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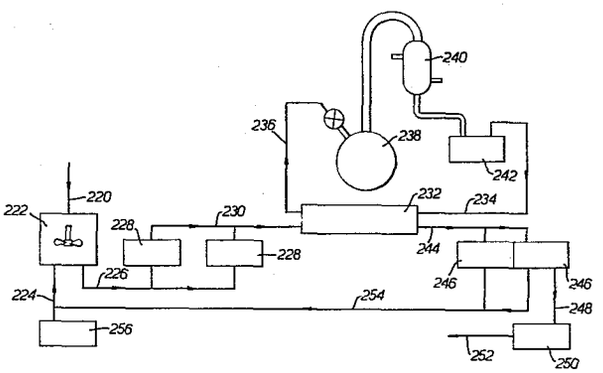
(54) **Extraction process.**

(57) Gold and/or other noble metals may be recovered from materials containing such metals, such as freshly mined ores or ores and slimes resulting from previous extraction procedures, using an aqueous leaching agent and extracting the desired metal from the resulting pulp or leach liquor. The leaching agent comprises a source of halide ions plus an oxidising agent, such as nitric acid.



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EXTRACTION PROCESS

This invention relates to the recovery of gold and other noble metal values from matter containing such values, such as ores or glacial deposits, and is particularly applicable to recovery from tailings slimes and other discarded materials from various previous methods for recovery of gold, silver and other noble metals.

As the chances of finding gold and other noble metals in viable concentrations decrease, the implementation of a more economically feasible process of recovery assumes greater commercial significance.

Previous attempts have been made to recover gold values from ores by leaching methods. Thus, the earliest method used for this purpose was to leach the ore with aqua regia, a highly corrosive mixture of concentrated nitric and hydrochloric acid. The use of such a chemical mix was fraught with difficulties, including serious danger to workmen and was abandoned, as was the use of chlorine gas in a strong acid solution, which method suffered similar disadvantages. Such processes have effectively been abandoned.

A later process, introduced in South Africa in about 1890 was the cyanidation process. In a typical process finely ground slime is allowed to settle in thickening tanks, the thickened portion is drawn off from the bottom and transferred for leaching with a large volume of dilute metallic cyanide plus lime etc., to cyanidation tanks, where, after agitation by aeration, the gold-bearing solution is bled off and the gold is removed from the solution by a contacting process with activated carbon. The gold adheres to the carbon, which is subsequently separated from the slimes by screening. This process is lengthy, usually requiring in excess of 30 hours.

The present invention seeks to overcome these problems. Furthermore, the present invention seeks to provide a process for the recovery of gold and/or other noble metal values which in comparison with current methods is less capital and labour intensive and which can give an effective extraction in a short period. It is also possible for the plant required to be mobile, skid mounted or easily moved and operable on a continuous basis. As the plant may be totally enclosed it is free of environmental objections to fumes and liquor pollution.

According to the present invention we provide a process for the recovery of gold and/or other noble metals from matter containing such metals, comprising contacting said matter in substantially finely divided form with an aqueous leaching agent comprising a source of halide ions and an oxidising agent capable of enhancing the leaching action of the halide ions and extracting the resulting soluble metal salts from the resulting pulp or leach liquor.

The metal-containing matter can be an ore which has been milled or screened, or otherwise treated, to remove or reduce bigger lumps of matter unlikely to be noble-metal bearing, e.g. lumps of granite. Alternatively, the matter may be in the form of a waste sand or slime, freshly mined or which possibly has been lying for some considerable time and which was previously thought not be be of commercial significance. The process of the invention preferably employs matter of comparatively low particle size, e.g. less than 0.5 mm.

The halide source, preferably chloride may be provided by inorganic halides such as calcium and magnesium chloride or polar organic halides. The halide component is preferably present in a concentration of from 3 to 15% by weight of the leach solution. A

A mixture of materials may be used, such as waste halide liquors resulting from other processes. A preferred halide is sodium chloride.

The oxidizing component of the leaching agent is preferably present in an amount of 2 to 15% by weight of the leaching agent and may be an acid such as nitric acid which is suitably used in a comparatively low concentration such as 5 to 12% by weight of the leaching agent, although it may be necessary to increase the nitric acid content, for example up to 25%, if the ore being treated has an alkalinity or carbonate content which will cause it to neutralise some of the nitric acid being used. However, less corrosive oxidizing agents may be employed, preferably inorganic nitrates (such as sodium, potassium or ammonium nitrate), but also organic nitrates, organic or inorganic peroxides, oxides, sodium chlorate, sodium hypochlorite, permanganates, chromates, dichromates and persulphates or other conventional oxidizing agents including gaseous air, ozone and oxygen. When using a nitrate as the oxidizing compound, enhancement of the leaching action of the halide may be obtained by adding low concentrations (for example 2 to 5% by weight) of acid such as nitric acid, hydrochloric acid, sulphuric acid or organic acids or by adding low concentrations of other conventional oxidizing agents as mentioned above.

It has been found that a mixture of acids gives an effective oxidising agent, especially a mixture of nitric acid plus sulphuric acid suitably with a halide such as sodium chloride as the other component of the leaching agent.

The leaching is preferably carried out at a pH not exceeding 3.5. The leaching is preferably carried out at a temperature in the range of 45° to 95°C, for

example about 90°C. It is desirable that the subsequent extraction of metal values is carried out at a lowered temperature, preferably in the range of 20°C to 45°C.

The metal values may be extracted from the pulp or liquor from the leaching step in several ways.

The metal values are preferably extracted from the leach liquor with a solvent which can be readily recovered for reuse. The preferred solvent is methyl isobutyl ketone which can be distilled for reuse leaving a residue of high purity metal, preferably gold. Other solvents such as diisobutyl ketone, amyl acetate, amyl chloride, diethyl ether, isopropyl ether, ethyl alcohol, acetone, chloroform, butyl acetate, methyl n-hexyl ketone, ethyl acetate and kerosene may be employed. The most suitable solvents are those with minimum solubility in water, acid solutions, salt solutions or any other resultant leach solution.

