and restrict the deportment of iron minerals, particularly pyrite and pyrrhotite, to the high-grade stream.

[0058] In one embodiment, the high-grade concentrate fraction can be reacted with a copper rich, low ferric ion solution in a single or multi-compartmented agitated autoclave or unagitated digester to produce an upgraded copper
concentrate, typically containing above about 45-50% Cu, according to reactions (1) and (2). The upgraded copper concentrate can be washed, dewatered, dried and mixed with fluxes before smelting by known methods, to produce blister copper and copper-containing smelter slag and smelter dust by-products. The copper rich, low ferric ion solution can be obtained by oxidative leaching of, for example, the low-grade concentrate, or optionally supplemented with overflow from the smelter dust leach circuit.

The low-grade concentrate slurry can be leached in a single or multi-compartmented agitated autoclave in the presence of sparged oxygen or air, ferric ions and sulphuric acid or optionally supplemented with overflow from the smelter dust leach circuit. The reaction conditions can be adjusted for temperature, copper and ferric ions in solution. The slurry pulp density can be adjusted to suit the reactivity and size distribution of the low-grade concentrate, and to control the thermal and water balance and/or free acid concentration across the leaching processes. Preferably, the oxidative leach reaction is continued in one or more compartments with or without interstage thickening until essentially all of the copper minerals contained in the low-grade concentrate are dissolved into the pregnant leach solution while the ferric ion concentration is controlled to below about 10 g/L. The product from the low-grade concentrate leach circuit can then be treated by suitable means, such as a conventional or pressure thickener, for solid/liquid separation purposes. The thickener underflow or overflow can be flashed cooled to release heat from the circuit. The copper-rich liquor overflow can then be forwarded as the soluble copper source for the high-grade concentrate leach or transferred under pressure to preserve the heat content for downstream processing. In this preferred embodiment, the copper-depleted slurry (underflow) is then forwarded to the tailings circuit.

The flotation tailings can be thickened to manage the water balance in the milling, flotation and tailings leach circuits, with the underflow forwarded optionally to a leach circuit where it is reacted with a ferric rich solution derived from the low-grade concentrate leach circuit supplemented with additional sulphuric acid and optionally, with the overflow from a separate smelter dust leach circuit, in order to promote the leaching of the residual copper and any impurity metals contained in the flotation tailings.

The tailings leach temperature and retention time can be adjusted to suit the reactivity of the copper and impurity metals contained in the flotation tailings. In this regard, flash steam from the oxidative leach can optionally be employed to provide additional heat to the tailings leach. The slurry exiting the flotation tailings leach circuit can then be subjected to solid/liquid separation by suitable means, such as a conventional CCD thickener. The leach residue, as the CCD underflow, can be disposed of in a suitable residue storage facility. The pregnant leach solution, as the CCD overflow, is processed by known methods for copper recovery or further purified by separation and recovery of uranium or other metals as by-products.

The optional leaching of the smelter dusts can be operated using flexible operating conditions in order to maximise impurity removal while at the same time assisting with the overall water balance across the high-grade and low-grade concentrate leach circuits. The smelter dust leach circuit can be carried out in a single stage, or alternatively in two separate stages. In the latter configuration, the first stage involves reaction with a relatively low acid leachant to dissolve the easily soluble copper component. This is recovered by means of a suitable solid/separation stage, such as a conventional CCD thickener. The pregnant solution, as the CCD overflow can then be forwarded to the low-grade concentrate leach circuit. The first stage leach residue, as the CCD underflow, is forwarded to the second leach stage where it is reacted with a stronger acid solution to increase ferric iron removal from the smelter dust residue as well as dissolve a greater proportion of the impurities in the original smelter dust. In this preferred embodiment, after solid/liquid separation by suitable means, such as a conventional CCD thickener, the CCD overflow is forwarded to the flotation tailings leach circuit, with the residue, as the CCD underflow, which contains any insoluble (refractory) copper, is transferred to the high-grade concentrate upgrading step.

The optional treatment of the smelter slag can also involve a combination of regrinding and flotation stages to recover a concentrate rich in copper that can be combined with the high-grade concentrate fraction or upgrading feed. This additional copper-containing feed assists with the reduction of any ferric iron present in the high-grade concentrate upgrading circuit. Alternatively, a side stream of sulphur dioxide from the smelter OFF-gases can be injected into the high-grade concentrate upgrading circuit to promote the ferric ion reduction reactions.

An advantage of the present invention is that the overall recovery of copper from the original copper-containing feedstock is maximised. An additional advantage is that variable ore blends, concentrate mixtures, or diverse sources of copper ore may be economically treated without the need to dispose of any untreated component of the original feedstock.

