Title: PROCESS FOR EXTRACTION OF NICKEL, COBALT AND OTHER METALS FROM LATERITE ORES

Abstract: The process of extraction of nickel, cobalt and other metals from laterite ores relates to hydrometallurgy with heap leaching and ion-exchange processing. The process includes preparation of leach solutions, that contain acid belonging to the group including hydrochloric acid, sulphuric acid, nitrogen acid and acids formed by bacterial activity, leaching of original laterite by means of prepared leach solutions with obtaining of pregnant leach solutions and processing of pregnant leach solutions. The process is characterized by the fact that the original laterite contains the mined laterite, leaching of the mined laterite and is made in the way of a continuous multistage countercurrent heap leaching. In this it is previously exposed to crushing, the crushed ore is granulated; granulated ore is exposed to roasting. Then at least one sequence of heaps is formed from the roasted granulated ore. Every heap is exposed to at least two stages of leaching. In every sequence of heaps at the first stage of its leaching the leaching solution is added to every heap.

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PROCESS FOR EXTRACTION OF NICKEL, COBALT AND OTHER METALS FROM LATERITE ORES

FIELD OF THE INVENTION

The present invention relates to hydrometallurgy, in particular, to hydrometallurgical extraction of nickel, cobalt and other metals from laterite ores. The present invention comprises a process for extraction of nickel, cobalt and other metals from laterite ores employing technologies of ore leaching with leach solutions based on the acid which belongs to the group comprising hydrochloric acid, sulfuric acid, nitrogen acid and acids formed by bacterial activity, to produce pregnant leach solutions; and pregnant solutions technologies involving ion-exchange processes.

It is known that oxidized ores, for example, those called laterite ores containing limonite and saprolit, are world largest potential sources of nickel (Ni) and cobalt (Co).

These ores cannot be concentrated by means of magnetic separation and froth flotation in contrast with ferronickel sulfide ores that may be easily concentrated till high enough level of nickel content using conventional processes, such as froth flotation and matte smelting. Unsuitability for beneficiation of the conventional processes leads to inefficiency of the laterite nickel ore mining.

PRIOR STATE OF ART

One of processes of nickel and cobalt extraction from laterite is a well-known Moa Bay process including acid leaching at elevated temperatures and pressures, when iron oxide and aluminum oxisulfate are not soluble to a large extent (Reznik I.D., Yermakov G.P., Shneyerson Ya.M. Nickel. M.: 000 "Science and Technologies", 2001.Volume 2: Oxidized Nickel Ores). Brief description of the Moa Bay process is given in section "State of the Prior Art " of description to patent US 4097575 (C01G53/10; published on 27.06.1978). The Moa Bay process stipulates for transforming of laterite ore with grinding coarseness minus 20 mesh (95 % of substance passes through a sieve 325 mesh as per American Standard) into pulp slurry with solid substance content approximately 45%, whereupon nickel and cobalt with sufficient sulfuric acid ($H_2SO_4$) at elevated temperature and pressure (e.g., 230 °C to 250 °C and 405 lb/in$^2$ to 580 lb/in$^2$ (from 28.5 kg/cm$^2$ to 40.8 kg/cm$^2$) are selectively leached for dissolving about 95 % of both nickel and cobalt within 60 - 90 min. After pressure relieve the leached pulp slurry is washed with countercurrent decantation and then delivered to dumps. Afterwards leach solution with rather low pH level (e.g., within the range of 0 - 0.5) is neutralized with coral slime till pH value of 2.4 in a series comprising four tanks with total time of holding about 20 minutes,
whereupon after separation of solid and liquid fractions a liquid product so treated (containing approximately 5.65 g/l of nickel, 0.8 g/l of iron and 2.3 g/l of aluminum) is precipitated by sulfides. Leach liquid is heated in autoclave at the temperature of about 120 °C (250 °F) and pressure 150 lb/inch² (10.55 kg/cm²), whereupon sulfide precipitation with the help of hydrogen sulfide (H₂S) acting as a precipitating agent is conducted. According to the initial flowchart of mixed sulfides treatment the sulfide residue is washed and settled to the solid phase content of about 65 per cent. Then it is oxidized in autoclave at the temperature of about 177 °C (350 °F) and pressure 700 lb/inch² (50 kg/cm²). Next the nickel and cobalt containing solution is neutralized with ammonia (NH₃) to pH value of 5.35 that is enough for the complete precipitation of residue iron, aluminum (Al) and chrome (Cr), where air acts as an oxidizing agent. Afterwards the sediment is separated and nickel- cobalt solution is corrected to pH value of approximately 1.5. To perform selective precipitation of all containing impurities of copper (Cu), lead (Pb) and zinc (Zn) ¾ S is added. The sediment is separated from the solution by means of filtration, whereupon nickel is extracted by different methods, one of which is processing of nickel-containing solution by hydrogen (¾) at elevated temperature and pressure with deriving of nickel powder. Compared to limonite some laterite ores, in particular, saprolit usually have high content of magnesium (Mg) and relatively low content of iron, which shall be removed for purposes of efficient extraction of nickel from leach solution and its efficient separation from iron, magnesium and other impurities. Typical laterite with high content of magnesium and iron usually contains, minimum, 5 and even over 10 % of magnesium by weight.

The Moa Bay process is not suitable for processing of such ores, since due to the high content of magnesium in ore (in the form of magnesium oxide (MgO)) the level of sulfuric acid consumption is unacceptably high. Common industrial practice stipulates for melting high-quality saprolit ores with over 2 % nickel content to produce either ferronickel or nickel matte.

As for limonite, here nickel is extracted from ore by means of high-pressure leaching with sulfuric acid as leaching agent and/or by means of reduction burning with subsequent ammonia leaching. Until quite recently acid leaching of saprolit ore has been hardly employed in industry, due to the absence of a simple and cost-efficient process of nickel extraction from each solution. Nickel and cobalt were extracted from limonite ore mainly by means of leaching under high pressure with sulfuric acid as leaching agent and/or by means of reduction burning with subsequent ammonia leaching. However, the process of High Pressure Acid Leaching (HPAL) and ammonia-carbonate leaching performed under atmosphere pressure have a number of disadvantages:

- instrumentation complexity leading to big capital expenditures;
- high power consumption level;
- sophisticated technology;
- condensability of initial ore and autoclave slurry that is an important factor in assessing technology capital capacity.

A process of extraction nickel, cobalt and other metals from laterite ores explained in the description to patent of the Russian Federation for invention No. 2418873 (C22B23/00; C22B3/24; published on 20.05.2011) is well known. The disadvantage of this process is that copper, iron and aluminum are extracted at one and the same stage of the ion-exchange extraction. This leads to unnecessary load onto the ion-exchange resin by aluminum and iron. At the same stage pregnant leach solution is neutralized. At the next stage of ion-exchange extraction nickel and cobalt are extracted simultaneously. The disadvantage of this method is that some additional technologies shall be used for nickel and cobalt separation. Together with this the method does not stipulate for integration of pregnant leach solution treatment with leaching processes, e.g., through recirculation of some solutions within the leaching-and-treatment cycle in general. No provision is made for ore roasting, thus limiting the degree of metals transfer into the pregnant solution.

A process of extraction of nickel, cobalt and other metals from laterite ores explained in the description to patent of the Russian Federation for invention No. 2393250 (C22B23/00; C22B3/06; published on 27.06.2010) is well known. According to this process ore is crushed and separated into two fractions - fine and course. The fractions are granulated separately using acid solutions. Leaching, for example, in heaps, shall be performed as two process streams. In this, pregnant solution produced by fine ore leaching within one leach process stream is delivered to leach solution preparation for course ore leaching within another process stream. At that the ore to acid solution weigh ratio shall be 1:3. The pregnant solution is neutralized particularly by means of introduction of the processed ore. Metals are extracted from pregnant solution through its treatment by sorption processes. Some part of magnesium and iron-bearing raffinate from the stage of nickel and cobalt sorption is delivered for leach solution preparation, the other part - for utilization. The disadvantage of this process is that it does not stipulate for copper extraction and utilization and does not unwind the stages of extraction of magnesium and iron from raffinate of nickel and cobalt loading. Together with this no provision is made for ore roasting, thus limiting the degree of metals transfer into the pregnant solution.

According to the description to patent of the Russian Federation for invention No. 2149910 (C22B23/10; published on 27.05.2000) there is a process of extraction of nickel and other metals from laterite orse including ore leaching under atmosphere pressure and at regular
temperatures. Laterite ores with high content of magnesium (for example, saprolit), such as Ni-Fe-Mg-ores containing no less than ~ 5% of magnesium, -10% of iron and 0.5% of nickel by weight are leached in heaps, tanks or through intermixing of ore and mineral acids, e.g., with hydrochloric acid (HCl), sulfuric acid and nitric acid (HNO3). After nickel dissolving pH value of the leach solution is brought to approximately 1 - 3 using oxides of magnesium and iron, generated during the process or fresh ore. After separation of solid phase from leach solution the solution is treated by ion-exchange resin, particularly, resin produced by Dow Chemical, called XFS-4195, due to which nickel is selectively absorbed, whereupon the residue products are nickel-depleted solution (raffinate) or hutch water returned into the leaching system for recycling. When hydrochloric or nitric acid is used as a leaching agent, nickel chloride or nickel nitrate is produced with further concentration though ion-exchange process. Nickel chloride or nickel nitrate solution is treated by means of pyrohydrolysis to produce nickel oxide and secondary acid, e.g., hydrochloric acid or nitric acid. The secondary acid may be used for preparation of leach solution and desorbing solution for the stage of nickel elution with nickel-loaded ion-exchange resin. Pyrohydrolysis allows separating magnesium and iron oxide, which may be re-used for correction of pH value of leach solution to the level of 1 - 3 for nickel extraction by means of ion-exchange. Pyrohydrolysis also provides for reduction of MgO only as a by-product or as a product used for neutralization, when pH level of raffinate reaches the values of 6 or 7 for precipitation and separation of iron and other impurities. Magnesium chloride solution (MgCl2) derived after filtration is bittern for pyrohydrolysis. Nickel oxide derived after pyrohydrolysis may be used for metallic nickel production, and also nickel oxide in combination with iron oxide may be used for ferronickel production. The disadvantage of this process is existence of neutralization stage when the pregnant leach solution is neutralized by magnesium and iron oxides accompanied by sediment settling, partial removal of acid, magnesium losses. There is no provision for copper and cobalt extraction. There is no provision for ore roasting, since it limits the degree of metals transfer from ore into the pregnant solution. The method contained in the description to RF patent No.2149910 does not allow solving the problem of integrated processing of oxidized nickel ore using reasonable labor, materials and energy input.

The closest analogue of the proposed invention is the process of extraction of nickel and other metals from laterite ores given in the description to patent of the Russian Federation for invention No.2355793 (C22B23/00; C22B3/08; published on 20.05.2009). This process includes initial laterite leaching using prepared leach solutions and deriving pregnant solutions and treatment of pregnant solutions. Here continuous multi-stage countercurrent heap leaching of
mined ore is employed. Provision is made for breaking (crushing) of initial mined ore and its separation by fractions, palletizing (granulation). The disadvantage of the method is that it does not explain thoroughly the stages of processing the pregnant leach solution. It is specified that the pregnant solution may be processed with certain well-known technologies, including ion-exchange ones. Consequently, there is no provision for optimization of the production process in general with reference to the parameters of pregnant solution production and the parameters of pregnant solution processing. There is no provision for ore roasting, thus limiting strength of granulated ore particles and degree of metals transfer into the pregnant solution. Not provision is made for leached heap flushing and their further use.

DISCLOSURE OF THE INVENTION

The objective of the present invention is creation of such process of extraction of nickel, cobalt and other metals from laterite ores using leaching technologies under atmosphere pressure and regular temperature and technologies of pregnant leach solutions processing employing ion-exchange processes and resins that will allow for the best extraction of nickel and cobalt from low-grade (poor) ore and/or from small deposits with low costs and power consumption involving high degree of materials regeneration and used reagents recycling. In general, it was essential to develop more competitive method of extraction of nickel and cobalt from laterite ores requiring less capital and operational expenditures compared to well-known established process streams. Besides, this method shall provide for copper extraction, acid-saving, and simplification of pregnant solution treatment compared to other known analogues.

The objective has been set to produce high-purity nickel product, in this purification of nickel from impurities shall be an integral part of pregnant solutions processing, however, not an additional (optional) operation. The method shall also reduce hazardous impact on the environment. In specific cases the process shall ensure that magnesium and iron content in circulating solutions does not exceed the permissible, provide for removal of excessive magnesium and iron from reusable solutions. In some cases additional objectives were targeted at integration of heap multi-stage leaching of mined ore and in-situ leaching within one method and optimization of the parameters of such integration. Together with this the proposed process offers development of the arsenal of hydrometallurgical technologies used for nickel, cobalt and other metals extraction and of the range of constituent technological processes.

The objectives in the general case of the proposed invention are achieved as follows.

The process of extraction of nickel, cobalt and other metals from laterite ores deals with preparation of leach solutions containing acid belonging to the group of hydrochloric, sulfuric, nitric and other acids formed by bacterial activity; leaching of initial laterite by means of
prepared leach solutions with generation of pregnant solutions. The process is characterized by the fact that the initial laterite contains mined laterite, leaching of the mined laterite is performed as continuous multistage countercurrent heap leaching. At that mined laterite is pre-crushed. Then crushed ore is granulated and roasted. Whereupon roasted granulated ore is used for forming, at least, one heap sequence. Every heap sequence undergoes leaching at least in two stages. In every heap sequence leach solution prepared from bittern derived at the second stage of previous heap leaching shall be fed to every heap at the first stage of leaching. At every intermediate leaching stage leach solution, prepared from bittern derived at the next stage of leaching of the previous heap, is fed. At the last leaching stage the original leach solution intended for heap leaching is delivered. After the last stage of leaching the heap is washed. Bittern produced at the first stage of leaching is delivered for processing as pregnant solution or as a constituent of pregnant solution. Continuous multistage countercurrent heap leaching shall incorporate such number of stages that will allow reduction of acid content in bitterns produced at first stages of heap leaching to the level required for the first stage of processing of the pregnant leach solutions or such bitterns are subject to neutralization to the specified acidity level through their filtration in a heap containing another type of rock prior to their delivery for further processing. Pregnant solutions are processed on a stage-by-stage basis using ion-exchange resins and ion-exchange processes arranged in several stages, with production at each stage of eluate enriched with certain metals corresponding to a certain stage and metal-depleted compared to the content of the metals in pregnant solutions and to the eluate derived at other stages. At least, one eluate derived at the adjusted processing stage is purified, and purified eluate is obtained, in this forming of such eluate is performed with increasing content of the respective ion-exchange resin through feeding to it of some part of the purified eluate. At least, some part of solutions generated in the course of processing, in which acidity and soluted impurities are preserved and nickel concentration is equal to or less than 0.3 g/l is delivered to leached heap flushing and then for preparation of leach solutions. At least, some part of solutions generated in the course of processing shall be neutralized deriving insoluble compounds of iron and other metals, whereupon these insoluble compounds are separated, at least, from some neutralized solutions by means of filtration in a leached and flushed heap or in a heap containing another type of rock.