Alternatively the metal values may be extracted using an activated material such as cellulose chaff or activated carbon, which can be readily commercially obtained. Alternatively other activated materials may be employed capable of acting as precipitants, such as oxalic acid. Methods of subsequently recovering gold from an activated material such as charcoal are well known and include calcining, and non-destructive methods such as stripping with hydrogen cyanide or nitric acid or other concentrated acid or alkaline materials, deposition using zinc chips and electrolytic procedures such as the well known Zadra process.

Alternatively, when the metal being extracted is gold, the gold may be extracted from the pulp or liquor from the leaching step using an organic complexing agent specific for gold and recovering the gold values from the complex so formed using an activated material capable of removing the gold values. The organic

complexing agent is preferably Rhodamine B which can be used in either oil-soluble or water-soluble form but can also be another organic complexing agent such as 5-(4-dimethylaminobenzylidene) (Rhodanine).

The activated material used to recover the gold from the organic complexing agent is suitably activated carbon or cellulose chaff, which may be regenerated for reuse as described above.

If desired an extracting solvent, such as methyl isobutyl ketone, and a gold complexing agent such as Rhodamine B may be used together.

The extraction, by any of the above methods, is preferably carried out by passing the pulp or leach liquor in cocurrent or countercurrent through a contactor in contact with a substantially immiscible liquid phase comprising an extractant for the metal values. A preferred contactor is of the solids-liquid bucket contactor type, for example as described in G.B. Patent Specification No. 1,145,894 and U.S. Specification No. 3,649,209. However the process may be carried out by other methods, such as the use of a vertical stirred and heated leach tank or a horizontal leaching vessel fitted with a screw conveyor or using an Akins classifier.

The invention will now be described by way of example with reference to the accompanying drawings and examples.

In the drawings, Figures 1 to 4 are flow diagrams of processes for recovering gold values in accordance with different embodiments of the invention.

Referring to Figure 1, gold-containing material to be processed is introduced on line 2 to a mixer 4 with a suitably low particle size of less than 0.5 mm. Dependent on the starting material, which may for example be a sand, slime or newly mined ore, preliminary

steps such as concentration, or steps to remove oversize material such as screening or the use of hydrocylones, may have been carried out. Leaching agent, suitably comprising nitric acid and a source of halide ions, is introduced to the mixer 4 on line 6. The mixture in mixer 4 is subjected to continuous thorough stirring and heating.

Mixture is withdrawn on line 8 via pump 10 and enters contactor 12 having first passed through deaerator 11, intended to assist in minimising problems with contaminant flotation in the contactor. Contactor 12 may suitably be of the solids-liquid bucket contactor type described in G.B. Patent No. 1,145,894 and U.S. Specification No. 3,649,209, but other contacting methods may be employed, such as a horizontal contactor with a screw conveyor or an Akins classifier. Activated carbon, cellulose chaff, or other suitable activated material, is introduced to contactor 12 on line 14 and preferably travels cocurrent with the mixture of leaching agent and gold-bearing material through the contactor. If desired, the streams of activated material and the mixture may be introduced at opposite ends of the contactor 12 and flow in countercurrent. The activated material suitably has a specific gravity of about 0.6 and not more than 1.2 when fully loaded. As the mixture of gold-bearing material and leaching agent initially will suitably have a specific gravity of about 1.3 to 1.8, the difference in specific gravities ensures that the activated material and ore/leaching agent mixture readily separate, so that each can be drawn separately from the contactor 12. Furthermore, the activated material is unaffected by the concentrations of nitric acid used in the process. The flow rate through contactor 12 can vary but a suitable residence time is from 2 to 6 hours. The contacting is

preferably carried out at a temperature of from 30°C (more preferably 50°C) to 95°C. Heating of contactor 12 can be carried out by any suitable means, for example by applying steam panels to the contactor 12.

It will be appreciated that, instead of carrying out both leaching and treatment with activated material in a single contactor 12, two contactors could be employed in series, in the first of which the heating agent and gold-bearing material are contacted, while liquor freed at least partly from solids is passed to a second contactor for contact with the activated material.

The activated material bearing the extracted gold values leaves the contactor 12 on line 16 and passes via line 31 to screen and liquor tank unit 18. The liquor passing through the screen is recycled on line 42 to the leaching agent feed to mixer 4. The activated material and gold values pass on line 44 to treatment unit 46 where the gold values are recovered by any suitable means, such as calcining, stripping with hydrogen cyanide or nitric acid, deposition or electrolytic processes. Gold values are withdrawn on line 20, while recovered activated material is recycled on line 22 to line 14. Alternatively unit 46 may simply act as a storage unit, gold recovery taking place off the plant. Make-up activated carbon, cellulose chaff or other active material is supplied on line 24. In an alternative process, the use of activated material in contactor 12 is replaced by the use in processing unit 18 of oxalic acid, used in stoichiometric excess to the gold present in the separated liquor, to deposit the gold.

The spent liquor and ore, sand or slime are withdrawn from contactor 12 on line 26. Any remaining carbon is drawn off via screen 28 and joined via line 30

with the carbon flow in line 16. Alternatively carbon on line 30 may be passed directly to treatment unit 46. The remaining material passes through desander 32. Spent pulp is withdrawn on line 34. Dependent on the nature of the waste and regulations governing its disposal, it may be desirable to add lime, or other neutralising agent, on line 36. The used leaching agent is recycled via line 38 to line 6. Make-up leaching agent is added on line 40.