Another advantage of the present integrated hydrometallurgical and pyrometallurgical process flowsheet is the ready transfer of solution products and energy from circuit to circuit so that the chemical efficiency and hence economics of the whole integrated process is improved.

The integrated hydrometallurgical and pyrometallurgical process flowsheet also facilitates the optimum recovery of by-products including uranium, cobalt, nickel and precious metals, particularly gold and/or silver, contained in the original feedstock.

The integrated hydrometallurgical and pyrometallurgical process flowsheet also facilitates the transformation of any arsenic contained in the original feedstock into a form that is environmentally benign and can be safely discharged for disposal in a conventional tailings/residue storage facility.

BRIEF DESCRIPTION OF THE DRAWINGS

The invention is now described by way of example, with reference to the accompanying drawings, in which:

FIG. 1 shows an overall flowsheet of one embodiment of the present invention covering an integrated hydrometallurgical and pyrometallurgical process for the recovery of a primary metal value from a base metal concentrate; and

FIG. 2 shows an overall flowsheet of another embodiment of the present invention covering an integrated hydrometallurgical and pyrometallurgical process for the recovery of a primary metal value from a base metal concentrate.

DETAILED DESCRIPTION OF THE INVENTION

The following detailed description of a preferred embodiment of the present invention refers to the overall process flowsheet outlined in FIG. 1. This particular preferred embodiment incorporates the optional treatment of the smelter dusts as well as the smelter slag in order to maximise the total recovery of the copper content of the original run-of-mine ore.
A high-grade concentrate [31], a low-grade concentrate [21] and tailings [11]. Typically the high-grade and low-grade concentrates will have copper contents of greater than about 25% and less than about 25%, respectively. The high-grade and low-grade concentrates can be reground and further treated by known flotation methods to enhance the copper split between the concentrate streams and/or influence the extent of reaction in the downstream processes. The concentrate streams may also be washed and dewatered to remove soluble impurities, such as halides, before further treatment. All of the copper from the run-of-mine slurry exiting the grinding circuit [10] reports to one of the three product streams derived from the flotation circuit [100] and will report for treatment in the overall integrated hydrometallurgical and pyrometallurgical flowsheet. By this means, the recovery of copper from the initial run-of-mine ore is enhanced.

Typically the high-grade concentrate [31] will contain the bulk of the simple copper sulphide minerals (covellite and chalcopyrite) and well as the bulk of the iron-containing copper sulphide minerals (chalcocite and bornite). Typically the low-grade concentrate [21] will contain most of the remaining copper sulphide and iron-copper sulphide minerals with a minor portion of the iron sulphide minerals (pyrite and pyrrhotite). Typically the tailings [11] will contain the remaining iron sulphide minerals, gangue minerals (ground rock) and oxide minerals of impurity metals. Other copper and value metal minerals may be present in concentrates [21] and [31].

The high-grade concentrate [31] is dewatered and forwarded to an appropriate repulp tank [131] using copper sulphate solution [26] after which any uranium leached can be recovered via thickener/filter [131A] where it is partially dewatered and/or washed in order to maintain the water balance around the high-grade circuit. The copper sulphate solution [26] overflows from the thickener treating the slurry exiting the primary oxidative leach reactor (autoclave) first compartment [122A] flash cooling thickening circuit [123] to the repulp tank [131]. The repulped high-grade concentrate [32A] is typically at the ambient pressure boiling point and is forwarded to a medium temperature reactor [132], typically operating at 110-190°C and more typically at 140-180°C, to which is added the overflow liquor [28] from the thickener [124] treating the slurry exiting the primary oxidative leach reactor (autoclave) last compartment [122B] and a portion of the smelter dust leach residue [40] exiting the smelter dust leach thickener [136]. The overflow liquor temperature from the thickener is typically 200-210°C or about 100°C from atmospheric thickening.

The so-called metathesis reactions (1) and (2) proceed within the upgrading reactor (autoclave or digestor) [132]. The reaction rate will vary with the size distribution, temperature and mineralogy of the high-grade concentrate [31] and the process conditions. Copper in solution replaces iron in the iron-containing copper sulphide minerals by displacement to raise the copper content of the upgraded concentrate [33], typically to above about 50%.

