CO-TREATMENT OF LIMONITIC LATERITES AND SULPHUR BEARING MATERIALS AS AN ALTERNATIVE TO THE HPAL PROCESS

C.J. FERRON AND C.A. FLEMING — SGS

ABSTRACT

The high pressure acid leach process (HPAL) is being commercially used in four plants around the world to treat limonitic laterites. The two major operating costs of the HPAL process are the sulphuric acid and the energy required to heat the pulp to the desired temperature (>240°C). As an alternative to acid injection and steam heating, the in-situ generation of sulphuric acid and heat, produced during the high temperature pressure oxidation of sulphur-bearing materials blended with laterites, has been investigated at the laboratory scale. Various sources of sulphur have been examined, in particular, elemental sulphur, pyrrhotite and a Cu-Ni low-grade bulk concentrate. Under the proper blend ratios, all sulphur sources tested generated sufficient acid to result in equivalent nickel and cobalt dissolutions.

A significant improvement to the process economics can result from the addition of sulphur sources containing valuable by-products, such as nickel/cobalt and/or precious metals (gold and PGMs). This is especially true in the latter case, when the HPAL can be operated under PLATSOL™ conditions (i.e. with the addition of 5-10g/L of chloride), to dissolve the base metals and precious metals simultaneously in the autoclave.

Various examples of the application of this concept are presented, together with a brief discussion of its implication on the autoclave design and the downstream processes to recover the values from the leach solutions.

INTRODUCTION

The first commercial pressure acid leach (PAL) plant (commissioned in 1959) to treat limonitic nickeliferous laterites was that of Freeport at Moa Bay in Cuba [1]. In the late 1990s, three more PAL plants were erected in Western Australia to process local limonitic (Cawse) and nontronic (Bulong, Murrin Murrin) laterites [2,3,4]. All these plants have been well described in the literature. Several other PAL projects are well into final developments, and would feed off limonitic laterites from Australia, Indonesia, Philippines, New Caledonia, Madagascar and Brazil.

Two of the major operating costs related to the PAL process for treating limonitic laterites are the heat required to carry out the reaction at the required temperature (240-260°C presently, some proposed projects at 270-280°C), and the sulphuric acid required to dissolve the nickel and the cobalt. It is not intended to present a detailed analysis of the chemistry and the thermo chemistry of the PAL process here, as there are numerous papers dealing with that topic [5,6,7]; but we will concern ourselves with the acid consuming reactions.

The major element in a limonitic laterite is obviously iron. Since most of the nickel and part of the cobalt are present in solid solution within the goethite/limonite structure, all of the iron has to be reacted with acid to reach the desired extraction levels (≥ 90%) of nickel and cobalt, as per the simplified reaction (1):

$$2FeOOH + 3H_2SO_4 = Fe_2(SO_4)_3 + 4H_2O \qquad (1)$$
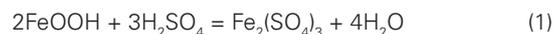
Under this scenario, nickel and cobalt are solubilized as per the simplified reactions (2) and (3):

$$NiO + H_2SO_4 = NiSO_4 + H_2O \qquad (2)$$

$$CoO + H_2SO_4 = CoSO_4 + H_2O \qquad (3)$$

SGS
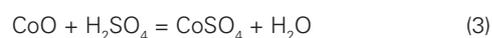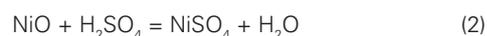
Stoichiometric acid consumption due to iron dissolution via equation (1) would be enormous (700-800 kg H_2SO_4/tonne or more for typical limonites); fortunately, at the high temperatures of the PAL process, iron hydrolyzes almost quantitatively, as per reaction (4):

$$Fe_2(SO_4)_3 + 3H_2O = Fe_2O_3 + 3H_2SO_4 \qquad (4)$$
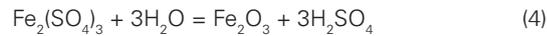
Therefore, under the conditions of the PAL process, acid consumption due to iron is quite small, and relates to the small amounts of basic iron sulphates/jarosites precipitated in the residue, and the small amount of iron sulphate not hydrolyzed.