Referring to Figure 2, gold-containing material to be processed is introduced, as ore or a slime slurry, on line 100 and passes through a 24 mesh screen 102 to give a suitably low particle size, e.g. less than 0.5 mm. The screened material passes on line 103 to thickener unit 104 (provided with an overflow line 134 to remove excess slurry water or brine) and thence on line 105 to vertically arranged heated leach tank 106. Recycled leaching agent, suitably comprising nitric acid and a source of halide ions is mixed via line 107 with the thickened material in line 103 prior to entry to leach tank 106. Make up nitric acid and halide are introduced to tank 106 on lines 141 and 142. The contents of leach tank 106 are subjected to intensive mixing and also heated, for example to about 90°C, by suitable means, such as the provision of a steam jacket (not shown) with steam introduced from boiler 108 on line 109. Under these conditions, the leach time is greatly reduced, for example in the range of 20 mins. to 3 hours.

The material from tank 106 passes via deaerator 110 (which may be dispensed with dependent on the materials employed) on line 111 to a horizontally arranged primary decanter 112, which is fitted with a plurality of weirs 113. Additional leaching agent and brine may be added to decanter 112 on line 114. Both

line 114 and decanter 112 may also be heated by steam provided from boiler 108 via steam supply line 115 and condensate return line 116. Solids removed by the action of weirs 113 is combined and withdrawn on line 117 to a neutralisation and discharge tank 118 and finally discharged from tank 118 on line 119. Neutralisation may be with lime or other neutralising agent dependent on the nature of the waste and regulations governing its disposal.

It is to be noted that, as the leaching is carried out at elevated temperature in tank 106, leaching proceeds rapidly with rapid settling of the solids from the leach liquor, so that, in certain circumstances, it may be possible to dispense with decanter 112 and withdraw solids and leach liquor directly from tank 106.

The leach liquor from decanter 112 leaves on line 120 and passes to a solids-liquid separator 121 which is of the settling type or may be an alternative type of solids-liquid separator. Solids is withdrawn on line 122 to neutralisation and discharge tank 118 for final discharge on line 119. The clear liquor from separator 121 passes on line 123 to heat exchanger 124 where it is cooled, for example to no hotter than 40°C for passage via line 125 to complexing vessel 126. Vessel 126 may take the form of a mixer settler, column or a Graesser bucket contactor or other type of contactor. The liquor is contacted in countercurrent fashion in vessel 126 with a supply of, for example, the commercially available oil soluble form of the complexing agent Rhodamine B, supplied from doser 127 via lines 128 and 129 in an organic carrier such as amyl acetate, methyl i-butylketone or the like, or a mixture thereof, which carrier is selected so as to have low solubility in aqueous-acid solution. The Rhodamine

B complexes specifically with the gold values which are then withdrawn on line 130, while discarded liquor leaves on line 131 for recycle.

The organic complex phase then passes to a recovery station where a suitable activated material is used to recover the gold values. In the form shown in the drawing, two fixed beds 132 and 133 of activated carbon are used, alternatively or consecutively. Recovered organic complexing agent is returned to complexing vessel 126 on line 129. The activated carbon with adsorbed gold values may be treated in any suitable way to recover the gold. It is to be noted that only relatively pure liquor reaches the activated carbon adsorption stage and so the carbon does not become clogged and has a long useful life and can be used to greater efficiency to adsorb large amounts of gold . It is estimated that gold values can be adsorbed at a level of up to 500 oz gold per ton of carbon. Further the activated carbon is not agitated, which aids clean separation after adsorption of the gold values. Further, the organic complexing agent readily separates therefrom. The consumption of organic complexing agent is relatively low as it can be efficiently recycled. Approximately 5 to 10 parts by weight of oil-soluble Rhodamine B per 1 part of gold chloride present in the liquor is suitably employed in the complexing stage.

While the drawing illustrates the provision of steam heating for leach tank 106 and decanter 112 from steam boiler 108, it may also be expedient to heat thickener unit 104 and/or to insulate all three vessels. Further, to minimise the energy requirements of the system, the brine and/or water overflow from thickener unit 104 is withdrawn on line 134 and passes to heat exchanger 124 where the liquid is heated by exchange with the gold-bearing liquor flow in line 123. The

heated brine and/or water passes on line 135 to a further heat exchanger 136 and is returned to slurry vessel 104 on line 140. Line 140 has a bleed-off stream 144 which may be used if there is need to reduce the volume of slimes in the system. Spent liquor on line 131 acquires some heat by passage through heat exchanger 136 and then passes via line 137 to heat exchanger 138 which is heated from boiler 108 and serves to heat the return liquor to the desired temperature for recycle on line 107 to the incoming thickened feed in line 105. Condensed steam leaves on line 139. Lines 135, 137 and 107 may be insulated to minimise heat loss.

Turning now to Figure 3, gold bearing material, having been subjected to suitable preprocessing as described above in connection with Figures 1 and 2, and suitably in the form of a pulp with brine, is introduced on line 220 to a stirred leaching vessel 222 to which is supplied an oxidising agent on line 224, suitably nitric acid. Alternatively other oxidising agents or mixtures of oxidising agents may be employed as described above. It may be possible to add gold bearing material, brine and acid directly to leach vessel 222 without preprocessing for easily leachable ores where problems in releasing the gold are not encountered. Thus with friable ores, high speed stirring of the leach vessel may achieve gold release, especially if a cleaning agent such as a surfactant is added, which also assists in deflocculating the particles. Leaching vessel 222 may suitably be lined with silicone elastomer or other materials to resist corrosion. It may be heated by any suitable means, such as by heating coils with steam or oil as heating fluid, to a temperature such as about 90°C so as to give a satisfactory leach time, for example from 20 mins. to 3 hours. The concentration of brine and oxidizing agent in leaching vessel 222 can

vary widely. For example, a suitable content of nitric acid is from 5 to 12% by weight of the aqueous medium, although this may be increased, for example up to 25%, if the ore being treated has an alkalinity or carbonate content which will cause it to neutralise some of the nitric acid being used. Likewise, the brine content may vary from very dilute to saturated, but is preferably in the range of from 5 to 30%. As an example, leach tank 222 may contain a solution of soluble inorganic chlorides (e.g. sodium, magnesium and potassium) in up to 15% nitric acid. The ore and leaching agent may be stirred for about 30 mins with a stirrer shaft speed of 5000 rpm.