The upgraded concentrate [33] and the remaining portion of the smelter dust leach residue [40] are subjected to washing, dewatering and drying steps [133] using known processes, or may be further treated as shown in FIG. 2 by flotation or other known separation methods to split the upgraded concentrate into streams of different grades for oxidative leaching [33A] and smelting [33B]. The hot upgraded concentrate dewatering liquor [37A, 37B] is recycled to the low-grade concentrate and tailings circuits. The dried upgraded concentrate [34] is combined with suitable fluxes [41] and led to a conventional copper smelter [134]. The smelter output (blister copper) is forwarded to a conventional copper anode casting/electrorefining circuit. Any precious metals (gold and silver) initially present in the high-grade concentrate is recovered with the upgraded concentrate and can be recovered from the anode slimes produced in the copper electrorefining circuit. The smelter off-gases are captured, cleaned and converted to sulphuric acid in a contact acid plant. Electrolyte bleed from the refinery can alternatively supplement the smelter dust leach solution, repulp low-grade or high-grade concentrate, or other circuit balance purposes depending on the water, copper or acid flows.

The dust [35] recovered from the smelter off-gases is slurried with a portion of the acidic raffinate [18C] exiting the solvent extraction circuit [115] used to recover soluble copper from the tailings [11] treatment circuit, or another suitable barren liquor or bleed electrolyte, together with a suitable amount of sulphuric acid [38] derived from the acid contact plant or from some other source. The dust leach [135] can be carried out in one stage, or in two or more counter current stages to conserve acid consumption and control undesirable impurity dissolution.

The leached smelter dust [36] is partially dewatered in a conventional thickener [136]. Dust leach thickener overflow [42] carries soluble impurities to the low grade leach or tailing leach process [112] depending on the copper balance. Dust leach thickener underflow [40] is returned upstream to the upgrader reactor (autoclave or digestor) [132] and/or the upgraded concentrate washing/filtering/drying circuit [133] and forms part of the feed [34] for the flash smelting circuit [134]. By this means the refractory copper content of the smelter dust will ultimately be recovered during the smelting/ refining steps.

The low-grade concentrate [21] produced in the run-of-mine ore beneficiation circuit [100] is dewatered in a conventional thickener or other suitable means to maintain the water balance through the low-grade leach circuit. It is then repulped [121] with primary oxidation reactor (autoclave) first compartment [122A] flash cooled thickener underflow slurry [27] and/or upgraded concentrated dewatering liquor [37A] and/or downgraded concentrate [33A] to form the primary oxidation feed [22]. The primary oxidation feed slurry [22] is oxidized in a compartmented agitated autoclave [122] into which oxygen and/or oxygen-enriched air [24] is injected. The first compartment of the primary oxidation autoclave [122A] is sized to accommodate about 80-95% of the oxidation reactions and is typically operated at about 200-220°C with and oxygen partial pressure of about 600 kPa. The operating temperature of the first compartment of the primary oxidation autoclave [122A] is maintained by means of a flash cooling/thickener recycle cooling circuit [123]. The primary oxidation first compartment partly discharges [23A] to the flash cooling thickener step [123] and permits thermal stability while operating the oxidative autoclave at high-concentrate slurry densities. The remaining partial oxidized slurry from the first compartment [238] flows to the downstream compartments of the primary oxidation autoclave [122B]. The ferrous iron in [37A] is oxidised in the autoclave ultimately to basic ferric sulphate but not without participating in
the oxidation of the low-grade concentrate. The return of [37A] to [121] ensures that any un-precipitated copper from the liquors in [132] is lock-cycled within the combined oxidation and upgrade autoclave circuits. Only small quantities of soluble copper exit the concentrate oxidation and upgrade circuits in liquor streams [25] and [37B].

[0080] The primary oxidation discharge slurry [23] exiting the last compartment of the primary oxidation autoclave [122B] is separated by suitable means such as a pressure thickener or flashed to atmospheric pressure prior to conventional thickening and [124]. The overflow stream [28] consists of an acidic copper sulphate solution and is forwarded to the upgrading autoclave [132] as the copper-rich solution or leachant (reagent) for the metathesis reactions that take place in the upgrading autoclave [132].

[0081] The primary oxidation product thickener underflow [25] is forwarded to the ferric leach circuit [125] for further dissolution of iron. The ferric iron content of the thickener underflow [25] is leached with the required portion [37B] exiting the upgrading thickener [133] overflow and the required portion [183] of the acidic raffinate exiting the solvent extraction circuit [115] used to recover soluble copper from the last stage of the tailings retreatment circuit. The acidic ferric solution [29] generated in the ferric leach circuit [125] is forwarded to the tailings leach stage [112] where it is used to dissolve the secondary copper sulphide-containing phases present in the flotation tailings.