In all implementations of the proposed invention its common features with the closest analogue are as follows:

- the process of extraction of nickel, cobalt and other metals from laterite ores including preparation of leach solutions containing acid belonging to the group of hydrochloric, sulfuric,
niterie and ether acids formed by bacterial activity, leaching of initial laterite by prepared leach solutions with production of pregnant solutions and processing of pregnant solutions is characterized by the fact that initial laterite contains mined laterite;

- leaching of the mined laterite is performed as continuous multistage countercurrent heap leaching;

  - at that mined laterite is pre-crushed;
  - then crushed ore is granulated;
  - whereupon granulated ore is used for forming, at least, one heap sequence;
  - every heap undergoes leaching at least in two stages;

- in every heap sequence leach solution prepared from bittern produced at the second stage of previous heap leaching shall be fed to every heap at the first stage of its leaching;

- at every intermediate leaching stage leach solution prepared from bittern produced at the next stage of the previous leaching is fed;

- at the last leaching stage original leach solution intended for heap leaching is fed;

- after the last stage of leaching the heap is flushed;

  - bittern produced at the first stage of leaching is delivered for processing as pregnant solution or as part of pregnant solution;

  - pregnant solutions are processed on a stage-by-stage basis using ion-exchange resins and ion-exchange processes arranged in several stages with production of eluate enriched with certain metals appropriate to a certain stage and metal-depleted as compared to the content of the metals in pregnant solutions and to the eluates derived at other processing stages.

In all implementations of the proposed invention its distinctive features from the closest analogue are as follows:

- the granules are previously exposed to roasting before forming from it a sequence of heaps;

- then the continuous multistage countercurrent heap leaching with such number of stages, which provides for the decreasing of acidity of bitterns gained at the first stages of the heap leaching up to the level, necessary for the first stage of processing of pregnant leach solutions is performed, or these bitterns are additionally exposed to neutralization up to the mentioned required level of acidity by means of their filtering on the waste heap before passing it for processing;

- at least one eluate, gained at the stage of processing corresponding to it, is purified from impurities, and the purified eluate is gained;

  - production of this eluate is made with additional loading of corresponding ion-exchange resin by means of adding to it a part of obtained purified eluate;
- at least several solutions, formed during processing, in which acidity and impurities are saved in a soluted form, and the concentration of nickel is not more than 0.3 g/l, are passed to flush of the leached heap and then for preparation of leach solutions;
- at least several solutions, formed during processing, are neutralized obtaining the insoluble iron and other metals compounds, and these compounds are separated from at least a part of these neutralized solutions by means of filtering on the leached and flushed heap or on the waste heap.

In the improved version of implementation the specified objective has been additionally achieved and the proposed invention is additionally characterized in that:
- heap building up height is from 1 to 12 m;
- heap leaching is performed, at least, within one process stream;
  - bitterns generated at different stages of heap leaching are collected in separate reservoirs;
  - original leach solutions for heap leaching are prepared with acid concentration of 0.2 - 2 mole/l and fed to the surface of the heap being at the last stage of leaching with spraying concentration of 10 l/h/m² - 20 l/h/m²;
- bittern produced at every stage of heap leaching (except the first one) shall be fed for spraying the surface of another heap that is at the previous stage of leaching (spraying concentration 10 l/h/m² - 20 l/h/m²);
- heap leaching is in such a quantity of stages and streams that at least in one stream the pH value of bittern produced directly at the first stage of leaching or after additional neutralization in a heap containing another type of rock is from 1 to 3;
- delivery of bittern with pH from 1 to 3 produced at the first stage of heap leaching process stream directly in the course of or after additional neutralization in a heap containing another type of rock, as a pregnant solution or part of pregnant solution, or blend of bitterns with pH from 1 to 3 produced at first stages of heap leaching in several process streams directly in the course of or after additional neutralization in a heap containing another type of rock;
- at least, the stages of extraction of copper, nickel and cobalt are performed as the stages of pregnant solution processing;
- the stage of copper extraction by ion-exchange is supposed to be the first stage of pregnant solution treatment, in this a resin capable of selective extraction of copper from solution with pH value from 1 to 3 is used;
- at the first stage of copper extraction copper-bearing resin and raffinate from the copper loading process are produced;
- at the second stage of copper extraction copper eluate is produced from copper-bearing resin by means of copper elution;
- copper eluate is fed for production of copper-bearing items;
- raffinate from the copper loading process is delivered to the second stage of pregnant solution treatment where nickel extraction by the method of ion-exchange is carried out;
- in this, nickel eluate is produced as eluate to be purified with production of purified eluate, where production shall be accompanied by adding appropriate resin through delivery onto it of the purified eluate;
- nickel eluate shall be extracted in two stages: at the first stage of extraction nickel eluate with impurities is produced, at the second stage of extraction nickel eluate is purified;
- at the first phase of the first stage of extraction nickel containing resin with impurities and raffinate from the nickel loading process are produced;
- at the second phase of the first stage of extraction more nickel is added to the resin, which is partially purified by means of nickel eluate delivery with further production of additionally loaded resin and additionally loaded raffinate with impurities;
- at the third phase of the first stage of extraction nickel eluate is produced from additionally loaded resin by means of elution with nickel-containing solution;
- at the fourth phase of the first stage of nickel extraction secondary eluate is produced from resin by means of elution by acid solution;
- secondary nickel eluate is fed for preparation of nickel-bearing solution designed for the third phase of the second stage of nickel extraction;
- raffinate of additional loading is fed as a solution with preserved acidity and soluted impurities for leached heap flushing and leach solution preparation;
- nickel eluate is fed to the second stage of nickel extraction;
- raffinate from the nickel loading process is divided into several parts, in this some part of raffinate from the nickel loading process is delivered for leached heap flushing and further for leach solutions preparation, and the other part of raffinate from the nickel loading process is neutralized with production of non-soluble compounds of iron and other metals; at that, at least, certain part of the other part of raffinate from the nickel loading process is neutralized using, at least, one of the materials belonging to the group including magnesium oxide, brucite, magnesite and later goes to the cobalt extraction stage;
- at the first phase of the second stage of nickel extraction nickel eluate shall be purified by means of iron sorption deriving a product in the form of a resin containing iron and other impurities and purified nickel eluate as raffinate from the iron loading process;
- some part of purified nickel eluate is fed to the second phase of the first stage of nickel extraction, some part is fed for preparation of nickel-bearing solution intended for the third phase of the first stage of nickel extraction, and the other part - for production of nickel-bearing items;
- iron-bearing resin is washed to remove nickel and then fed to the stage of iron elution with acid solution with production of ferric eluate and washed resin;
- ferrous eluate is delivered as a solution with preserved acidity and soluted impurities for leached heap flushing and then for leach solutions preparation;
- at the first stage of cobalt extraction cobalt-bearing resin and raffinate from the cobalt loading process are produced;
- cobalt-bearing resin is fed to the stage of cobalt sorption with acid solution to produce cobalt eluate and free-of-cobalt resin;
- cobalt eluate is fed for production of cobalt-bearing items;
- at least, some part of raffinate from the cobalt loading process is delivered to the flushed leached heap or to the heap containing another type of rock in the form of a neutralized solution with further production of insoluble compounds of iron and other metals with separation by means of filtration in a flushed leached heap or in a heap containing another type of rock with production of magnesium solution;
- solution derived as a result of leached heap flush is fed for leach solutions preparation.

It is supposed that:
- the term "ore crushing" means any ore breakage used in industry;
- the term "granulation" covers such terms as "granulation", "lumping", "pelletizing", "tabletting" and etc. well-known within the industry;
- the term "solution" may be used for pulp slurry designation;
- the term "process stream" generally means process or production line or chain, etc.;
- ponds, basin etc. may be used inter alia for designation of reservoirs for bittern collection;
- for the sake of brevity some clipped terms may be used in such cases, when a clipped term is clear to the specialists, for example: "acid" instead of "acid solution"; "solution" instead of "aqueous solution"; "leached heap" instead of "leached ore heap"; "nickel extraction stage" instead of "pregnant solution processing stage for nickel extraction" etc.

The term "extraction" may, depending on the context, mean "extraction from ore into solution", "extraction from solution into resin", "extraction from resin into eluate".

The term "ferric process type ore" may be used for limonite.

This "magnesium process type ore" may be used for saprolit.
With Fegard to the first stage of multi-stage countereurrent heap leaching (for example, on attached diagrams) the term of "neutralization stage" may be used, as far as at this stage the most significant reduction of bittern acidity is reached.

For the sake of brevity the text of the present description and attached figures may contain such abbreviations as L (laterites, i.e. ferric process type ores), LS (leach solution), PS (pregnant solution), HL (heap leaching), ISL (in-situ leaching), L/S (liquid volume to solid phase mass ratio).

It is supposed that in the framework of the proposed method heap sequence may be formed not only as a separate heap but also as a composite heap, parts of which are to be leached one-by-one in the same order, in which sequence heaps are leached.

In the 1-st particular case the proposed method differs additionally from the improved implementation. The difference lies in the fact that mined ore of the ferric process type, i.e. limonite, is used as the whole of the mined laterite.

In the first refinement to the 1-st particular case an additional difference of the proposed method lies in the fact that original leach solutions intended for heap leaching are prepared with acid concentration of ~ 0.5 mole/l with solution to ore ratio L/S from 1.5 m³/t to 3 m³/t.

In the second refinement to the 1-st particular case an additional difference of the proposed method lies in the fact that mined ore belonging to ferric process type is crushed to the class of 5 mm.

In the third refinement to the 1-st particular case an additional difference of the proposed method lies in the fact that granulation of the crushed mined ore belonging to ferric process type is carried out to formation of the granules of 10 - 20 mm with addition of, at least, 0.1 - 2 % of sodium salt, 0.5 - 5.5 % elemental sulfur, 5 - 15 % of water and process water of the ore mass.

In the fourth refinement to the 1-st particular case an additional difference of the proposed method lies in the fact that firing of granulated crushed mined ore belonging to ferric process type shall be performed at the temperature of 300 - 700 °C and with delivery of sharp water vapor superheated to the temperature of 100 - 200 °C.

In the 2-nd particular case an additional difference of the proposed method from an improved implementation lies in the fact that mined ore belonging to magnesium process type, i.e. saprolit, is used as the whole of the mined laterite.

In the first refinement to the 2-nd particular case an additional difference of the proposed method lies in the fact that original leach solutions intended for heap leaching are prepared with acid concentration of ~ 0.5 mole/l with solution to ore ratio L/S from 1.5 m³/t to 3 m³/t.

In the second refinement to the 2-nd particular case an additional difference of the
The proposed method lies in the fact that mined ore belonging to magnesium process type is crushed to the class of 10 mm.

In the third refinement to the 2-nd particular case an additional difference of the proposed method lies in the fact that granulation of crushed mined ore belonging to magnesium process type is carried out to formation of granules of 20 - 40 mm with addition of, at least, 5 - 15% of water and process water of the ore mass.

In the fourth refinement to the 2-nd particular case an additional difference of the proposed method lies in the fact that firing of granulated crushed mined ore belonging to magnesium process type is carried out at the temperature of 200 - 500 °C.

In the 3-rd particular case an additional difference of the proposed method from an improved implementation lies in the fact that mined ore in the form of mixture containing mined ores belonging to ferric and magnesium process types, i.e. limonite and saprolit, is used as the whole of the mined laterite.

In the first refinement to the 3-rd particular case an additional difference of the proposed method lies in the fact that original leach solutions intended for heap leaching are prepared with acid concentration of ~ 0.5 mole/l with solution to ore ratio L/S from 1.5 m³/t to 3 m³/t.

In the second refinement to the 3-rd particular case an additional difference, of the proposed method lies in the fact that mined ore is crushed to the class of 5 mm.

In the third refinement to the 3-rd particular case an additional difference of the proposed method lies in the fact that granulation of crushed mined ore is carried out to formation of granules of 10 - 20 mm with addition of, at least, 0.1 - 2 % of sodium salt, 0.5 - 5.5 % elemental sulfur, 5 - 15 % of water and process water of the ore mass.

In the fourth refinement to the 3-rd particular case an additional difference of the proposed method lies in the fact that firing of granulated crushed mined ore is carried out at the temperature of 300 - 700 °C and with delivery of sharp water vapor superheated to the temperature of 100 - 200 °C.

In the 4-th particular case an additional difference of the proposed method from an improved implementation lies in the fact that mined ore belonging to ferric and magnesium process types divided by types is used as the whole of the mined laterite. In this the difference lies in the fact that heap leaching is carried out, at least, in two process streams, when at least in one of these process streams the mined ore belonging to ferric type, i.e. limonite, is leached and, at least, in one other of these process streams the mined ore belonging to magnesium process type, i.e. saprolit, is leached.

In the first refinement to the 4-th particular case an additional difference of the proposed
method lies in the fact that original leach solutions intended for heap leaching are prepared with acid concentration of ~ 0.5 mole/l with solution to ore ratio L/S from 1.5 m³/t to 3 m³/t.

In the second refinement to the 4-th particular case an additional difference of the proposed method lies in the fact that mined ore belonging to the ferric process type is crushed to the class of 5 mm.

In the third refinement to the 4-th particular case an additional difference of the proposed method lies in the fact that granulation of mined ore belonging to the ferric process type is performed till formation of granules of 10 - 20 mm with addition of, at least, 0.1 - 2 % of sodium salt, 0.5 - 5.5 % elemental sulfur, 5 - 15 % of water and process water of the ore mass.