Analyses of nitric acid and chloride contents in tank 222 may be carried out and previous analysis of the carbonate and sulphide content of the ore enable adjustment of the acid added to give a final leach acidity of 15% HNO_3. A typical mix in tank 222 is ore (dry) 6 kg, sodium chloride 2.5 kg and 15% nitric acid 18 litres.

The leached slurry passes from vessel 222 on line 226 to cooling tanks 228 where the slurry is cooled, suitably by heat exchange. Cooling tanks 228 may be lined with corrosion resistant materials such as polythene and polybutadiene and may be stirred. Preferably the slurry is cooled to a temperature of 45°C or less before being passed on line 230 to a contactor 232. Tanks 228 may act as settling tanks so that the material passed on line 230 is a rich liquor separated from residual ore. Contactor 232 is preferably a bucket contactor, for example, as described in G.B. Patent Specification No. 1145894 and U.S. Specification No. 3649209. However the contacting may be carried out by other equipment, such as an Akins classifier.

In contactor 232, the leach liquor is contacted

in countercurrent with a stream of solvent entering the contactor on line 234. As an example, a 6" bucket contactor may be used with a flow rate of 60 litres per hour and a rotation speed of 6 to 8 rpm. Extra cooling may be supplied by heat exchange with cold brine. Solvent may be supplied to the contactor at about 6 to 15 litres per hour. The preferred solvent is methyl isobutyl ketone, although other solvents may be employed, with or without the addition of a gold complexing agent, as described above. As a further example, the rate of liquid flow through the contactor may be of the order of 120 litres per hour with a volume ratio of leach liquor to solvent of about 10:1 and a retention time in the contactor of about 10 minutes. The methyl isobutyl ketone carrying extracted gold values leaves contactor 232 on line 236 and passes to solvent still 238 where the methyl isobutyl ketone is distilled by heating to its boiling point of 103 to 106°C. After addition of alkali to neutralise the acid content, gold precipitates as a powdered product. The gold can be extracted with a non-polar solvent from still 238. We have found the gold to be of high purity of at least 80% and usually of 95 to 100% purity. If desired, residual organic impurities or inorganic salts can be removed, for example by washing and/or calcining. As an alternative to distilling off the methyl isobutyl ketone solvent, the solvent may be heated, for example up to 90°C, at which temperature the gold is precipitated as a high purity fine powder.

The distilled methyl isobutyl ketone is liquified in condenser 240 and collects in solvent tank 242 for recycle to contactor 232 on line 234. This simple recovery of solvent by distillation is efficient and gives a low loss of solvent. If desired the methyl isobutyl ketone can be cleaned before recycle by any

suitable means, for example using zeolites, activated carbon, or exchange resins. Thus when, for example, an acidic leaching agent has been employed some acid may have been taken up by the methyl isobutyl ketone. This may be removed by distilling over an alkali such as dilute caustic soda or calcium hydroxide or by other suitable means. Other washing procedures may be necessary to clean the solvent from contaminants such as iron chlorides. Washing may be carried out in a second small contactor.

Stripped leach liquor leaves contactor 232 on line 244 and passes to leach residue settling tanks 246, or any alternative type of solids-liquid separator or settling ponds. Solids residue is withdrawn on line 248 to residue neutralisation tank 250 and final discharge on line 252. The recovered leach liquor from tanks 246 is recycled on line 254 to line 224 which supplies acid to leach tank 222. Make up acid is supplied from reservoir 256.

Figure 4 illustrates an alternative process for use with feedstock ores from which it is particularly difficult to release the gold values. In this modified flow sheet, gold bearing feedstock in brine passes on line 268 to leach tank 270 where acid such as nitric acid is added on line 272 to give leaching conditions within tank 270. Tank 270 is heated by suitable means (not shown). Leached pulp is withdrawn on line 274 to cooling tanks 276 where the temperature is reduced by suitable means such as heat exchange prior to passage of the pulp on line 278 to contactor 280 similar to contactor 232 shown in Figure 3. In the same way as described in Figure 3, the leached pulp is brought into contact in contactor 280 with a stream of solvent, preferably methyl isobutyl ketone fed to the contactor on line 282. The solvent carrying extracted gold values

is withdrawn on line 284 and the solvent distilled off in still 284 leaving gold product. Alternatively the gold is precipitated by heating the solvent below its boiling point. Solvent is condensed in condenser 288, and stored in vessel 290 for recycle on line 282.

To avoid high loss of gold values in the residual pulp from contactor 280, the pulp is led on line 292 to further reaction tanks 294, where it is further extracted with an additional solvent under alkaline conditions, suitably alcohol and sodium hydroxide. Separated liquor is recycled on line 296 to leach tank 270. Alcohol/water extract passes via lines 298 to distillation column 300 where the alcohol/water azeotrope distils leaving further residual gold. The solvent is condensed in condenser 302 and stored in tank 304 prior to its recycle (not shown).

Various modifications to the described process may be used to improve yield of gold values. Thus it has been found advantageous in the processes illustrated in Figures 3 and 4 to stir the leached pulp with about 0.005% by weight of a complexing agent specific for gold such as Rhodamine B, suitably with a retention time of 5 to 10 minutes, before contact with the methyl isobutyl ketone solvent, giving a very high transfer of gold freed in the leach to the solvent phase.