[0082] The flotation tailings [11] derived from the run-of-mine ore beneficiation circuit [100] are thickened [111] in order to maintain the water balance in the milling and flotation circuits [100]. The thickener overflow [16] returns to the upstream circuits [100] or to storage (not shown) before reuse. The thickener underflow [12] is leached at atmospheric pressure in a suitable agitated reactor [112] with sulphuric acid [17], derived from the contact acid plant used to treat the smelter off-gases, and supplemented by the ferric rich liquor [29] derived from the ferric leach circuit [125] that treats the primary oxidation thickener underflow [25]. Any copper-containing phases together with uranium and other soluble impurity minerals present in the tailings underflow will be dissolved by the acidic ferric leachant. Direct steam or flash steam from the autoclave can be injected into the tailings leach tanks to raise the slurry temperature to increase the rate of reaction and maximise the extent of leaching the copper and uranium contents.

[0083] The leached slurry [13] is subjected to CCD and washing stages [113] using recycled raffinate [18A], with the CCD underflow [14A] being discharged as final residue to a suitable tailings impoundment. The pregnant CCD overflow [14] is clarified by known means (not shown) before the soluble copper is recovered by known solvent extraction and electrowinning stages [114]. The barren liquor [15] is further processed [118] to recover uranium, impurity and/or by-product metals by solvent extraction, precipitation or other known methods. The barren raffinate solution [18] is distributed to wash and leach applications [18A, 18B, 18C] in the three process circuits, or may be stored for future use.

EXAMPLE 1

[0084] A copper-iron sulphide concentrate containing about 34.5% copper and 1.9% iron and consisting predominantly of chalcocite and having a size range of about 80% passing 75 micron was subjected to pressure oxidation at 225°C. using a pulp density of 20% solids (w/w). More than 99% of contained copper was leached after 120 minutes.

[0085] A second portion of the same concentrate was ground to 80% passing 23 micron and was reacted at a pulp density of 40% solids (w/w) at 180°C. with a copper sulphate solution containing 90 g/L of copper derived from the above pressure oxidation test. No reagents or chemicals were added to the upgrading step. Copper in solution dropped to less than 1 g/L after 15 minutes, and the copper content of the concentrate upgraded to above 45% w/w while the iron content was lowered to about 14% w/w.

EXAMPLE 2

[0086] The same copper-iron sulphide concentrate described in Example 1 was subjected to pressure oxidation at 210°C. and a pulp density of 25% solids (w/w) for a period of 180 minutes. Copper extraction was in excess of 97%.

[0087] Another portion of the same concentrate was reacted at a pulp density of 21.5% solids (w/w) at 180°C. with a copper sulphate solution containing 82 g/L of copper derived from the above pressure oxidation step. No reagents or chemicals were added to the upgrading stage. Copper in solution dropped to less than 6 g/L after 180 minutes, and the copper content of the concentrate upgraded to above 55% w/w while the iron content was lowered to below 9% w/w.

[0088] In the preceding description of the invention and in the claims which follow, except where the context requires otherwise due to express language or necessary implications, the words “comprise” or variations such as “comprises” or “comprising” are used in an inclusive sense, i.e., specify the presence of the stated features, but not to preclude the presence or addition of further features in various embodiments of the invention.

[0089] It is to be understood that this invention and the preferred embodiments are not limited to the particular materials described, as these may vary. It is also to be understood that the terminology used herein is for the purpose of describing particular embodiments only, and is not intended to limit the scope of the present invention in any way.

[0090] It is also to be noted that, as used herein, the singular forms of “a”, “an” and “the” include the plural unless the context clearly requires otherwise. Unless defined otherwise, all technical and scientific terms herein have the same meanings as commonly understood by one of ordinary skill in the art to which the invention belongs.

1. A method for the recovery from a metal value bearing concentrate of one or more metal values including at least a primary metal value, the method using an integrated hydro-metallurgical and pyrometallurgical process and including the steps of:
   i. performing a separation grinding step by subjecting the metal-bearing concentrate to one or more beneficiation steps to enable the formation of at least three separate metal-bearing fractions including a high-grade metal concentrate fraction, a low-grade metal concentrate fraction, and a tailing fraction;
   ii. subjecting the low-grade metal concentrate fraction to an acidic leaching process to enable formation of an acidic upgrading solution containing a solubilised form of the primary metal value;
   iii. subjecting the high-grade metal concentrate fraction to a metal value upgrading step via reaction with at least a portion of the acidic upgrading solution formed in step
(ii) to produce an upgraded metal concentrate fraction having a higher w/w concentration of the primary metal value; and

iv. subjecting the upgraded metal concentrate fraction derived from step (iii) to a smelting process to obtain at least the primary metal value.