In the fourth refinement to the 4-th particular case an additional difference of the proposed method lies in the fact that firing of granulated mined ore belonging to the ferric process type is performed at the temperature of 300 - 700 °C with delivery of sharp water vapor superheated to the temperature of 100 - 200 °C.

In the fifth refinement to the 4-th particular case an additional difference of the proposed method lies in the fact that mined ore belonging to the magnesium process type is crushed to the class of 10 mm.

In the sixth refinement to the 4-th particular case an additional difference of the proposed method lies in the fact that granulation of crushed mined ore belonging to magnesium process type is performed to formation of granules of 20 - 40 mm with addition of, at least, 5 - 15 %, of water and process water of the ore mass.

In the seventh refinement to the 4-th particular case an additional difference of the proposed method lies in the fact that firing of granulated crushed mined ore belonging to magnesium process type is carried out at the temperature of 200 - 500 °C.

In the context of developed implementation an additional difference of the proposed method from the improved implementation lies in the fact that initial laterite particularly contains ore at the in-situ belonging to the magnesium process type, i.e. saprolit. The leaching of the ore in-situ is performed in the form of underground in-situ leaching with production of in-situ leach bitterns, whereupon in-situ leach bitterns are delivered for leach solutions preparation. At that the said in-situ leach bitterns are used for preparation of leach solutions assigned for continuous multi-stage countercurrent heap leaching. And part of solutions produced in the course of processing to be delivered for leach solutions preparation is used for leach solutions preparation assigned for in-situ leaching. At that leach solutions intended for in-situ leaching are prepared with acid concentration of, minimum, 0.5 mole/l.

In the refinement to the developed implementation an additional difference of the
proposed method lies in the fact that original leach solutions intended for heap leaching are prepared with acid concentration of preferably ~ 0.5 mole/l and solution to ore ratio L/S from 1.5 m³/t to 3 m³/t. At that leach solutions designed for in-situ leaching are prepared with acid concentration of preferably ~ 0.75 mole/l and solution to ore ratio L/S from 3 m³/t to 6 m³/t.

In the 5-th particular case an additional difference of the proposed method from an improved implementation lies in the fact that ferric process ore is used as mined ore and is crushed to the class of 5 mm.

In the first refinement to the 5-th particular case an additional difference of the proposed method lies in the fact that granulation of mined crushed ore belonging to the ferric process type is performed till formation of granules of 10 - 20 mm with addition of, at least, 0.1 - 2 % of sodium salt, 0.5 - 5.5 % elemental sulfur, 5 - 15 % of water and process water of the ore mass.

In the second refinement to the 5-th particular case an additional difference of the proposed method consists in the fact that firing of granulated mined ore belonging to the ferric process type is performed at the temperature of 300 - 700 °C and with delivery of sharp water vapor superheated till the temperature of 100 - 200 °C.

In the 6-th particular case an additional difference of the proposed method from an improved implementation lies in the fact that magnesium process ore is used as mined ore and is crushed to the class of 10 mm.

In the first refinement to the 6-th particular case an additional difference of the proposed method lies in the fact that granulation of crushed mined ore belonging to magnesium process type is performed till formation of granules of 20 - 40 mm with addition of, at least, 5 - 15% of water and process water of the ore mass.

In the second refinement to the 6-th particular case an additional difference of the proposed method lies in the fact that firing of granulated crushed mined ore belonging to magnesium process type is performed at the temperature of 200 - 500 °C.

In the 7-th particular case an additional difference of the proposed method from improved implementation lies in the fact that at the stage of copper extraction the ion-exchange resin belonging to chelating type is used as a resin, at the second stage of copper extraction copper elution is performed with ammonia solution of pH value not less than 7 for further production of copper ammine solution as copper eluate and free-of-copper resin, whereupon purified from copper resin is washed with water from ammonia ion with further deriving ammonia ion solution and regenerated resin.

In the first refinement to the 7-th particular case an additional difference of the proposed method lies in the fact that copper is extracted, at least, from some part of copper ammine
solution derived at the stage of copper extraction by means of crystallization with production of copper salt and ammonia regeneration, at that some regenerated ammonia is fed for ammonia solution preparation, which, in turn, is used for copper elution.

In the second refinement to the 7-th particular case an additional difference of the proposed method lies in the fact that ammonia ion solution made by means of washing the resin purified from copper is fed for preparation of ammonia solution intended for copper elution.

In the third refinement to the 7-th particular case an additional difference of the proposed method lies in the fact that regenerated resin is returned to the first stage of copper extraction.

In the 8-th particular case an additional difference of the proposed method from improved implementation lies in the fact that acidity of raffinate from the copper loading process is maintained within the pH range from 1 to 2, at the first stage of nickel extraction ion-exchange resin belonging to chelating type is used as a resin, at the second phase of the first stage of nickel extraction purified nickel eluate with nickel concentration of 60 - 90 g/l and pH value of 1 - 2 is used as purified nickel eluate, at the third phase of the first stage of nickel extraction nickel solution with nickel concentration of 40 - 70 g/l and acid concentration of 1 - 1.5 mole/l is used as nickel-bearing solution for production of nickel-eluate from additional loading resin, that at the fourth phase of the first stage of nickel extraction acid solution with acid concentration of 1 - 1.5 mole/l is used as acid solution for deriving secondary nickel eluate from a resin, whereupon the resin is washed with water for deriving regenerated resin and process water, which, in turn, is fed for preparation of acid solution intended for the fourth phase of the first stage of nickel extraction.

In the refinement to the 8-th particular case an additional difference of the proposed method lies in the fact that, at least, some part of regenerated resin produced by means of its washing with water is returned to the first phase of the first stage of nickel extraction.

In the 9-th particular case an additional difference of the proposed method from improved implementation lies in the fact that at the second stage of nickel extraction anion resin AB-17x8 or another similar anion exchange resin is used as a resin at the second stage of nickel extraction, whereupon purified nickel eluate with nickel content of 80 - 100 g/l is derived as raffinate from the iron loading process.

In the 10-th particular case an additional difference of the proposed method from an improved implementation lies in the fact that nickel salt as a nickel-bearing product is produced, at least, from some part of that part of purified nickel eluate produced as raffinate from the iron loading process which is delivered for production of nickel-bearing items by means of crystallization.
In the refinement to the 10-th particular case an additional difference of the proposed method lies in the fact that sulfuric acid solution is used as acid contained in solutions, by elution of which nickel eluate and secondary nickel eluate are produced at the first stage of nickel extraction, whereupon nickel salt (NiSCV7H2O) is derived by means of crystallization as a nickel-bearing product.

In the 11-th particular case an additional difference of the proposed method from improved implementation lies in the fact that cathode nickel as a nickel-bearing product is produced by means of electrolysis, at least, from that part of purified nickel eluate derived as raffinate from the iron loading process that is fed for production of nickel-bearing items.

In the refinement to the 12-th particular case an additional difference of the proposed method lies in the fact that the first stage of nickel extraction based on anion resin AB-17x8 or another similar anion resin is performed with sorption solution pH value from 1 to 3, whereupon anion resin containing iron is washed with water with further production of nickel-bearing process water, upon that iron elution is performed with acid solution of acid concentration 0.5 - 1 mole/l. Then washed anion resin is regenerated by sodium hydroxide with pH value of 5 - 8 for further production of bittern intended for anion resin and regenerated anion resin, whereupon regenerated anion resin is washed with water from sodium ion to pH value of 4 - 5.

In the particular case of the refinement to the 12-th particular case an additional difference of the proposed method lies in the fact that regenerated anion resin washed from sodium ion is returned to the first phase of the second stage of nickel extraction.

In the particular case of the refinement to the 12-th particular case an additional difference of the proposed method lies in the fact that, at least, some part of nickel-bearing process water derived by washing of iron-bearing anion resin with water, is added to nickel eluate fed to the first phase of the second stage of nickel extraction.

In the particular case of the refinement to the 12-th particular case an additional difference of the proposed method lies in the fact that, at least, some part of bittern produced in the course of anion resin regeneration is fed for preparation of sodium hydroxide solution, which participates in anion resin regeneration.

In the particular case of the refinement to the 12-th particular case an additional difference of the proposed method lies in the fact that, at least, some part of bittern produced in
the course of anion resin regeneration is fed for preparation of acid solution with concentration of 0.1 - 1 mole/1, which participates in iron elution.

In the 13-th particular case an additional difference of the proposed method from improved implementation lies in the fact that some part of raffinate from the nickel loading process fed to the stage of cobalt extraction shall be neutralized to pH value of 3 - 5.

In the first refinement to the 13-th particular case an additional difference of the proposed method lies in the fact that some part of raffinate from the nickel loading process with cobalt concentration of 0.8 г/1 and over is used as raffinate from the nickel loading process fed to the stage of cobalt extraction.

In the second refinement to the 13-th particular case an additional difference of the proposed method lies in the fact that at the stage of cobalt extraction ion-exchange resin is used as a resin. Besides, cobalt elution is carried out by mineral acid solution with concentration of 0.5 - 2 mole/1, preferably - 1.5 mole/1.

In the particular implementation of the second refinement to the 13-th particular case an additional difference of the proposed method lies in the fact that cobalt-free ion-exchange resin is washed with water and returned to the stage of cobalt sorption, at that process water derived in the course of washing process is fed for preparation of mineral acid solution to be used at the stage of cobalt elution.

In the particular implementation of the second refinement to the 13-th particular case an additional difference of the proposed method lies in the fact that, at least, some part of cobalt eluate fed for production of cobalt-bearing items is used for production of cobalt salt by means of crystallization.

In the third refinement to the 13-th particular case an additional difference of the proposed method lies in the fact that some part of raffinate from the cobalt loading process is fed to the heap containing another type of rock as a neutralized solution with production of insoluble compounds of iron and other metals, whereupon insoluble compounds are separated in a heap, and magnesium solution is produced.

In the 14-th particular case an additional difference of the proposed method lies in the fact that, at least, some part of magnesium solution derived after filtration of raffinate from the cobalt loading process in a flushed leached heap is purified from magnesium, whereupon some part is fed for crushed ore granulation, and the other - for leach solutions preparation.

In the first refinement to the 14-th particular case an additional difference of the proposed method lies in the fact that, at least, some part of magnesium solution produced in the course of raffinate from the cobalt loading process filtration in a flushed leached heap is purified from
niagne§ium y mearis pyrohydrolysis w t h groductiqn pf magnesium o¾de and regenerated acid.

In the second refinement to 14-th particular case an additional difference of the proposed method lies in the fact that, at least, some part of magnesium solution produced in the course of raffinate from the cobalt loading process filtration in a flushed leached heap is purified from magnesium by means of crystallization with production of magnesium salt and water.

In the 15-th particular case an additional difference of the proposed method lies in the fact that, at least, some part of magnesium solution derived after filtration of raffinate from the cobalt loading process in a flushed leached heap shall be purified from magnesium with production of a solution requiring the fishery water quality criterion, which is released to the ambient environment.

In the particular implementation of the first refinement to the 14-th particular case an additional difference of the proposed method lies in the fact that, at least, some part of regenerated acid produced by means of pyrohydrolysis of magnesium solution is returned to the cycle and fed to leach solutions preparation.

In the particular implementation of the first refinement to the 14-th particular case an additional difference of the proposed method lies in the fact that, at least, some part of regenerated acid produced by means of pyrohydrolysis of magnesium solution is used for neutralization with production of insoluble compounds of iron and other metals in that part of raffinate from the nickel loading process, which is fed to the stage of cobalt extraction.

In the 16-th particular case an additional difference of the proposed method lies in the fact that water from available sources without additional cleaning is used for compensation water losses within the cycle, therefore, it is added, at least, into one of the following solutions: raffinate from the nickel loading process; raffinate from the cobalt loading process; purified from magnesium solutions fed for crushed ore granulation or for leach solutions preparation.

Features of the proposed invention that are common with the features of the closest analogue shall provide realization of technical solution purpose.

Distinctive features together with general features provide for reaching of the technical result.

Granulated ore firing increases the share of soluble compounds of metals in the ore, raises
the degree of nickel and other metals extraction, and improves ore granules strength. Strong ore granules increase the speed and steadiness of solution motion within the heap, and that reduces the share of heap areas "falling out" of leaching process. Maximum allowable height of the heap shall be increased (till 12 m compared to the value of 7 m that is typical for the closest analogue).

Reducing acidity of bitterns produced at the first stages of heap leaching directly or with additional neutralization performed in heaps containing another type of rock till the level necessary for implementing the first stage of pregnant solutions processing allows simplifying and cheapening the construction and assembling of pregnant solution processing area, as far as fed pregnant solution in the area of processing shall not be neutralized. This also allows preventing expenditures for neutralizing materials and for neutralization waste recovery in the processing area, as far as neutralization of specified bitterns is reached along with metals leaching or with application of free neutralizing resource of another type of rock (spent rock etc.). At that preliminary firing of ore granules (and possibility of heaps height increasing) accelerates leaching and neutralization, reduces the quantity of heap leaching stages necessary for set level of neutralization.

Deriving purified metal eluate allows using purified eluate for additional loading of resin containing this metal. In the process of such additional loading impurities ions passed into the resin from sorption solution along with metal pass into elution solution (into purified metal eluate). The result is that sorption cycle for this metal is increased. Such sorption efficiency increase is especially important for the proposed method, as far as processed solutions may contain relatively high content of contaminating impurities in processed solutions compared to known methods that includes operation of preliminary neutralization of fed solution in the area of processing that is accompanied by precipitation of some impurities. In the proposed method reaching of technical result is conditioned by retaining contaminating impurities in a pregnant solution. Impurities (and acid) are partially involved in solutions cycle and returned from treatment for leach solution preparation. Impurities contained in a prepared leach solution prevent further impurities passing from ore into leach solution at the stages of leaching and increase the share of metals in the extracted mass. Restriction of nickel concentration (maximum 0.3 g/l) in generated solutions fed from processing to leach solutions preparation is, in average, optimum, well-balanced, calculated on the basis of experiments with regard to this concentration influence on the requirements to parameters of pregnant solution treatment stages. Reduction of nickel concentrations till the values less than 0.3 g/l is hindered due to increased content of impurities in processed solutions (without regard to purified eluates) provided by the proposed
Neutralization of solutions produced in the course of the treatment with yield of insoluble iron compounds and other metals is conducted insofar it is required to decrease iron concentration, taking into account, that these solutions or part of them after these insoluble compounds separation can be used further to prepare leach solutions or for granulation of crushed ore or can be discharged to the environment. Separation of these insoluble compounds by means of filtration in the flushed heap or gangue heap provides decrease in costs for separation, disposal and deactivation of the insoluble compounds as there is no need to install filter equipment. Besides, while precipitating such compounds plug up pores on the surface of the leached ore or gangue. As a result, migration of detrimental impurities from ore particles or gangue to the environment is reduced and expenses for rehabilitation of the area are saved.