It will be appreciated that those vessels in which the leaching agent is employed are advantageously made of materials such as titanium which are resistant to the concentrations of leaching agent such as halide and nitric acid being conventionally used. If higher concentrations of leaching agent are employed, it is possible to employ vessels carrying a corrosion resistant ptfe coating or synthetic rubber at modest cost.

The highly oxidising leach is particularly suitable for the processing of metal-containing ore bodies where there is contamination with refractory materials, such as non-oxidised arsenic. Being in a totally enclosed vessel the oxidation does not release poisonous materials and these can be subsequently precipitated by known methods.

All or part of the apparatus is preferably enclosed to minimise problems with any potentially corrosive reactants. The recycling of leaching agent reduces these problems and minimises the cost of reagents. Clearly the process can be successfully operated on a continuous basis.

Desirably, gold recoveries of greater than 80%, preferably greater than 90%, are achieved.

The process of the invention can be carried out on a scale to suit a particular mining operation. Thus, for example, apparatus for carrying out the process may be scaled so that it can be mounted on a lorry. Such an apparatus can then be readily transported to the scene of operations and used, for example, to process slime heaps along a previously worked riverbed. Such an apparatus could process, for example, some 100,000 tons of ore per annum, dependent of course on the type of ore and residence times employed. Larger apparatus may be constructed on site, on the surface or in a mine cavern, having a contactor of for example, 2 to 3 metres diameter and a throughput of 250,000 to 800,000 tons of ore per annum, again dependent on the ore being processed and the conditions of processing. The throughput can also be increased, for example, by a factor of 5 to 1, or even 10 to 1, by suitable pre-processing of the feedstock ore, for example by separation and concentration processes.

The above description with reference to Figures 1 to 4 relates to the use of halide and nitric acid as leaching agent. We have found it to be particularly advantageous to employ a leaching agent where the oxidising agent comprises a mixture of acids such as a mixture of nitric acid and sulphuric acid, especially in conjunction with methyl isobutyl ketone as an extractant. This is illustrated in the following examples.

Example 1.

The starting material used was an ore of head grade 2.21g of gold per tonne (assayed by neutron activation) and 2.5g per tonne (by aqua regia assay). 100g of ore were mixed with 300g of an aqueous solution containing 10% by weight sulphuric acid, 5% by weight nitric acid, 17.7% by weight potassium chloride and 13.3% by weight sodium chloride. The mixture was vigorously stirred at 95°C for 30 minutes, allowed to cool and filtered. The residue was washed three times with water and the washings combined with the initial filtrate.

100 ml of the resulting solution were shaken at room temperature with 10 ml methyl isobutyl ketone for 3 minutes and then the layers allowed to separate. The methyl isobutyl ketone layer was subjected to analysis by atomic absorption spectroscopy. The analysis showed that the amount of gold extracted from the 100g of ore was 218 ppm, i.e. an extraction efficiency of 98.8%.

Example 2.

100g of an ore of head grade 3.96g per tonne of gold (assayed by neutron activation) was attrition milled for 15 minutes, then mixed with a solution containing 15% by weight nitric acid, 10% by weight sulphuric acid and 13.3% by weight sodium chloride. The solution was stirred at 93°C for 30 minutes, filtered

and washed as in Example 1 and then extracted as described in Example 1 with methyl isobutyl ketone. The gold extracted from 100g of ore was 314 ppm, i.e. an extraction efficiency of 79.3%.

Example 3.

In a run conducted on a larger scale, 3 kg of ore of head grade 33.8g per tonne was heated at 95°C with 9.2 kg of an aqueous solution containing 10% by weight sulphuric acid, 5% by weight nitric acid and 13.5% sodium chloride. The solution was stirred continuously and heated by recirculating the mixture through a titanium coil heated by an oil burner. The resulting slurry was cooled and then passed through a solids/liquids bucket type contactor in countercurrent to a stream of methyl isobutyl ketone. The ratio of slurry to methyl isobutyl ketone in the contactor was 10:1.

Analysis by atomic absorption spectroscopy of the methyl isobutyl ketone phase from the contactor showed that 27.7 ppm of gold had been extracted - an efficiency of 82%.

CLAIMS

1. A process for the recovery of gold and/or other noble metals from matter containing such metals, characterised in that said matter is contacted in substantially finely divided form with an aqueous leaching agent comprising a source of halide ions and an oxidising agent capable of enhancing the leaching action of the halide ions and the resulting soluble metal salts are extracted from the resulting pulp or leach liquor.

2. A process according to claim 1 characterised in that the extraction of metal values is carried out by passing the pulp or leach liquor in cocurrent or countercurrent through a contactor in contact with a substantially immiscible liquid phase comprising an extractant for the metal values.

3. A process according to claim 1 or 2 characterised in that the oxidizing agent is present in an amount of 2 to 15% by weight of the leaching agent.

4. A process according to any one of claims 1 to 3 characterised in that the oxidising agent comprises nitric acid.

5. A process according to claim 4 characterised in that the content of nitric acid is from 5 to 12% of the aqueous leaching agent.

6. A process according to any one of the preceding claims characterised in that the extraction of metal values is carried out using an activated material capable of removing metal values from the leach liquor, the activated material bearing the metal values is separated and the metal recovered therefrom.

7. A process according to claim 6 characterised in that the activated material is cellulose chaff or activated carbon.

8. A process according to any one of the preceding claims 1 to 5 characterised in that the metal is gold.

9. A process according to claim 8 characterised in that the extraction of gold values is carried out with an organic complexing agent specific for gold and the gold values are recovered from the complex so formed.

10. A process according to claim 9 characterised in that the complexing agent is Rhodamine B.

11. A process according to any one of claims 1 to 5 characterised in that the extraction of metal values is carried out with an organic solvent.