2. The method according to claim 1, wherein the primary metal value present in the metal value-bearing concentrate is copper.

3. The method according to claim 2, wherein in step (ii) the acidic upgrading solution is an acidic copper sulphate solution.

4. The method according to claim 1, wherein the one or more beneficiating steps of step (i) include milling, gravity, magnetic and/or flotation processes.

5. The method according to claim 1, further including the step of subjecting the tailing fraction obtained in step (i) to an acidic leaching process to form an acidic solution for use in the processes of step (ii) or step (iii).

6. The method according to claim 5, further including the step of subjecting the tailing fraction to further hydrometallurgical treatment to recover any additional soluble metal values that may be present.

7. The method according to claim 1, further including the step of subjecting smelter flue dust or slag derived from the smelting process of step (iv) to hydrometallurgical treatment with an acid solution to form an underflow stream containing insoluble components and an overflow stream containing soluble forms of the primary metal value, wherein the underflow stream is recycled for use in step (iii) or step (iv) and/or the overflow stream is recycled for processing with the low-grade metal concentrate fraction or tailing fraction for facilitating recovery of any contained metal value.

8. The method according to claim 7, wherein the step of treating the smelter flue dust or slag is a one or two stage process, whereby in the second stage the underflow product formed is further treated, prior to use in step (iii) or step (iv), by reaction with a stronger acid solution to facilitate further removal of any ferric iron or other leachable impurities that may be present.

9. The method according to claim 7, wherein the underflow stream is further treated through regrinding and/or flotation processes to provide a concentrate feed for use in step (iii) or step (iv) for combination with the high-grade metal concentrate fraction.

10. The method according to claim 1, wherein the acidic leaching process of step (ii) includes a pressure oxidation circuit where an oxidative leach reaction is continued until substantially all of the primary metal value present in the low-grade metal concentrate fraction is dissolved in the acidic upgrading solution and the ferric ion concentration of the acidic upgrading solution is below about 10 g/L.

11. The method of claim 10, wherein any leach residues formed in the pressure oxidation circuit are further processed or recycled with the tailing fraction.

12. The method according to claim 1, wherein the metal value-bearing concentrate includes a precious metal value, and wherein any leach residues formed in any of the hydrometallurgical and pyrometallurgical steps are further processed for recovering the precious metal value.

13. The method according to claim 12, wherein the further process for recovering the precious metal value is via cyanide leaching.

14. The method according to claim 1, wherein the metal value-bearing concentrate includes an ore slurry containing at least one of the minerals selected from the group consisting of covellite, chalcocite, chalcopyrite, bornite and enargite.

15. The method according to claim 1, wherein the high-grade metal concentrate fraction includes greater than 20-35% w/w of the primary metal value and the low-grade metal concentrate fraction includes less than 20-35% w/w of the primary metal value.

16. The method according to claim 1, wherein step (i) includes subjecting the separated high-grade metal concentrate fraction and/or low-grade metal concentrate fraction to further processing including grinding, flotation, filtering, dilution, thickening, washing and/or other cleaning processes.

17. The method according to claim 1, wherein prior to step (iii) or step (iv), the difference between the concentration of the primary metal value in the high-grade metal concentrate fraction compared to that in the low-grade metal concentrate fraction is increased through further processing to facilitate the recovery of the primary metal value.

18. The method according to claim 1, wherein the hydrometallurgical upgrading step (iii) comprises the use of a single or multi-compartmented autoclave or digestor to produce an upgraded metal concentrate fraction having an increased primary metal value concentration.

19. The method according to claim 1, wherein a source of sulphur dioxide obtained from off-gases of the smelting process of step (iv) is used in the hydrometallurgical upgrading step (iii) to facilitate reduction of any ferric ions that may be present.

20. The method according to claim 19, wherein the primary metal value concentration in the upgraded metal concentrate fraction is increased to a concentration greater than above about 45-50% w/w.

21. The method according to claim 1, wherein prior to the smelting process step (iv), the upgraded metal concentrate fraction obtained in step (iii) is further processed by at least one of washing, flotation, dewatering, or drying and mixing with fluxes.

22. The method according to claim 1, wherein prior to the smelting process step (iv), the upgraded metal concentrate fraction obtained in step (iii) is subjected to further separation grading to form a further upgraded feed for the smelting process step (iv) and a secondary recycle feed for step (iii).

23. The method according to claim 1, further including the recovery of the metal values of uranium, cobalt, nickel, gold and/or silver, when present in the metal value bearing concentrate.

24. The method according to claim 1, further including the transformation of any arsenic, when present in the metal value bearing concentrate, into an environmentally safe form.