Forming the ore heaps higher than 12 m is detrimental for uniformity of the heap and reduces completeness of ore leaching at the base of the heap. Heaps lower than 1 m require larger areas and are characterized by increased evaporation losses during dry season and excessive water intake in wet season and also require more complicated irrigation equipment.

Heap leaching in at least one process stream allows to use not only one but, if necessary, two or more streams and to leach ore of various types in different streams to optimize leaching processes for each technological type of ore.

Heap flushing with solutions with residual acidity and further filtration in the flushed heaps of solutions which contain insoluble compounds is arranged with solutions which acidity reduces gradually (from flushing to filtration) and improves deactivation of production wastes.

Collection of the bittern produced at various stages of heap leaching in separate reservoirs facilitates optimal selection of parameters of the leach solution for each stage of leaching.

Preparation of original leach solutions for heap leaching with acid concentration within the range of 0.2 mole/l ÷ 2 mole/l enables rational consumption of acid and restricts corrosion load to the equipment. If concentration level is less than 0.2 mole/l leaching velocity decreases unacceptably. If concentration level is over 2 mole/l expenses for keeping equipment available increase unacceptably and acid consumption for metal extraction increases too.

Density of heap irrigation at each stage of leaching within the range of 10 l/h/m² ÷ 20 l/h/m² corresponds to velocity of leach solution permeability through heaps stacked from ore granules roasted in compliance with the present method (including the cases where bittern produced during the further leaching stage of the previous heap is used directly as a leach solution at any stage of leaching except the last one).
Heap leaching including such number of stages and such number of streams, that at least in one stream pH value of the bittern produced directly at the first stage of leaching or after preliminary neutralization on a gangue heap varies from 1 to 3, allows to start treatment of the bittern as a pregnant leach solution without pre-neutralization during the leach solution treatment while the first stage of treatment can employ commercially available ion-exchange raisins with high selectivity properties towards the extracted metal from solutions with pH ranging from 1 to 3,

If blend of bitterns with pH between 1 and 3 collected at the first stages of heap leaching in several process streams directly or after preliminary neutralization on a gangue heap is reported to treatment as a pregnant leach solution, a bittern of one of the process streams with pH less than 1 without pre-neutralization can be used if pH of bittern blend ranges from 1 to 3.

A stage of copper extraction, a stage of nickel extraction and a stage of cobalt extraction are included as arranged into several stages of the pregnant leach solution treatment to provide additional extraction of copper. Leach solution

Extraction of copper by ion exchange method together with use of resin selectively extracting copper from a solution with pH ranging from 1 to 3 as the first stage of the pregnant leach solution treatment allows to avoid pre-neutralization of the pregnant leach solution during its treatment as the specified pH range of the solution to be sorbed is more favorable for copper extraction than for extraction of nickel and cobalt with application of commercially available ion-exchange materials and methods.

Extraction of nickel as the second stage of the pregnant leach solution treatment gives advantage of that the stage of copper extraction is accompanied by decrease in raffinate acidity compared with acidity of the pregnant leach solution and so pre-neutralization of the raffinate from the copper loading process is not needed prior to its supply to the stage of nickel extraction.

Two-step nickel extraction process combines extraction of nickel at the first step and extraction of iron at the second step in one treatment cycle of the pregnant leach solution and produces purified nickel eluate resulting from iron extraction from nickel eluate by means of purified nickel eluate at the first step of the nickel extraction stage to complete loading of the resin loaded with nickel.

Pumping of some portion of the raffinate from nickel loading to the leached heap flushing ensures partial decrease in acidity of the leached ore and subsequently lowers negative impact of the leached ore to environment. Flushing of each heap after completion of the last stage of its leaching by a portion of a raffinate from the nickel loading process and further use of this raffinate portion to prepare a leach solution allows to return acid entrained in the leached heap.
bask to the cycle.

The raffinate from nickel loading is preferentially used for flushing of the leached heap if cobalt concentration in it is less, for example, 0.8 g/l, i.e. cobalt extraction from solution will be inefficient. Besides, after heap flushing such raffinate or its portion can be used for preparation of a leach solution. As the raffinate from the nickel loading process contains cobalt its application for preparation of leach solutions contributes to recirculation of cobalt at heap leaching and gradual elevation of cobalt concentration in the pregnant leach solution and so to increased cobalt concentration in the raffinate from the nickel loading process. The portion of the raffinate from the nickel loading process in which cobalt concentration is too high, for example, 0.8 g/l, should rather be neutralized for the further stage of cobalt extraction. As a result cobalt extraction efficiency increases.

Feeding of the raffinate from the cobalt loading process or its portion to a flushed leached heap ensures further reduction of acidity of the leached ore.

The use of limonite and/or saprolit as ores to be heap-leached in different particular cases separately or in parallel process streams or as a blend facilitates adjusting of parameters of hydrometallurgical process if raw material parameters change. Preparation of the leach solution intended for heap leaching with acid concentration of ~ 0.5 mole/l and L/S ratio from 1.5 m³/t to 3 m³/t ensures optimal equilibrium between liquid consumption for solution preparation and concentration of metals to be extracted in pregnant leach solution for the majority of laterite ores. L/S ratio below 1.5 m³/t reduces nickel extraction to solution and increases suspension viscosity that worsens agitation. L/S ratio being more than 3.0 m³/t increases residual acidity of the bittern and can require more numbers of leaching stages.

Roasting of limonitic laterite or blend of limonitic and saprolitic laterite at a temperature of 300 °C ÷ 700 °C together with injection of live steam at 100 °C ÷ 200 °C significantly enhances amount of soluble metal compounds especially in limonite ore. At lower temperature the process is less efficient. At higher temperature ferrites are formed and nickel extraction becomes less efficient and energy consumption increases. Steam prevents oxidation by atmospheric oxygen which at high temperature oxidizes ion containing minerals found in limonitic laterite to acid-resistant hematite sharply decreeing cobalt and nickel extraction.

Roasting of saprolitic laterite at a temperature of 200 °C ÷ 500 °C increases amount of soluble metal compounds and enhances ore permeability. At a lower temperature the process is not efficient enough. Elevation of the temperature above that does not improve results of roasting.

The selected preparation conditions of ore (crushing, granulation and roasting) for heaps
significantly reduce acid consumption for heap leaching.

Heap leaching of only a portion of initial laterite together with in-situ leaching of the rest amount of the mined laterite represented by saprolit allows to reduce costs for ore lifting assuming that we can apply common technologies of in-situ leaching which shows good results in respect of saprolitic laterite.

Use of the bittern of saprolit ore in-situ leaching for preparation of a leach solution intended for heap leaching ensures higher concentration of nickel and other metals in the pregnant leach solution which than is treated by means of ion-exchange processes and ensures more complete utilization of acid considering that a bittern of in-situ leaching of saprolitic laterite contains relatively small amount of metals to be recovered.

Preparation of a leach solution intended for in-situ leaching of saprolitic laterite with acid concentration of preferentially ~ 0.75 mole/1 (i.e. higher than in average for heap leaching) ensures acceptable level of metal extraction to a solution taking into account that in the course of in-situ leaching of saprolitic laterite considerable amount of acid is consumed by ore.

Preparation of a leach solution for heap leaching with acid concentration of ~ 0.5 mole/1 at solution versus ore L/S ratio from 1.5 m³/t to 3 m³/t together with preparation of a leach solution intended for in-situ leaching with acid concentration of ~ 0.75 mole/1 at solution versus ore ratio L/S ratio from 3 m³/t to 6 m³/t allows us to calculate and to maintain for given concentration values optimal ration of ore weight in one heap versus underground ore weight, in this case it is 2, when using equal solution volume for both heap and in-situ leaching. It insures efficient integration of heap and in-situ leaching, rational and economical consumption of a leach solution. Besides, elevated L/S ratio for in-situ leaching is required as hydrometallurgical conditions are worse as compared with parameters of a granulated ore heap.

DESCRIPTION OF THE DRAWINGS

The present invention is clarified using examples and the following pictures.

Figure 1. Degree of nickel extraction to a leach solution from limonitic and saprolitic laterite versus leaching period.

Figure 2. Stream-sheet of the present hydrometallurgical technique.

Figure 3. Stream-sheet of the continuous multi-stage countercurrent heap leaching (reservoirs for collection of bittern are not shown).

Figure 4. Stream-sheet of integration of in-situ leaching and continuous multi-stage countercurrent heap leaching.

Figure 5. Stream-sheet of the pregnant leach solutions treatment (copper extraction stage).
Figure 6. Stream-sheet of the pregnant leach solutions treatment (nickel extraction stage).

ΦΗΓ. 7. Stream-sheet of the pregnant leach solutions treatment (cobalt extraction stage).


Figure 9. Stream-sheet of the present method using as example hydrometallurgical treatment of mined limonitic laterite.

Figure 10. Stream-sheet of the present method using as example hydrometallurgical treatment of limonitic and saprolitic laterite blend.

Figure 11. Stream-sheet of the present method using as example hydrometallurgical treatment of mined limonitic laterite.

Figure 12. Stream-sheet of the present method using as example joint hydrometallurgical treatment of excavated saprolitic laterite and saprolitic ore in-situ.

Figure 13. Stream-sheet of the present method using as example hydrometallurgical treatment of excavated limonitic and saprolitic laterite separated by type.

Figure 14. Stream-sheet of the present method using as example joint hydrometallurgical treatment of excavated limonitic laterite and saprolitic laterite in-situ.

In addition the examples shows production of finished metallurgical products from eluates of metals extracted from oxide ore by the present method.

EMBODIMENTS OF THE INVENTION

Hydrometallurgical process which applies the present invention involves preparation of laterite for leaching, leaching by irrigation of ore with mineral acids, for example, $\text{H}_2\text{SO}_4$, $\text{HCl}$, $\text{HNO}_3$, and bio-acids yielded in the course of bio-leaching and further ion-exchange sorption treatment of the pregnant leach solution.

Laterite ores are treated depending on amount of iron and magnesium oxides present in them. The ores are divided into categories by the relative content of magnesium and iron in them for example, saprolitic laterite and limonitic laterite.

Limonitic laterite contains at least 0.5 % of nickel, 0.06 % of cobalt, 10 % of iron and 1.5 % of magnesium, and saprolitic laterite contains at least 0.3 % of nickel, 0.01 % of cobalt, 3 % of iron and 7.5 % of magnesium.

Iron content in saprolitic laterite can be at a high level, for example, 12.39 weight %.

Table 1 shows chemical composition of limonitic and saprolitic laterite.
Table 1. Saprolitic and limonitic laterite chemical composition

<table>
<thead>
<tr>
<th>Laterite type</th>
<th>Ni</th>
<th>Co</th>
<th>Fe</th>
<th>Mn</th>
<th>Mg</th>
<th>Ca</th>
<th>SiO₂</th>
</tr>
</thead>
<tbody>
<tr>
<td>Saprolitic laterite</td>
<td>0.84</td>
<td>0.025</td>
<td>12.39</td>
<td>0.37</td>
<td>8.79</td>
<td>8.5</td>
<td>31.4</td>
</tr>
<tr>
<td>Limonitic laterite</td>
<td>1.22</td>
<td>0.12</td>
<td>37.87</td>
<td>1.6</td>
<td>3.45</td>
<td>7</td>
<td>20</td>
</tr>
</tbody>
</table>

According to Table 1 content of magnesium in saprolitic laterite is rather high, approximately 8.79% by weight. Content of magnesium in limonitic laterite is considerably less - approximately 3.45% by weight.

Difference in leachability found between saprolit ore and limonite ore and studied in the course of tests under static conditions are clearly described by curves in Figure 1 proving that leaching of these ores are completely different under similar conditions in sulfuric acid leaching medium.

As for saprolit and limonite ores they were leached by sulfuric acid under the following conditions:

- Particle size - 10 mm;
- Concentration of \( \text{H}_2\text{SO}_4 \)- 100 g/l;
- L/S ratio - 3 m³/t;
- Temperature of leaching - ambient temperature (23 °C);
- Duration of leaching - 1488 hours;
- Agitating - 1 time per day.

Leaching under static conditions at ambient temperature provided recovery of Ni 52.84% for saprolitic laterite and recovery of 29.03% for limonitic laterite. The findings show that it is not rational to leach limonitic laterite directly. To increase nickel recovery at leaching of limonitic laterite we exposed it to special reduction roasting with steam treatment.

Presence of fine-grain and clay particles (minerals) in laterite requires, first of all, granulation (pelletizing) of ore in case of more efficient method of heap leaching to provide rather high velocity and uniform permeability of leach solutions through the total ore amount without disturbing the shape and high mechanical strength on compression of ore granules.

Stream-sheet in Figure 2 shows the following stages of hydrometallurgical treatment of laterite in compliance with the present invention in a general case:

1. crushing of the excavated ore 1;
2. granulation of the crushed ore 2;
3. roasting of granulated crushed ore 3;
IV - stacking of heap series 5 of roasted ore 4;
V - preparation of leach solutions (LS) intended for heap leaching 6;
VI - continuous multi-stage countercurrent heap leaching with yield of pregnant leach solutions (PL) 7 and leached heaps 8;
VII - flushing of leached heaps 8 by solutions 9 to produce flushed heaps 10 and acid solution 11 containing soluted impurities;
VIII - treatment of pregnant leach solutions 7 to produce solutions 9 with certain acidity and soluted impurities and nickel concentration under 0.2 g/l and to produce solutions 12 neutralized to yield insoluble compounds (IC) of iron and other metals;
IX - removing of insoluble compounds by filtration through the flushed heap 10 or a gangue heap to produce solution 13 containing magnesium.

Stream-sheet in Figure 4 includes additional stage X (in-situ leaching of initial saprolitic laterite 15 to produce bitterns 17) of hydrometallurgical treatment of lateritic ores in compliance with developed implementation of the present invention. Preparation of leach solutions in this implementation includes preparation of leach solutions for in-situ leaching 16.