12. A process according to claim 11 characterised in that the solvent is methyl isobutyl ketone.

13. A process according to any one of the preceding claims characterised in that the contacting with leaching agent is carried out at an elevated temperature and the metal values are subsequently extracted at a lower temperature.

14. A process according to claim 13 characterised in that the leaching is carried out at a temperature in the range of 45° to 95°C and the extraction is carried out at a lowered temperature in the range of 20°C to 45°C.

15. A process according to any one of the preceding claims characterised in that the leaching agent

comprises a halide and a mixture of nitric acid with sulphuric acid.

16. Gold and/or other noble metals when recovered by the process of any one of the preceding claims.

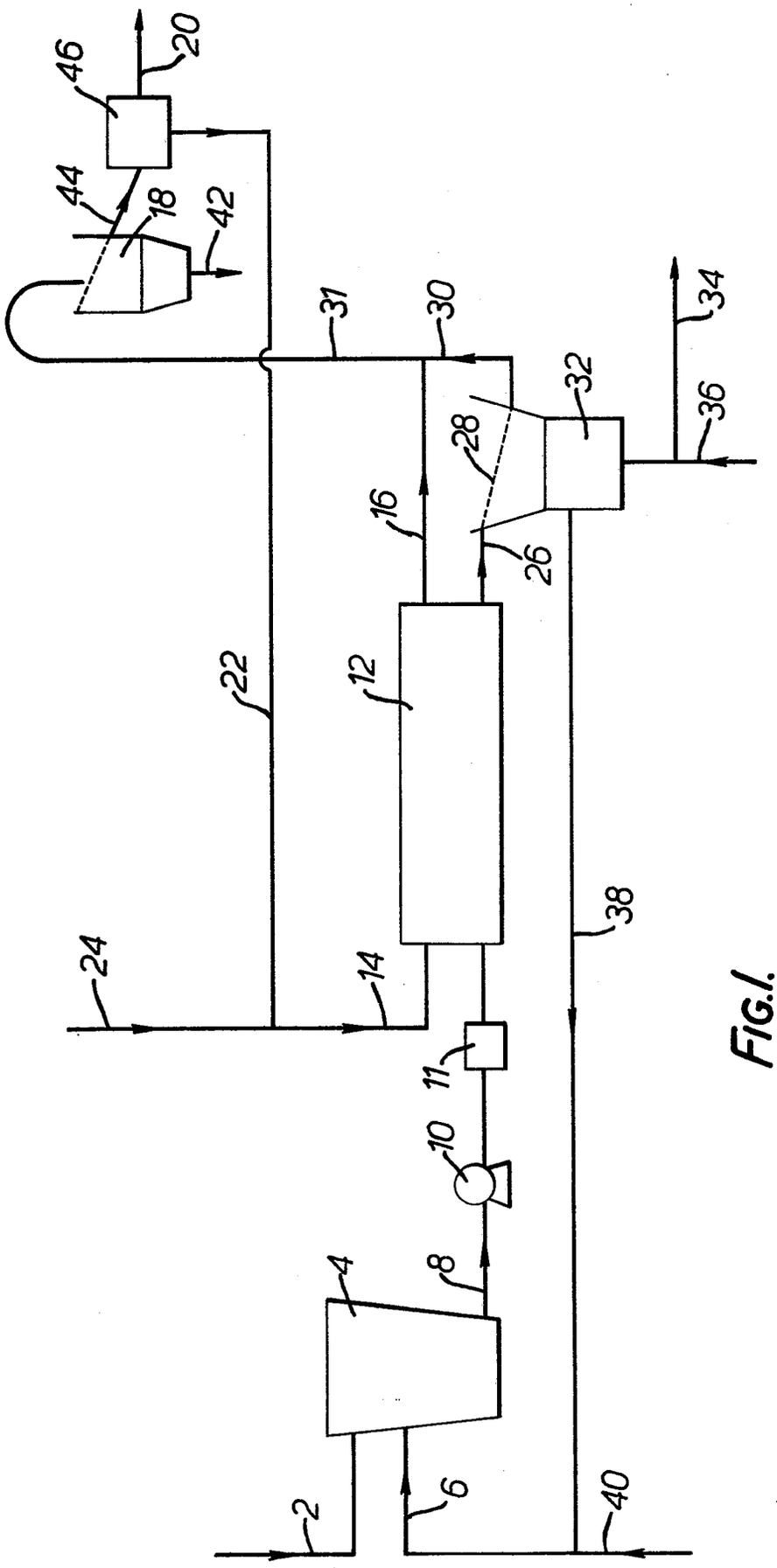

FIG.1.

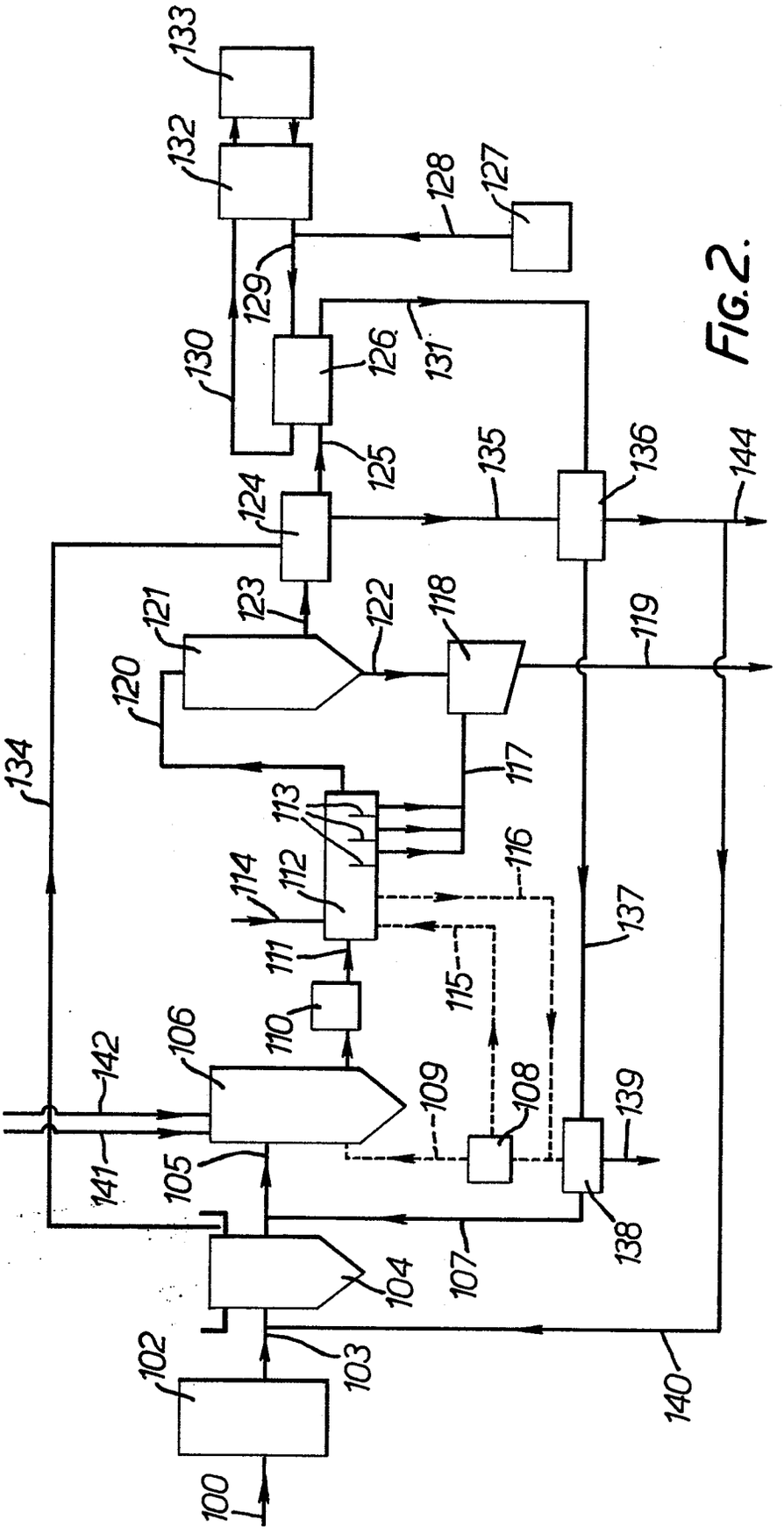
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FIG.2.

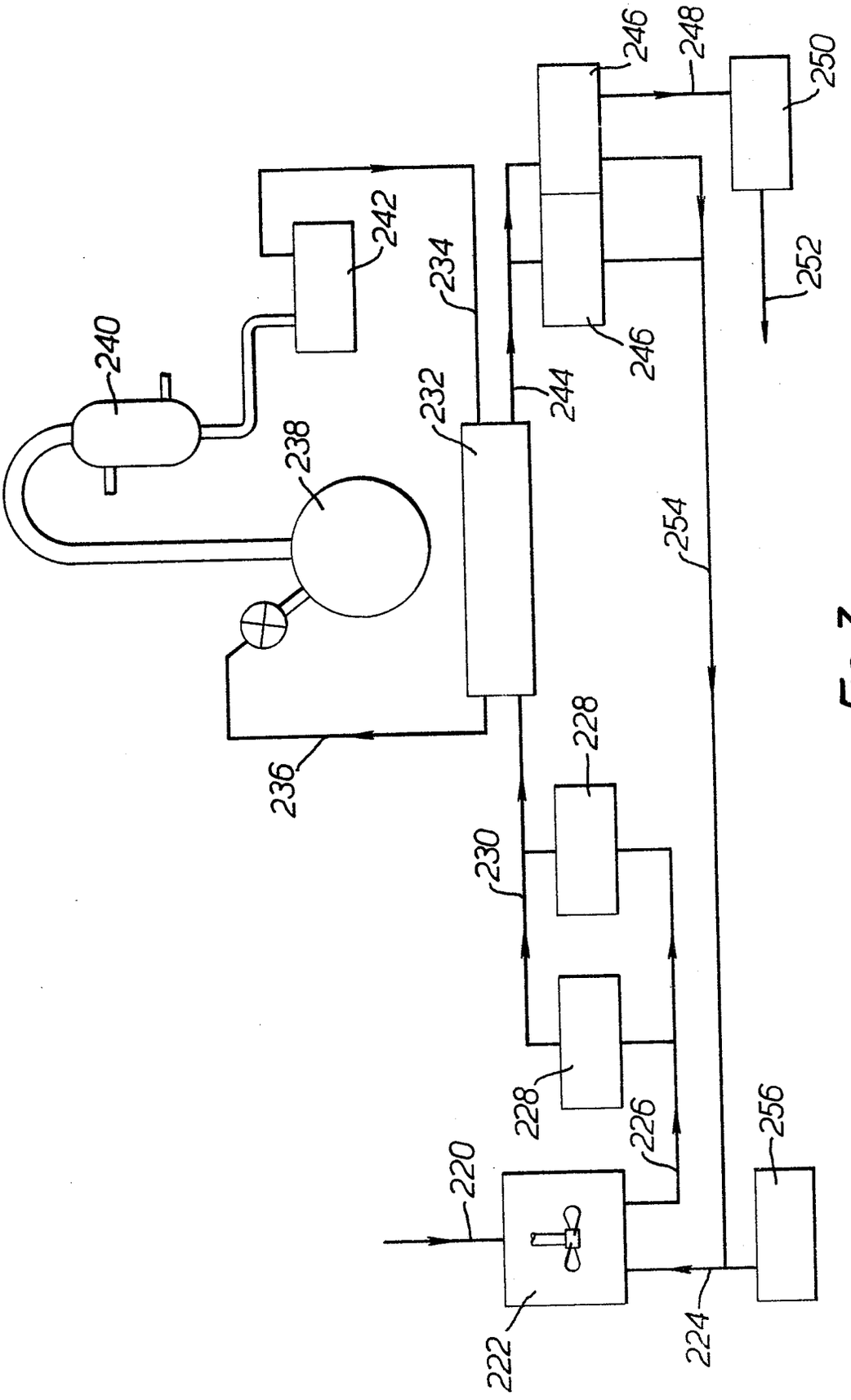
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FIG. 3.