Prior to heap leaching of the excavated limonite ore it undergoes preliminary treatment (example 1 for one of the best modes of implementation, see Figure 9):
- crushing (I) up to size - 5 mm;
- ore is palletized (II) up to size 10 ÷ 20 mm, with addition of sulfur in amount of at least 0.5 % ÷ 5.5 %, salt, at least, 0.1 % ÷ 2 % and 5 % ÷ 15 % of water or process water;
- roasting (III) at 300 °C ÷ 700 °C, with supply of live steam of 100 °C ÷ 200 °C, ore unloading and cooling down to 100 °C avoiding its coming into contact with atmospheric oxygen;
- staking of heaps (IV) of limonitic laterite of up to 12 m high.

Prior to heap leaching of saprolitic laterite it undergoes preliminary treatment (example 2 for yet another one of the best modes of implementation, see Figure 11):
- crushing (I) up to size - 10 mm;
- ore is palletized (II) up to size 20 ÷ 40 mm, with addition of 5 ÷ 15 % of water or process water and if necessary binding agents such as liquid glass or/and cement;
- roasting (III) at 200 °C ÷ 500 °C;
- staking of heaps (IV) of saprolitic of up to 6 m high.

Heap leaching of both limonite and saprolit involves treatment with mineral acids, for example, H₂SO₄, or/and HCl, or/and HN0₃, or/and acids generated by bacteria. Concentration of acids employed for lateritic ores leaching is at least 0.2 mole/l. Concentration of the acid can be
either constant or tends to increase in the course of ore leaching. Relative volume of solution is equal to:

\[ \text{L/S ratio} = 1.5 \text{ m}^3/\text{t} \div 3 \text{ m}^3/\text{t}. \]

Heap leaching VI of lateritic ores is a continuous countercurrent process including at least two stages, preferentially three stages, given in Figure 3. Countercurrent heap leaching is characterized by reverse stream of ore and leach solution. At the first stage VI-1 bittern 62, collected from the second stage VI-2 of heap leaching is neutralized on a fresh heap 5. At the second stage VI-2 of heap leaching a heap 5 is introduced into the process after the first stage VI-1 is leached by bittern 61, collected from the third stage VI-3 of heap leaching. At the third stage VI-3 of heap leaching a heap 5 after the second stage of VI-2 is irrigated by leach solution 6 from stage V of leach solution preparation. The solution generated after the first stage VI-1 of heap leaching is the pregnant leach solution 7 which is pumped to further treatment VIII. If the fourth stage VI-4 and etc. stages are involved then ore (ore heap) from the third stage VI-3 of heap leaching is directed to the fourth stage VI-4, and from the fourth stage is sent to the fifth stage, etc. The spent ore after the final stage VI-3 of heap leaching is sent to flushing VII, and the solutions stream in the reverse direction from flushing VII to preparation V of leach solutions 6 from preparation of leach solutions V to the final stage VI-3, from the final stage VI-3 to the first stage VI-1 of heap leaching VI, from the first stage VI-1 to treatment VIII of the pregnant leach solution 7.

Saprolitic laterite which meets the natural criteria such as degree of ore watering, permeability potential of ores, occurrence of impermeable layers and mineral form of metal compounds relatively easy destroyed by solvent is qualified for in-situ leaching.

In the present invention in-situ leaching X is conditionally assumed as the final stage of continuous countercurrent multi-stage, including minimum three stages, process of heap and in-situ leaching (See Figure 4) which first stages are integrated in a stream-sheet of heap leaching VI presented in Figure 3.

The stream-sheet of heap and in-situ leaching of laterite ores uses mineral acids, for example, \( \text{H}_2\text{SO}_4 \), or/and \( \text{HCl} \), or/and \( \text{HN0}_3 \), or/and acids, generated by bacteria. In this process acidity is adjusted both prior to in-situ leaching and heap leaching. Concentration of acids used is at least 0.75 mole/1 for in-situ leaching and 0.5 mole/1 for heap leaching. Concentration of the acid to be used is constant or tends to increase in the course of ore leaching. Volume of leach solution per ore weight is equal to L/S ratio = 1.5 m³/t ÷ 3 m³/t for heap leaching and L/S ratio = 3 m³/t + 6 m³/t for in-situ leaching, so weight \( T(\text{heap leaching}) \) of ore used for one heap leaching versus weight \( T(\text{in-situ leaching}) \) of ore for in-situ leaching is equal to:
T(HL)/T(ISL) = 2,
as for each process of leaching (HL and ISL) similar amount of leach solution is used.

The pregnant leach solution 7 collected and neutralized to pH values of 1 ÷ 3 is sorbed
VIII on ion-exchange resin to separate copper, nickel and cobalt from the solution as their
eluates which are processed to recover finished products.

The first stage VIII-Cu of processing VIII is copper extraction from the pregnant leach
solution 7, the stream-sheet of which is given in Figure 5. The first sub-stage VIII-Cu-1 (copper
sorption) uses chelate resin 18, for example, Dowex M4195, to produce resin loaded with
copper 20. The second sub-stage VIII-Cu-2 (copper elution) includes addition of ammonia
solution 21 1% ÷ 10% to extract copper and to produce copper eluted resin 23. At the end of the
process the resin is washed (stage VIII-Cu-3) with water 24 to yield an ammonia-free resin 18
directed to stage VIII-Cu-1. During the loading process raffinate 19 from which copper has been
extracted is produced. The solution resulted from copper elution is a copper-ammonia solution
22 which is subjected to crystallization (stage VIII-Cu-4), to produce copper hydroxide 28 and
regenerated ammonia 27. The third solution produced after resin washing is, solution 25
containing ammonia ions which is used to prepare (stage VIII-Cu-5) ammonia solution for
copper elution 21 from ammonia 29 and regenerated ammonia 27.

The second stage VIII-Ni of the process (See Figure 6) is nickel extraction from the
raffinate from the copper loading process 19 which is performed in two sub-stages 1-Ni and 2-
Ni. The first phase 1-Ni-1 of the first sub-stage 1-Ni is nickel sorption on a chelate resin, for
example Dowex M4195, at acidity with pH values of 1 ÷ 2 to produce resin loaded with nickel
31. In the second phase 1-Ni-2 resin loading with nickel is completed to produce additionally
loaded resin 32 by the solution 33a (purified nickel eluate) with nickel concentration of 60 g/l ÷
100 g/l and pH values of 1 ÷ 2. The first nickel elution where the additionally loaded resin is
mainly unloaded from nickel 32 is run during the third phase 1-Ni-3 by solution 35 containing 40
g/l ÷ 70 g/l of nickel and 1 mole/l ÷ 1.8 mole/l of acid to yield unloaded resin 34. The fourth
phase 1-Ni-4 includes the second elution with complete unloading of resin 34 from nickel to
produce fully unloaded resin 36 by acid solution 37 at a concentration of 1 mole/l ÷ 1.8 mole/l.
After nickel elution (the fifth phase 1-Ni-5) resin 36 is washed with water 38 to get acid-free
resin 30 to be returned to the phase 1-Ni-1. At the end of the first phase of the first sub-stage of
the nickel separation stage (i.e. nickel sorption) the raffinate from the nickel loading process 39
is produced containing basic impurities of iron, magnesium and cobalt; a portion of the raffinate
is added to solution 9 for heap flushing VII after heap leaching, then the collected solution 11 is
used to prepare a leach solution V. The other portion of the raffinate from the nickel loading
process 39b after cobalt concentration in raffinate 39 is adjusted to minimum 0.8 g/1 due to the solution recirculation in the leaching-treatment cycle goes to the third stage of the process VIII-Co-cobalt extraction stage. The second solution generated after the phase of additional loading (the second phase 1-Ni-2), is raffinate of additional loading 40 which is added to solution 9 for heap flushing VII after heap leaching. The third solution produced after elution during the third phase of the first sub-stage of nickel separation stage is nickel eluate 41 (80 g/1 ÷ 100 g/1 by Ni) with iron impurities; it goes to the second sub-stage 2-Ni of nickel separation stage VIII-Ni -production of a purified nickel eluate 33. The fourth solution produced after nickel elution during the fourth phase 1-Ni-4 of the first sub-stage 1-Ni of the nickel separation stage VIII-Ni, is the a secondary nickel eluate 42 - acid nickel solution, which together with acid solution 43 and solution 33b with nickel at a concentration of 60 g/1 ÷ 100 g/1 and pH values of 1 ÷ 2 is used for preparation of (phase 1-Ni-6) elution solution 35 for the third phase 1-Ni-3 of the first stage of nickel separation. The fifth solution 44 produced after resin washing to eliminate acid is used together with acid solution 45 for preparation (phase 1-Ni-7) of elution solution 37 of acid for the fourth phase 1-Ni-4 of the first sub-stage of nickel separation stage.

The second sub-stage 2-Ni of stage VIII-Ni of nickel separation is production of the purified nickel eluate 33. The first phase 2-Ni-1 of the second sub-stage is sorption of iron and other impurities from nickel eluate 41 produced at the first sub-stage 1-Ni bearing iron impurities. Iron sorption is run on anion exchanger 46 of brand AB-17x8 in form of $\text{SO}_4^{2-}$ or $\text{Cl}^-$ depending on the acid to be used at pH value of 1 ÷ 3 of the solution to produce resin loaded with iron 47. Then resin 47 is washed (phase 2-Ni-2) with water to remove nickel solution 48 and to have resin washed 49. Than elution of iron (phase 2-Ni-3) is performed with acid solution 50 at a concentration of 0.5 mole/l ÷ 1 mole/l to have the iron eluted resin 51. After that resin is regenerated (phase 2-Ni-4) by solution 52 of sodium hydroxide with pH values of 5 ÷ 8, with production of regenerated resin 53. The final phase 2-Ni-5 is resin washing 53 with water 54 to adjust pH values to 4 ÷ 5, to gain the resin free from sodium ion 46. Iron sorption results in purified nickel eluate 33, one portion of nickel eluate 33a is consumed in phase 1-Ni-2 and the second portion of nickel eluate 33b is consumed in phase 1-Ni-6, the third portion of the nickel eluate 33c is consumed for production of final product at a refining stage 2-Ni-8 with yield of metal nickel 61 and/or metal nickel powder 62 and/or nickel salt ($\text{NiSO}_4\cdot\text{7H}_2\text{O}$) 63 during crystallization phase 2-Ni-9. The second generated solution 55 is process water containing significant amount of nickel coming from washing of resin loaded with iron from nickel 47; the water is mixed with a nickel eluate 41 from the first sub-stage of the nickel separation stage to the second sub-stage 2-Ni (iron sorption phase 2-Ni-1). The third solution
produced during iron sorption is ferrous eluate 56 which is mixed with solution 9 and is used for leaching of heap VII after its heap leaching. The fourth solution 57 produced after resin regeneration 51 and containing sodium ions is divided into portions where one portion 57a is used together with solution 58 for production (phase 2-Ni-6) of elution solution 50 for the iron elution phase and the other portion 57b being mixed with the fifth solution 59, produced after washing of the regenerated resin to eliminate sodium ion 53 is used for preparation (phase 2-Ni-7) of solution 52 of resin regeneration where it is adjusted with sodium hydroxide 60.

The third stage VIII-Co of treatment VIII is cobalt extraction (See Figure 7) from portion 39b of the raffinate from the nickel loading process 39 with cobalt concentration of 0.8 g/l. At the first sub-stage VIII-Co-1 portion 39b of raffinate from the nickel loading process is neutralized in the presence of brucite or/and magnesium oxide, or/and magnesium carbonate or/and sodium hydroxide 64 to adjust pH values to approximately 3 to 6. Resulting slurry 65 depending on iron content in the raffinate from the nickel loading process contains ferrous cake (insoluble iron and other metals compounds) in L/S ratio = 10 m³/t ÷ 100 m³/t. At the second sub-stage VIII-Co-2 of the cobalt extraction stage neutralized solution (slurry) 65 is sorbed on resin 66, for example, Lewatit TP207, Lewatit TP220, Purolit S960, Dowex M4195. This sub-stage produces resin 67 loaded with cobalt. Then at the third sub-stage VIII-Co-3 cobalt elution is run 68 by mineral acid solutions, for example H₂SO₄, or/and HC1, or/and HN0₃, preferentially H₂SO₄, to cobalt eluted have ion-exchange resin 69. At the fourth sub-stage VIII-Co-4 the eluted ion-exchange resin 69 is washed to remove acid 70 and to have a washed resin 66 which is returned to the cobalt sorption sub-stage VIII-Co-1. As a result of cobalt sorption (sub-stage VIII-Co-2 of cobalt extraction stage) a raffinate from the cobalt loading process 12 as a solution containing a significant amount of iron hydroxide solids is produced and streams to separation IX of iron insoluble compounds (IC) (ferrous cake) and other metals by means of filtration on a spent heap (i.e. leached) and washed at flushing stage VII by the solutions mixture 9 or to a gangue heap. It produces solution 13, containing magnesium. Slurry volume 12 pumped to iron filtration per ore weight L/S ratio is 0.1 m³/t ÷ 1 m³/t. The second solution produced at the cobalt extraction stage is a cobalt eluate 71, collected at the elution sub-stage VIII-Co-3 which serves to produce metal cobalt 72 at the sub-stage of electrolysis VIII-Co-5 and/or cobalt salt (CoSO₄·7H₂O) 73 at the crystallization sub-stage VIII-Co-6. The third resulting solution 74 is process water which together with acid solution 75 is used for preparation of (sub-stage VIII-Co-7) a elution solution 68 for a cobalt elution sub-stage VIII-Co-3.

Figure 8 additionally presents a stream-sheet of finished magnesium product production from solution 13 collected after filtration of the raffinate from the cobalt loading process 12
containing high concentration of iron, at stage IX. The first portion 13a of the magnesium solution 13 is consumed at stage V (preparation of leach solution), the second portion 13b of the magnesium solution 13 is consumed for the granulation stage II, the third portion 13c is consumed for production of regenerated acid 76 and magnesium oxide 77 at pyrohydrolysis stage XI and/or magnesium salt (MgSO$_4$*7H$_2$O) 78 at the stage of crystallization XII. The regenerated acid 76 is sent to the stage V of leach solution preparation 6, and the magnesium oxide 77 can be either a finished product or a recirculating reagent, used at the cobalt extraction stage VIII-Co for neutralization (sub-stage VIII-Co-1) of raffinate from the nickel loading process 39b. Resulting magnesium salt 78 is a finished product. Depending upon the size of the deposit, ore grade by nickel, occurrence of limonite or saprolit, possibility of their separation as well as availability of conditions for in-situ leaching (ISL) of saprolitic laterite we distinguish the following peculiarities in implementation of the present method of nickel, cobalt and other metals extraction from laterite:

- heap leaching of excavated limonitic laterite (Figure 9);
- heap leaching of the blend of excavated limonitic and saprolitic laterite (Figure 10);
- heap leaching of excavated saprolitic laterite (Figure 11);
- heap leaching of excavated saprolitic laterite and in-situ leaching of saprolitic laterite (Figure 12);
- separate heap leaching of excavated limonitic and saprolitic laterite (Figure 13);
- heap leaching of excavated limonitic laterite and in-situ leaching of saprolitic laterite (Figure 14).