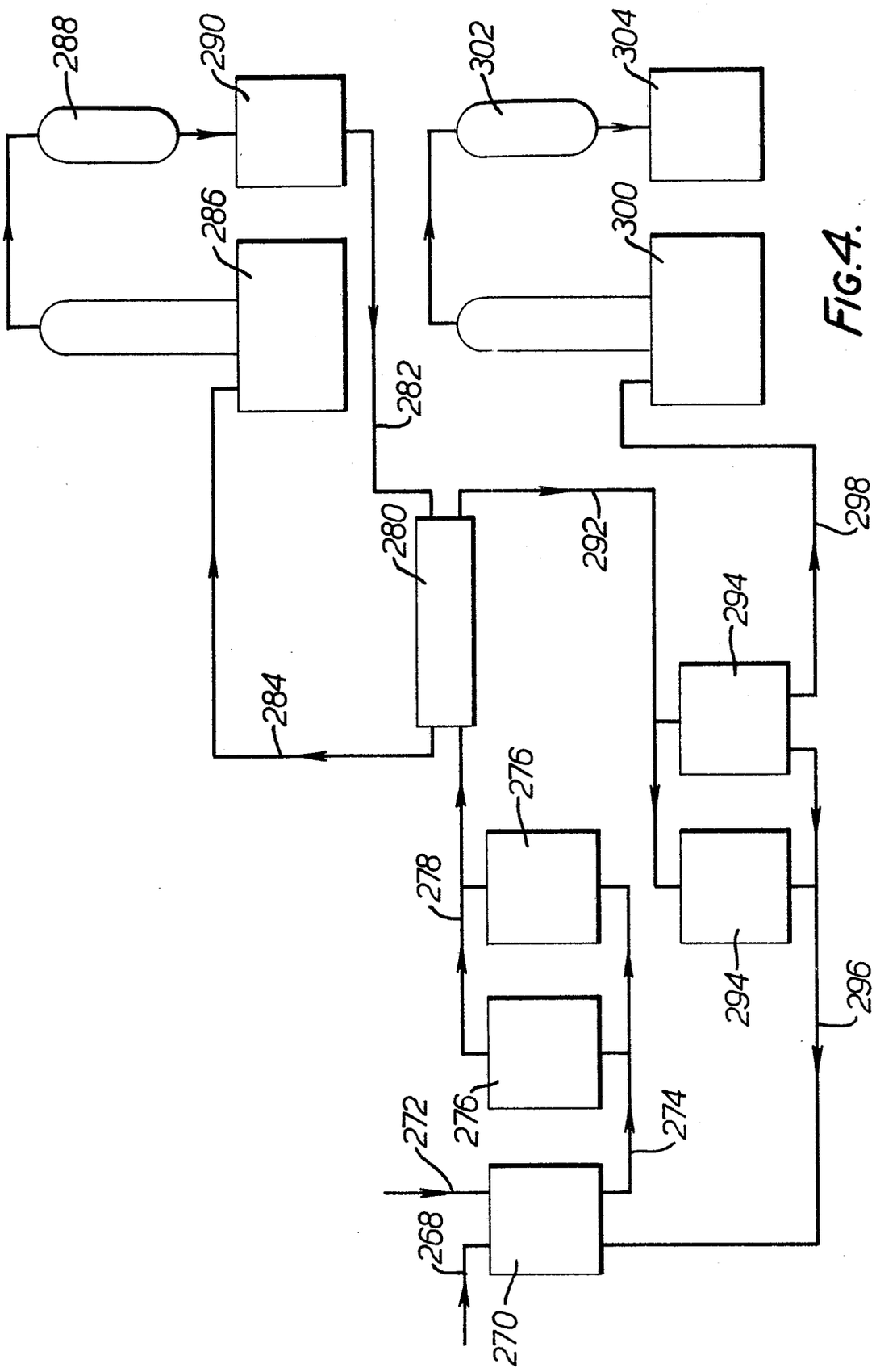
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FIG.4.

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EUROPEAN SEARCH REPORT

0124213

EP 84 30 1172

DOCUMENTS CONSIDERED TO BE RELEVANT

Category	Citation of document with indication, where appropriate, of relevant passages	Relevant to claim	CLASSIFICATION OF THE APPLICATION (Int. Cl. ³)
Y	DE-A-2 418 441 (H. CRONJÄGER) * Pages 10, 13 *	1	C 22 B 11/04
Y	DE-B-1 189 720 (EMPRESA AUXILIAR DE LA INDUSTRIA) * Column 3; claims 1, 2 *	1	
A	US-A-2 835 569 (F. REYNAUD et al.)		
A	GB-A-2 066 799 (ASAKA RIKEN)		
A	DE-C- 678 506 (H. SIBRALA)		
A,D	GB-A-1 145 894 (R. GRAESSER)		TECHNICAL FIELDS SEARCHED (Int. Cl. ³) C 22 B 11/04
A,D	US-A-3 649 209 (J. COLEBY)		

The present search report has been drawn up for all claims

Place of search	Date of completion of the search	Examiner
BERLIN	25-05-1984	SUTOR W

CATEGORY OF CITED DOCUMENTS

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