Stream-sheet of hydrometallurgical process in compliance with the present invention among other possible stream-sheets can include any of the stream-sheets represented in Figures 9, 10, 11, 12, 13 and 14. Indices "L" or /and "S" accompanying some designations in the Figures indicate association of the item with limonite or/and saprolit correspondingly.

Figure 9 shows an example of implementation of the process of complex hydrometallurgical processing of limonitic laterite:

- crushing (I$_L$) down to size - 5 mm;
- granulation (granulation II$_L$) is arranged up to the size 10 ÷ 20 mm with addition of sulfur 2a in the amount at least 0.5 % ÷ 5.5 %, salt 2b in the amount at least 0.1 % ÷ 2 % and 5 % * 15 % of water or process water;
- roasting (III$_L$) is run at a temperature of 300 °C ÷ 700 °C with supply of live steam 3a at 100 °C ÷ 200 °C, ore being unloaded and cooled to 100 °C excluding ore reaction with atmospheric oxygen;
- limonite ore heaps (IVL) are up to 12 m high.

Figure 10 describes an example of hydrometallurgical processing of laterite, limonite and saprolit, which are difficult to separate by types due to economical technological conditions.

Figure 11 depicts an example of hydrometallurgical processing of excavated saprolit.

Differences between hydrometallurgical processing of saprolite ore and hydrometallurgical processing of limonite ore are found in the conditions of the following process stages:

- crushing (Is) down to size - 10 mm;
- granulation (granulation lis) up to the size 20 ÷ 40 mm with supply of 5 % ÷ 15 % water or process water, and if necessary a binding agent is added, for example, a liquid glass or/and cement;
- roasting (lls) is run at a temperature of 200 °C ÷ 500 °C;
- heaps (IVs) of saprolit are stacked up to 6 s.

Figure 13 presents a stream-sheet uniting the stream-sheets given in Figure 9 and Figure 11 and describing the process with separate processing routs of limonite and saprolit. The general stages for these two parallel processes are as follows:

- preparation V of a leach solution 6 which is divided into a portion 6L consumed for heap leaching VI L of limonite ore, and a portion 6s consumed for heap leaching Vis of saprolit;
- treatment VIII of the pregnant leach solution 7 which includes a solution 7L (a pregnant leach solution after heap leaching VI L of limonite ore) and 7s (a pregnant leach solution after heap leaching Vis of saprolit); a solution 9 resulting after treatment VIII and bearing soluted impurities, breaks down into a solution 9L consumed for flushing VIII of a spent limonite ore heap and a solution 9s consumed for flushing Vlls of a spent saprolit heap.

As an example of application of the present invention for bench tests we examined dynamic column leaching of limonite and saprolit (hereinafter "limonite ore" and "saprolitic ore" for short). Numbering of stages of the laboratory process might not coincide with the corresponding numeration of the above described examples of the full-scale process.

Study of granulation was conducted in 6 columns: 3 columns with limonite ore and 3 columns with saprolitic ore.

For granulation of limonite ore we used excavated limonite ore which underwent the following treatment:

- crushing down to size -5 mm;
- granulation with addition of sulfur 2 %, salt 1 %, water 5 % up to granule size:
  a) 5 mm ÷ 10 mm (column No 1);
  6) 10 mm ÷ 20 mm (column No 2);
b) 20 mm ÷ 40 mm (column No 3);
- granules roasting at a temperature of 500 °C and steam supply at 100 °C.

For granulation of saprolitic ore we used excavated saprolitic ore which were subjected to the following treatment:

- crashing down to size -10 mm;
- granulation with addition of water 5 % up to granule size:
  a) 10 mm ÷ 20 mm (column No 4);
  b) 20 mm ÷ 40 mm (column No 5);
  b) 40 mm ÷ 50 mm (column No 6);
- granules roasting at a temperature of 200 °C.

Chemical composition of the granulated ore is given in Table 2.

Table 2. Chemical composition of the granulated ore

<table>
<thead>
<tr>
<th>No</th>
<th>Ore type</th>
<th>Chemical composition, %</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Ni</td>
</tr>
<tr>
<td>1</td>
<td>Granulated limonite ore</td>
<td>0.87</td>
</tr>
<tr>
<td>2</td>
<td>Granulated saprolitic ore</td>
<td>0.63</td>
</tr>
</tbody>
</table>

After granulation the ore was put in columns of 1 m high and 0.2 m in diameter. Then a sulfuric acid solution was added into the upper part of the column at a velocity of 10 l/h/m², with concentration of 50 g/l (0.5 mole/l) and of volume equal to L/S ratio = 3 m³/t during 90 days. After leach solution seepage through the ore the bittern was adjusted with sulfuric acid to a concentration of 50 g/l (0.5 mole/l). The results of the experiment aimed at determination of the optimal granulation size of limonite and saprolitic ore are given in Table 3.

Table 3. Degree of Ni extraction at leaching vs. granulation size of limonite and saprolitic ore

<table>
<thead>
<tr>
<th>No</th>
<th>Ore type</th>
<th>Ni extraction at various granulation size, %</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>5 mm ÷ 10 mm</td>
</tr>
<tr>
<td>1</td>
<td>Granulated limonite ore</td>
<td>78.89</td>
</tr>
<tr>
<td>2</td>
<td>Granulated saprolitic ore</td>
<td>–</td>
</tr>
</tbody>
</table>
According to Table 3, the optimal granulation size for hematite ore is 10 mm to 20 mm, and for saprolitic ore is 20 mm to 40 mm. Larger granulation of ore decreases Ni recovery from ore which smaller granules decrease the filtration velocity in the columns and at a full-scale process can cause pore plugging up and cessation of leaching.

For the study of heap leaching, we used findings of dynamic leaching in Column No. 2 for granulated limonite ore of 10 mm to 20 mm and Column No. 5 for granulated saprolitic ore of 20 mm to 40 mm. The results are given in Table 4.

In the study described above, concentration of free acid $\text{H}_2\text{SO}_4$ which remained in the leach solution varied from 19.09 g/l to 27.7 g/l (pH<1). A high acidity can result in certain difficulties at the ion exchange stage of nickel extraction. The solution after leaching is filtered through a newly stacked heap of ore to neutralize the residual free acid in the solution down to pH value of 1-3. Bitterns collected after leaching in Column No. 2 and Column No. 5 were neutralized in another two additional columns of the dimensions similar to the ones of Columns No. 1÷No. 6 filled with:

- granulated limonite ore of size 10 mm to 20 mm (Column No. 7);
- granulated saprolitic ore of size 20 mm to 40 mm (Column No. 8).

Bitterns are neutralized by supply of bittern after Column No. 2 to the upper part of Column No. 7 and supply of bittern after Column No. 5 to the upper part of Column No. 8. Bitterns were supplied at a velocity of 10 l/h/m².

Finding of heap leaching test are given in Table 4.

Table 4. Finding of heap leaching simulation test (Table 4 is ending on the next page).

<table>
<thead>
<tr>
<th>b</th>
<th>Ore weight, m</th>
<th>Chemical composition of bittern after leaching and neutralization (g/l)</th>
<th>Recovery, %</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Ni</td>
<td>Co</td>
</tr>
<tr>
<td>1</td>
<td>30</td>
<td>83.6</td>
<td>2.46</td>
</tr>
</tbody>
</table>
The advantage of the process according to the present invention is that a neutralizing agent is ore. So it let save acid and neutralizing reagents such as sodium bicarbonate, sodium hydroxide, magnesium carbonate, brucite, lime, etc. There is no necessity in special hardware for a solution neutralization process.

Pregnant leach solution (PLS) treatment

For this purpose the pregnant leach solution collected after Column No7 was used (granulated limonite ore).

The collected pregnant leach solution is treated by ion exchange process including copper extraction stage, nickel extraction stage and cobalt extraction stage.

Copper is absorbed on a chelate resin Dowex M4195. Active functional group was bis-pycoliamine. As it is an amine resin, the resin is protonated with acid solution.

The first stage of the ion-exchange bench test is sorption of copper impurities as a resin Dowex M4195 has better selectivity for copper against nickel. Depending upon ore copper concentration in the pregnant leach solution can be elevated to 0.1 g/l.

The experiment on copper sorption was done in a glass tube of 15 mm in diameter and 100 mm high tightly filled with a resin Dowex M4195. The solution was poured with velocity of 5 m/h and at L/S ratio equal to 443.

Elution was made by 5% ammonia solution. The test findings are given in Table 5.
Table 5. Bench test findings on the copper extraction stage

<table>
<thead>
<tr>
<th>Number</th>
<th>Stage</th>
<th>Solution</th>
<th>Vp/Vc</th>
<th>Solution amount, l</th>
<th>Chemical composition of solutions, g/l</th>
<th>Resin capacity, g/l</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Ni</td>
<td>Co</td>
</tr>
<tr>
<td>1</td>
<td>Cu-1</td>
<td>Pregnant leach solution</td>
<td>339.70</td>
<td>60</td>
<td>2.54</td>
<td>0.20</td>
</tr>
<tr>
<td>2</td>
<td></td>
<td>Raffinate from the copper loading</td>
<td>339.70</td>
<td>60</td>
<td>2.53</td>
<td>0.20</td>
</tr>
<tr>
<td>3</td>
<td>Cu-2</td>
<td>Ammonia</td>
<td>3.00</td>
<td>0.53</td>
<td></td>
<td></td>
</tr>
<tr>
<td>4</td>
<td></td>
<td>Copper eluate</td>
<td>3.00</td>
<td>0.53</td>
<td>1.36</td>
<td>11.21</td>
</tr>
<tr>
<td>5</td>
<td></td>
<td>Water</td>
<td>1.00</td>
<td>0.18</td>
<td></td>
<td></td>
</tr>
<tr>
<td>6</td>
<td></td>
<td>Process water</td>
<td>1.00</td>
<td>0.18</td>
<td>0.01</td>
<td>0.11</td>
</tr>
</tbody>
</table>

The second stage of the ion-exchange bench test of the pregnant leach solution treatment is nickel extraction from a raffinate from the copper loading process collected after copper extraction from the pregnant leach solution by sorption.

Nickel extraction stage is divided into two sub-stages.

The experiment on the first sub-stage of nickel extraction was run in a glass tube of 25mm in diameter and 250 mm long tightly packed with resin Dowex M4195 and with bottom-up supply of the solution.

The first phase of the first sub-stage is nickel sorption. A raffinate from the copper loading process was added at a velocity of 5 m/h to yield a resin loaded with nickel and a raffinate from the nickel loading process.

The second phase (additional loading) included supply of an additional loading solution with nickel concentration of 90 g/l and pH value of 1.8 with at a velocity 1 m/h to produce resin maximum loaded with nickel and a raffinate of additional loading.
The third phase included initial elution of nickel by eluting solution No1 with concentration 60 g/l by nickel and 100 g/l by sulfuric acid supplied at a velocity of 1 m/h to produce a partially eluted resin and nickel eluate.

The fourth phase includes secondary nickel elution by eluting solution No2 with sulfuric acid concentration of 150 g/l supplied at a velocity of 1 m/h to produce a fully eluted resin and secondary nickel eluate.

The fifth stage included resin washing with water supplied at a velocity of 1 m/h to eliminate acid solution to yield washed resin and process water. The results of the test for the first sub-stage of nickel extraction are given in Table 6.

Table 6. The results of the test for the first sub-stage of nickel extraction (Table 6 is ending on the next page)
The experiment on the first sub-stage of nickel extraction was run in a glass tube of 25mm in diameter and 250 mm long tightly packed with anionite AB-17x8, and with bottom-up supply of solution.

The first phase of the second sub-stage of nickel sorption includes sorption of iron (III) from nickel eluate collected in the previous experiment. The sorption was run on anionite AB-17x8, put in $\text{SO}_4^{2-}$ form, with solution supply at a velocity of 1 m/h to produce lauded with iron (III) resin and a purified nickel eluate.

The second phase included washing of anionite with water supplied at a velocity of 1 m/h to eliminate nickel. As a result there was resin loaded with iron (III) washed from nickel anionite and nickel solution.

The third phase included elution of iron (III) by eluting solution with acid concentration of 150 g/l, supplied at a velocity of 1 m/h to yield iron eluted anionite (III), but loaded with sulfuric acid and a ferrous (III) eluate.

The fourth phase included regeneration of anionite in $\text{SO}_4^{2-}$ form by a regenerating solution of sodium hydroxide with pH value of 8 supplied at a velocity of 5 m/h to obtain anionite regenerated in $\text{SO}_4^{2-}$ form and process water.

At the fifth phase the anionite was washed with water supplied at a velocity of 5 m/h to eliminate sodium and, as a result, to obtain a washed regenerated anionite and process water.

The results of the test for the second sub-stage of nickel extraction are given in Table 7.

Table 7. The results of the test for the second sub-stage of nickel extraction (Table 7 is ending on the next page)
The third stage of bench test of ion-exchange process is cobalt extraction from the portion of raffinate from the nickel loading process.

Cobalt extraction stage uses recalculating raffinate of nickel loading with cobalt concentration adjusted to over 0.8 g/l.

For the test on cobalt extraction the raffinate from the nickel loading process collected earlier was adjusted by nickel up to 0.8 g/l.

At the first stage of cobalt extraction stage a portion of raffinate from the nickel loading process was neutralized by brucite to adjust pH value to 5. The process was run in a glass beaker of 1 liter with mechanical agitating for 0.5 hour. It resulted in a neutralized portion of raffinate from the nickel loading process (slurry).

The second phase included cobalt sorption from the neutralized portion of raffinate from the nickel loading process (slurry) by addition of resin Lewatit TP 207 in L/S ratio equal to 15. Mechanical agitating for 30 minutes to facilitate cobalt sorption was carried out. After sorption the raffinate of cobalt loading was separated from the resin loaded with cobalt on the screen.
At the second phase cobalt was stripped by eluting solution of sulfuric acid with a concentration of 100 g/l and relative volume $V_p/V_c$, equal to 2. Elution was run in a glass beaker of 200 ml with continuous mechanical agitating for 30 minutes to obtain cobalt eluate and eluted resin separated on a screen.

At the third phase the resin was washed with water from acid ($V_p/V_c$ equal to 1) in a glass beaker of 200 ml with continuous mechanical agitating for 30 minutes to obtain washed resin and process water to be separated on a screen.

The findings of the experiment on cobalt extraction are given in Table 8.

Table 8. The findings of the experiment on cobalt extraction

<table>
<thead>
<tr>
<th>number</th>
<th>Phase</th>
<th>Solution</th>
<th>$V_p/V_c$</th>
<th>Solution amount, l</th>
<th>Chemical composition of solutions, g/l</th>
<th>Resin capacity, g/l</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Co-1</td>
<td>Raffinate of nickel loading</td>
<td>15.00</td>
<td>0.60</td>
<td>Ni 0.11 Co 0.80 Fe 11.49 Mg 5.44 H2SO4 pH=1.52</td>
<td>Ni 1.65 Co 11.81 Fe 5.82</td>
</tr>
<tr>
<td>2</td>
<td></td>
<td>Neutralized raffinate of nickel loading</td>
<td>16.50</td>
<td>0.66</td>
<td>0.10 0.73 10.44 4.95 pH=5</td>
<td></td>
</tr>
<tr>
<td>3</td>
<td>Co-2</td>
<td>Neutralized raffinate of nickel loading</td>
<td>16.50</td>
<td>0.66</td>
<td>0.10 0.73 10.44 4.95 pH=5</td>
<td></td>
</tr>
<tr>
<td>4</td>
<td></td>
<td>Raffinate of cobalt loading</td>
<td>16.50</td>
<td>0.66</td>
<td>0.00 0.01 10.09 4.95 pH=2.5 pH=5</td>
<td></td>
</tr>
<tr>
<td>5</td>
<td>Co-3</td>
<td>Eluting solution</td>
<td>2.00</td>
<td>0.08</td>
<td>Ni 0.00 Co 0.00 Fe 0.00 H2SO4 pH=5 30.67 100.00</td>
<td></td>
</tr>
<tr>
<td>6</td>
<td></td>
<td>Cobalt eluate</td>
<td>2.00</td>
<td>0.08</td>
<td>Ni 0.00 Co 0.00 Fe 0.00 H2SO4 pH=5 30.67 100.00</td>
<td></td>
</tr>
<tr>
<td>7</td>
<td>Co-4</td>
<td>Water</td>
<td>1.00</td>
<td>0.04</td>
<td>Ni 0.00 Co 0.00 Fe 0.00 H2SO4 pH=5 30.67 100.00</td>
<td></td>
</tr>
<tr>
<td>8</td>
<td></td>
<td>Process water</td>
<td>1.00</td>
<td>0.04</td>
<td>Ni 0.00 Co 0.00 Fe 0.00 H2SO4 pH=5 30.67 100.00</td>
<td></td>
</tr>
</tbody>
</table>

The next stage is ferric hydroxide separation from solution (slurry) after cobalt sorption.

For the test two types of solution were prepared with iron concentration of 7.5 g/l and 47.8 g/l, then the solutions were adjusted by magnesium oxide to pH value 4.
Filtration test was run in 4 columns of 200 mm in diameter and 1 rhpung which were previously used:
- columns No2 and No7 - granulated limonite ore after leaching;
- columns No5 and No 8, granulated saprolitic ore after leaching.

Cake resulting from leaching and flushing of granulated limonite ore was put in Columns No 1 and No2. Cake resulting from leaching and flushing of granulated saprolitic ore was put in Columns No 3 and No4.

Slurry of ferric hydroxide with iron concentration of 7.5 g/l was poured into columns No2 and No5, and slurry with iron concentration of 47.8 g/l was poured into columns No 7 and No 8.

Lower concentration of solids in the slurry increases significantly solids penetration through ore layer and higher concentration of solids reveals quick plugging up of ore pores in the column. The slurry was poured in L/S ratio of 1 and the amount of solids at the exit was measured.

The results of the filtration test are given in Table 9.

<table>
<thead>
<tr>
<th>No</th>
<th>Solution type</th>
<th>Solution composition for test No1, g/l</th>
<th>Solution composition for test No2, g/l</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>H$_2$SO$_4$</td>
<td>Fe</td>
</tr>
<tr>
<td>1</td>
<td>Raffinate of cobalt loading</td>
<td>pH=4.3</td>
<td>7.5</td>
</tr>
<tr>
<td>2</td>
<td>Solution after slurry filtration through cake of spent granulated limonite ore</td>
<td>pH=3.4</td>
<td>0.34</td>
</tr>
<tr>
<td>3</td>
<td>Solution after slurry filtration through cake of spent granulated saprolitic ore</td>
<td>pH=2.5</td>
<td>0.13</td>
</tr>
</tbody>
</table>

**COMMERCIAL PRACTICABILITY**

Benefit test and full-scale test carried out under control of the applicants showed that implementation of the present method is possible with use of conventionally available hydrometallurgical equipment, reagents and materials.
CLAMS

1. The process of extraction of nickel, cobalt and other metals from laterite ores, including preparation of leach solutions, that contain acid belonging to the group including hydrochloric acid, sulphuric acid, nitrogen acid and acids formed by bacterial activity, leaching of original laterite by means of prepared leach solutions with obtaining of pregnant leach solutions and processing of pregnant leach solutions, said process being characterized by the fact that the original laterite contains the mined laterite, leaching of the mined laterite is made in the way of a continuous multistage countercurrent heap leaching, in this it is previously exposed to crushing, the crushed ore is granulated, then at least one sequence of heaps is formed from the granules, every heap is exposed to at least two stages of leaching, in every sequence of heaps at the first stage of its leaching the leach solution is added to every heap, this leach solution is made of bittern prepared from bittern obtained at the second stage of leaching of the previous heap, at every intermediate stage of its leaching the leach solution, which was prepared from bittern gained at the next stage of leaching of the previous heap, is added, at the last stage of its leaching the original leach solution assigned for heap leaching is added, on completion of the last stage of its leaching it is flushed, and the bittern gained at the first stage of its leaching is delivered for processing as a pregnant leach solution or as a component part of the pregnant leach solution, the processing of pregnant leach solutions is made stepwise using ion-exchange resins and ion-exchange processes arranged into several stages with obtaining at every stage of processing the eluate enriched by metal corresponding to this stage of processing, and depleted by other metals in comparison with metal content in pregnant leach solutions and eluates gained at other stages of processing, said process being different in that the granules are previously exposed to roasting before forming from them a sequence of heaps, then the continuous multistage countercurrent heap leaching with as many stages as it is necessary to provide for the decreasing of acidity of bitterns gained at the first stages of the heap leaching up to the level necessary for the first stage of processing of pregnant leach solutions is performed, or these bitterns are additionally exposed to neutralization up to the mentioned required level of acidity by means of filtering them on the waste heap before passing them on for processing, at least one eluate gained at the stage of processing corresponding to it, is purified from impurities, and the purified eluate is gained, production of this eluate is made with additional loading of corresponding ion-exchange resin by means of adding to it a part of obtained purified eluate, at least a part of the solutions formed during processing, in which acidity and impurities are saved in a soluted form, and the concentration of nickel is not more than 0.3 g/l, is passed to flush of the leached heap and then for preparation of leach solutions, at least a part of the solutions, formed during processing, is
neutralized obtaining the insoluble iron and other metals compounds, and these compounds are separated from at least a part of these neutralized solutions by means of filtering on the leached and flushed heap or on the waste heap.

2. Process according to claim 1, which is different in that the heaps are formed with a high of 1 m + 12 m, heap leaching is done in at least one process stream, bitterns, obtained at different stages of heap leaching, are gathered in separate storages, original leach solutions, assigned for heap leaching, are prepared with acid concentration of 0.2 mole/1 ÷ 2 mole/1 and are passed to heaps surfaces, which are at the last stage of leaching, with irrigation with density of 10 l/h/m² ÷ 20 l/h/m², bittern obtained at every, except the first, stage of the heap leaching is passed for the irrigation of the other heap, which is at the previous stage of leaching with irrigation with density of 10 l/h/m² ÷ 20 l/h/m², heap leaching is done in such quantity of stages and in such quantity of streams, that at least in one stream the value of the bittern pH, obtained directly at the first stage of leaching or after additional neutralization on the waste heap, can be from 1 till 3, as the pregnant leach solution or as a component part of the pregnant leach solution the bittern with a pH value from 1 till 3 is passed for processing, this bittern is obtained at the first stage of one process stream of heap leaching directly or after additional neutralization on the waste heap, alternatively the mixture of bitterns with pH from 1 till 3 can be added, these bitterns are obtained at first stages of the heap leaching in several process streams directly or after additional neutralization of bitterns on the waste heap, as processing stages of the pregnant leach solution at least the following is done: a stage of copper extraction, a stage of nickel extraction, a stage of cobalt extraction; as the first stage of the pregnant leach solution processing the stage of copper extraction by means of ion exchange is done, and as a resin the ion exchange resin, capable to extract copper selectively from the solution with pH from 1 to 3 is used, at the first phase of stage of copper extraction the copper-loaded resin and the raffinate from the copper loading process are obtained, at the second phase of stage of copper extraction the copper eluate is obtained from copper-loaded resin by means of copper elution, the copper eluate is passed for gaining of copper-containing products, raffinate from the copper loading process is passed to the second stage of the pregnant leach solution processing, as this second stage the stage of nickel extraction by means of ion-exchange is done, and the nickel eluate is obtained as the eluate, which is purified from impurities with obtaining of the purified eluate, this eluate obtaining is done with additional loading of the corresponding resin to it by means of passing to it a part of obtained purified eluate, the stage of nickel extraction is done in two steps, at the first step of stage of nickel extraction the nickel eluate with impurities is formed, at the second step of stage of nickel extraction the nickel eluate is purified from impurities, at the first phase of the first step
of stage of nickel extraction the nickel loaded resin, contaminated by impurities, and raffinate from the nickel loading process are obtained, at the second phase of the first step of stage of nickel extraction the nickel loaded resin is additionally loaded by nickel and is partly purified from impurities by means of passing to it a nickel purified eluate with obtaining of the additionally loaded resin and the raffinate from the additional loading, contaminated by impurities, at the third phase of the first step of stage of nickel extraction the nickel eluate is obtained from the additionally loaded resin by means of elution with nickel containing solution, at the fourth phase of the first step of stage of nickel extraction the secondary nickel eluate is obtained from the resin by means of elution with acid solution, the secondary nickel eluate is passed for preparation of the nickel containing solution for the third phase of the first step of stage of nickel extraction, raffinate from the additional loading process is passed as a solution, in which the acidity and impurities are saved in the soluted form, for leached heap flushing and then for preparation of leach solutions, the nickel eluate is passed for using in the second step of stage of nickel extraction, raffinate from the loading process is divided into parts, and some part of the raffinate from the loading process is passed for leached heap flushing and then for preparation of leach solutions, and the other part of the raffinate from the nickel loading process is neutralized with obtaining of insoluble iron and other metals compounds, and at least one part of this another part of the raffinate from the nickel loading process is neutralized using at least one of the materials belonging to a group containing magnesia, brucite, magnesite, and the neutralized part of the raffinate from the nickel loading process is passed to the stage of the cobalt extraction, at the first phase of the second step of stage of nickel extraction the nickel eluate is purified by means of iron extraction, after that the resin loaded by iron and other impurities and the purified nickel eluate as a raffinate from the iron loading process, the purified nickel eluate is partly given to the second phase of the first step of stage of nickel extraction, partly given for preparation of nickel containing solution for the third phase of the first step of stage of nickel extraction, partly given for obtaining of nickel containing products, the iron loaded resin is washed from nickel and is passed to the iron elution phase, which is done by means of acid solution with obtaining of the ferrous eluate and the washed resin, the ferrous eluate is given as solution, in which the acidity and impurities are saved in a soluted form, for leached heap flushing and then for preparation of leach solutions, at the first phase of stage of cobalt extraction the cobalt loaded resin and the raffinate from the cobalt loading process are obtained, the cobalt loaded resin is passed to the cobalt elution phase, this elution is made by means of acid solution with obtaining of cobalt eluate and cobalt eluted resin, the cobalt eluate is given for obtaining of cobalt containing products, at least one part of the raffinate from the cobalt
loading process is passed as solution, neutralized with plating iron and other metals containing compounds, to the flushed leached heap or to the waste heap with separation of these insoluble compounds by means of filtering correspondingly on the flushed leached heap or on the waste heap and with obtaining of magnesium solution, the solution getting as a result of leached heap flushing, is passed for preparation of leach solutions.

3. The process according to claim 2, which is different in that the mined limonite is used as all of the mined laterite.

4. The process according to item 3, which is different in that the original leach solutions assigned for the heap leaching are prepared with the acidity concentration of ~ 0.5 mole/l with the proportion of the solution to the ore L/S being from 1.5 m$^3$/t to 3 m$^3$/t.

5. The process according to claim 3, which is different in that the crushing of the mined limonite is done up to class of 5 mm.

6. The process according to claim 3, which is different in that the granulation of the mined limonite is done till the granules with the size of 10 mm ÷ 20 mm are formed with increments: at least 0.1 % ÷ 2 % of salt, 0.5 % ÷ 5.5 % of elemental sulfur, 5 % ÷ 15 % of water or process water of the ore mass.

7. The process according to claim 3, which is different in that the roasting of the granulated crushed mined limonite is done with the temperature of 300 °C ÷ 700 °C and with adding of sharp water vapor with the temperature of 100 °C ÷ 200 °C.

8. The process according to claim 2, which is different in that the mined saprolit is used as all of the mined laterite.

9. The process according to claim 8, which is different in that the original leach solutions assigned for the heap leaching are prepared with the acidity concentration of ~ 0.5 mole/l with the proportion of the solution to the ore L/S being from 1.5 m$^3$/t to 3 m$^3$/t.

10. The process according to claim 8, which is different in that the crushing of the mined saprolit is done up to class of 10 mm.

11. The process according to claim 8, which is different in that the granulation of the crushed mined saprolit is done till the granules with the size of 20 mm ÷ 40 mm are formed with increments: 5 % ÷ 15 % of water or process water of the ore mass.

12. The process according to claim 8, which is different in that the roasting of the granulated crushed mined saprolit is done with the temperature of 200 °C ÷ 500 °C.

13. The process according to claim 2, which is different in that the mixture of mined limonite and mined saprolit is used as all of the mined laterite.

14. The process according to claim 13, which is different in that the original leach solutions
assigned for the heap leaching are prepared with the acidity concentration of ~ 0.5 mole/l with the proportion of the solution to the ore L/S being from 1.5 m³/t to 3 m³/t.

15. The process according to claim 13, which is different in that the crushing of the mined ore is done up to class of 5 mm.

16. The process according to claim 13, which is different in that the granulation of the crushed mined ore is done till the granules with the size of 10 mm ÷ 20 mm are formed with increments: at least 0.1 % + 2 % of salt, 0.5 % ÷ 5.5 % of elemental sulfur, 5 % + 15 % of water or process water of the ore mass.

17. The process according to claim 13, which is different in that the roasting of the granulated crushed mined ore is done with the temperature of 300 °C ÷ 700 °C and with adding of sharp water vapor with the temperature of 100 °C ÷ 200 °C.

18. The process according to claim 2, which is different in that the mined limonite and the mined saprolit divided into types are used as all of the mined laterite, the heap leaching is done in at least two process streams, in at least one process stream the mined limonite is leached and in at least one other process stream the mined saprolit is leached.

19. The process according to claim 18, which is different in that the original leach solutions assigned for the heap leaching are prepared with the acidity concentration of ~ 0.5 mole/l with the proportion of the solution to the ore L/S being from 1.5 m³/t to 3 m³/t.

20. The process according to claim 18, which is different in that the crushing of the mined ore is done up to class of 5 mm.

21. The process according to claim 18, which is different in that the granulation of the crushed mined limonite is done till the granules with the size of 10 mm ÷ 20 mm are formed with increments: at least 0.1 % ÷ 2 % of salt, 0.5 % ÷ 5.5 % of elemental sulfur, 5 % ÷ 15 % of water or process water of the ore mass.

22. The process according to claim 18, which is different in that the roasting of the granulated crushed mined limonite is done with the temperature of 300 °C ÷ 700 °C and with adding of sharp water vapor with the temperature of 100 °C ÷ 200 °C.

23. The process according to claim 18, which is different in that the crushing of the mined saprolit is done up to the class of 10 mm.

24. The process according to claim 18, which is different in that the granulation of the crushed mined saprolit is done till the granules with the size of 20 mm ÷ 40 mm are formed with increments: 5 % ÷ 15 % of water or process water of the ore mass.

25. The process according to claim 18, which is different in that the roasting of the granulated crushed mined saprolit is done with the temperature of 200 °C ÷ 500 °C.
36. The process according to claim 2, which is different in that the original laterite particularly contains the ore at the in-situ, which belongs to saprolit, leaching of the ore at in-situ is done by way of In-situ leaching with obtaining of bitterns of In-situ leaching, and these bitterns are passed for preparation of leach solutions, the above mentioned bitterns of In-situ leaching are used for preparation of leach solutions assigned for continuous multistage countercurrent heap leaching, and the part of solutions, which are formed during processing and are passed for preparation of leach solutions, is used for preparation of leach solutions assigned for the In-situ leaching, these leach solutions, assigned for the In-situ leaching, are prepared with the acidity concentration of at least 0.5 mole/l.

27. The process according to claim 26, which is different in that the original leach solutions assigned for the heap leaching are prepared with the preferable acidity concentration of ~ 0.5 mole/l with the proportion of the solution to the ore L/S being from 1.5 m³/t to 3 m³/t, the leach solutions assigned for the In-situ leaching are prepared with the preferable acidity concentration of ~ 0.75 mole/l with the proportion of the solution to the ore L/S being from 3 m³/t to 6 m³/t.

28. The process according to claim 26, which is different in that the limonite is used as the mined ore, its crushing is done up to the class of 5 mm.

29. The process according to claim 28, which is different in that the granulation of the crushed mined limonite is done till the granules with the size of 10 mm ÷ 20 mm are formed with increments: at least 0.1 % ÷ 2 % of salt, 0.5 % ÷ 5.5 % of elemental sulphur, 5 % ÷ 15 % of water or process water of the ore mass.

30. The process according to claim 28, which is different in that the roasting of the granulated crushed mined limonite is done with the temperature of 300 °C ÷ 700 °C and with adding of sharp water vapor with the temperature of 100 °C ÷ 200 °C.

31. The process according to claim 26, which is different in that the saprolit is used as the mined ore, its crushing is done up to the class of 10 mm.

32. The process according to claim 31, which is different in that the granulation of the crushed mined saprolit is done till the granules with the size of 20 mm ÷ 40 mm are formed with increments: 5 % ÷ 15 % of water or process water of the ore mass.

33. The process according to claim 31, which is different in that the roasting of the granulated crushed mined saprolit is done with the temperature of 200 °C ÷ 500 °C.

34. The process according to any of the claims 2 - 33, which is different in that at the stage of copper extraction the chelating type ion exchange resin is used as a resin, at the second stage of copper extraction the copper elution is done by means of ammonia solution with pH of no less than 7 with obtaining of the copper ammine solution as a copper eluate and with obtaining of the
resin purified from the copper, the resin purified from the copper is washed by the water from the ammonium ion with obtaining of ammonium ion solution and the regenerated resin.

35. The process according to claim 34, which is different in fact that from at least one part of the copper ammine solution, formed on the stage of copper extraction, the copper is extracted by means of crystallization with obtaining of the cuprate and with the regeneration of ammonia, by means of which the copper elution is made.

36. The process according to claim 34, which is different in that the ammonium ion solution, obtained by means of washing of the purified from the copper resin, is passed for preparation of the ammonia solution, which is used for the copper elution.

37. The process according to claim 34, which is different in that the regenerated resin is passed back to the first stage of copper extraction.

38. The process according to any of claims 2 - 33, which is different in that the acidity of the raffinate from the copper loading process is in the pH value range of 1 to 2, on the first step of stage of the nickel extraction the chelating type ion exchange resin is used as a resin, on the second phase of the first step of stage of nickel extraction the purified nickel eluate with the nickel concentration of 60 g/l ÷ 90 g/l and pH value of 1 to 2 is used as the purified nickel eluate, on the third phase of the first step of stage of nickel extraction the nickel solution with the nickel concentration of 40 g/l ÷ 70 g/l and the acidity concentration of 1 mole/l to 1.5 mole/l is used as nickel containing solution for obtaining nickel from the additional loading resin of the nickel eluate, on the fourth phase of the first step of stage of nickel extraction the acid solution with the acid concentration of from 1 mole/l to 1.5 mole/l is used as the acid solution for obtaining a secondary nickel eluate from the resin, then this resin is washed by water with obtaining of the regenerated resin and the wash water. The wash water is passed for preparation of the acid solution for the fourth phase of the first step of stage of the nickel extraction.

39. The process according to claim 38, which is different in that at least one part of the regenerated resin obtained by washing it by water is passed back to the first phase of the first step of stage of nickel extraction.

40. The process according to any of claims 2 - 33, which is different in that the anion resin AB-17x8 or an analogous anion resin is used as a resin on the second step of stage of nickel extraction, as a raffinate from the iron loading process the purified nickel eluate with nickel content of 80 g/l ÷ 100 g/l is obtained.

41. The process according to any of claims 2 - 33, which is different in that from at least one part of the part of the purified nickel eluate obtained as a raffinate from the iron loading process, which is passed for obtaining of nickel containing products, the nickel salt is obtained as a nickel
containing product by means of crystallization.

42. The process according to claim 41, which is different in that as the acid containing in solutions, with the help of which by means of elution the nickel eluate and the secondary nickel eluate are obtained sulphuric acid on the first step of stage of nickel extraction, by means of crystallization the nickel salt, which has the formula NiSO₄·7H₂O, is obtained as nickel containing product.

43. The process according to any of claims 2 - 33, which is different in that from at least a part of the part of the purified nickel eluate, obtained as a raffinate from the iron loading process, which is passed for obtaining of nickel containing products, the nickel cathode is obtained as nickel containing product by means of electrolysis.

44. The process according to any of claims 2 - 33, which is different in that from at least one part of the part of the purified nickel eluate, obtained as the raffinate from the iron loading process, which is passed for obtaining of nickel containing products, the nickel powder as nickel containing product is obtained by means of electrolysis.

45. The process according to claim 40, which is different in that the first phase of the second step of stage of nickel extraction is done on the anion resin AB-17x8 or on the analogous anion resin, if the value of pH of the solution before loading is from 1 to 3, the iron loaded anion resin is washed by water with obtaining of the wash water, containing nickel, the iron elution is made by acid solution with the acidity concentration of 0.5 mole/l ÷ 1 mole/l, then the regeneration of the washed anion resin is made by sodium hydroxide with pH of from 5 to 8 with obtaining of the regenerated anion resin bittern and the regenerated anion resin, then the regenerated anion resin is washed by water till it gets rid of sodium ion enough for the pH increasing from 4 to 5.

46. The process according to claim 45, which is different in that the regenerated anion resin washed from the sodium ion is passed back to the first phase of the second step of stage of the nickel extraction.

47. The process according to claim 45, which is different in that at least a part of the wash water, containing nickel, obtained by the iron loaded anion resin washing by water, is added to the nickel eluate, passed to the first phase of the second step of stage of nickel extraction.

48. The process according to claim 45, which is different in that at least a part of the regenerated anion resin bittern is passed for preparation of sodium hydroxide, which is used for regeneration of the anion resin.

49. The process according to claim 45, which is different in that at least one part of the regenerated anion resin bittern is passed for preparation of the acid solution with the acidity concentration of 0.5 mole/l ÷ 1 mole/l, which is used for the iron elution.
50. The process according to any of claims 2 - 33, which is different in that the part of raffinate from the nickel loading process, being passed to the stage of cobalt extraction, is neutralized up to pH with value of from 3 to 5.

51. The process according to claim 50, which is different in that as the part of raffinate from the nickel loading process, being passed to the stage of cobalt extraction, the part of raffinate from the nickel loading process, in which the cobalt concentration reaches the value of 0.8 g/l and more, is used.

52. The process according to claim 50, which is different in that at the stage of the cobalt extraction the ion exchange resin is used as a resin, the step of cobalt elution is done by means of mineral acid with the acidity concentration of 0.5 mole/l ÷ 2 mole/l, preferably 1.5 mole/l.

53. The process according to claim 52, which is different in that the cobalt eluted resin is washed by water and is passed back to the step of cobalt loading, the wash water formed as a result of washing is passed for preparation of the mineral acid solution, which is used during cobalt elution.

54. The process according to claim 52, which is different in that at least from a part of cobalt eluate, passed for obtaining of cobalt containing products, the cobalt salt is obtained by means of crystallization.

55. The process according to claim 50, which is different in that a part of raffinate from the cobalt loading process is passed as solution, neutralized with obtaining of insoluble iron and other metals compounds, to the waste heap with separation of these insoluble compounds on it and with obtaining of magnesium solution.

56. The process according to claim 2, which is different in that at least a part of the magnesium solution, obtained as a result of raffinate from the cobalt loading process filtering on the flushed leached heap, is purified from magnesium and is passed partly for the granulation of the crushed ore and partly for the preparation of leach solutions.

57. The process according to claim 56, which is different in that the purification of the part of the magnesium solution, obtained as a result of the raffinate from the cobalt loading process filtering on the flushed leached heap, from magnesium is done by means of pyrohydrolysis with obtaining of magnesium oxide and the regenerated acid.

58. The process according to claim 56, which is different in that the purification of the part of the magnesium solution, obtained as a result of the raffinate from the cobalt loading process filtering on the flushed leached heap, from magnesium is done by means of crystallization with obtaining of magnesium salt and water.

59. The process according to claim 2, which is different in that at least a part of the magnesium
solution, obtained as a result of the raffinate from the cobalt loading process filtering on the flushed leached heap, is purified from magnesium with obtaining the solution, corresponding to fishery water quality criterion and is passed to the outside environment.

60. The process according to claim 57, which is different in that at least apart of the regenerated acid, obtained by means of pyrohydrolisis of the magnesium solution, is taken back to processing and is passed for preparation of leach solutions.

61. The process according to claim 57, which is different in that at least a part of the regenerated acid, obtained by means of pyrohydrolisis of the magnesium solution, is taken back to work and is used in processing of pregnant leach solutions.

62. The process according to claim 57, which is different in that at least a part of the magnesium oxide, obtained by means of pyrohydrolisis, is used during neutralization with obtaining of insoluble iron and other metals compounds, of the part of the raffinate from the nickel loading process, which is passed to the stage of cobalt extraction.

63. The process according to claim 2, which is different in that for compensating the water loss in circulation the water from available sources without additional purification is used, in so doing said water is added in at least one of the following solutions: raffinate from the nickel loading process; raffinate from the cobalt loading process; solutions purified from magnesium, passed for the granulation of the crushed ore or for preparation of leach solutions.
IV
Heap building up

VI-1
Stage No. 1

VI-2
Stage No. 2

VI-3
Stage No. 3

Pregnant leach solution to processing

Leach solution

To Heap flushing VII

Fig. 2
Pregnant leach solution

VI

ion exchange resin Dowex M4196

25, 27

29

Ammonia

VIII-Cu-5

Preparation of eluant

VIII-Cu-1

Copper loading

VIII-Cu-2

Copper elution

VIII-Cu-3

Resin washing of ammonium ion

VIII-Cu-4

Crystallization

Regenerated ammonia Cu(OH)₂

Process water

Copper eluate [Cu(NH₃)₄]

Water

22

21

22

19

To nickel extraction VII-Ni

18

20

23

24

25

27

28

Fig. 3
Fig. 5
Initial laterite ore

I
Crushing

II
Granulation

III
Roasting

13a
Acid bath (H₂SO₄)

IV
Heap building up

V
Preparation of leach solution

VI
Heap leaching

VII
Heap flushing

VIII
Processing of pregnant leach solution

IX
Separation of insoluble compound

X
Pyrohydrolysis

XI
Crystallization

XII
Magnesium salt MgSO₄•7H₂O

Fig. 8
Fig. 13
Fig. 14