The situation for magnesium is quite different. The dissolution of Mg will depend on mineralogy, but typically, 50-60% of the MgO in limonites reacts with acid, as per reaction (5):

$$MgO + H_2SO_4 = MgSO_4 + H_2O \qquad (5)$$
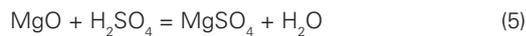
Magnesium sulphate does not hydrolyze under PAL conditions and, therefore, acid consumption due to magnesium is significant, which accounts for the limit put on MgO content for PAL users.

Aluminium in limonites also usually dissolves rapidly (reaction 6), but under the conditions of the commercial PAL plants hydrolyzes only partly [8], as per reaction (7):

$$2AlOOH + 3H_2SO_4 = Al_2(SO_4)_3 + 4H_2O \qquad (6)$$

$$3Al_2(SO_4)_3 + 14H_2O = 3Al_2O_3 \bullet 4SO_3 \bullet 9H_2O + 5H_2SO_4 \qquad (7)$$

Hydrolysis of aluminum sulphate to alunite regenerates some of the acid, but not all. Other elements, such as a manganese, chromium and silica consume acid, but to a lesser extent, due to their normally low concentration in the laterite.

The following are specific correlations have been presented to relate acid consumption with feed composition.

CVRD [9] presented the correlation as shown in equation (8):

$$\text{Net acid consumption (kg/t)} = 137.55 + 64.2\,(\%Al) + 33.99\,(\%Mg) \qquad (8)$$
$$\text{with } r^2 = 0.958$$

RNO [10] presented the following correlation (9):

$$\text{Net acid consumption (kg/t)} = 55 + 36.1\,(\%Mg) + 28.8\,(\%Al) \qquad (9)$$

Both of these correlations confirm that the acid consumption in treating limonitic/nontronic laterites is mostly related to aluminium and magnesium.

The published mineral compositions and acid consumptions for the commercial PAL plants [1,11] are presented in Table I.

Regarding heat balance, without going into any in-depth analysis, it is clear that the overall heat balance is negative, and the required heat has to be provided to the process. When acid is generated on site (acid plant, S burner), substantial heat is available to be re-injected into the process: the production of one tonne of H_2SO_4 theoretically generates 5.4 GJ of heat, the equivalent of 132 kg of fuel. In a conventional acid plant, about 70% of the heat is recovered in a usable form, but in a high-efficiency acid plant, heat recovery can reach $\geq 90\%$ [12,13].

Presently, heat is provided via direct steam injection for all four PAL plants, and this leads to significant dilution of the pulp.

The idea of generating heat and acid in situ within the laterite pulp by oxidizing sulphur compounds is not new. In the late 1960s and early 1970s, Republic Steel in the USA investigated the HSO process (hydrothermal sulphidization oxidation) to be applied to both limonite and saprolite laterites, as well as to sulphide concentrates. If only limonite laterites were treated, sulphur and pyrite could be added to the oxidation step (without prior sulphidization), which was carried out at 250°C under oxygen overpressure. The process was demonstrated in a 20 kg/h continuous pilot plant, and was deemed technically sound by Parsons Engineering [8,14].

1974 patents awarded to Inco [15,16] also promoted the idea of blending limonitic laterites with a sulphur compound of iron (pyrite, pyrrhotite, furnace matte, iron sulphate), sulphur, sulphur dioxide), and oxidizing the mixture with oxygen in an autoclave at high temperature.

More recently, the same concepts have come back into consideration. A 1996 Australian patent awarded to WMC extended Inco's process to include a variety of oxidized and sulphide Ni-Co ores; here again, sulphur addition was proposed as a heat and acid source [17]. The addition of pyrite as a heat and acid source has also been proposed for the Platsol treatment of highly oxidized PGM concentrates [18].

Although the idea has been around for almost 30 years, very little published data is available to allow a full re-evaluation of the concept of co-treatment of oxide/sulphide ores. Also, the possibility of conducting the co-treatment under Platsol conditions (i.e. at high temperature with a small addition of chloride) offers the interesting prospect of using precious metals (PM)- containing sulphur sources, which would allow their dissolution during pressure oxidation and their subsequent recovery from the leach solution rather than from the oxidized residue.

	PLANT			
	MOA BAY	CAWSE	BULONG	MURRIN-MURRIN
Feed				
% Ni	1.30	1	1.11	1.24
% MgO	0.9	2.6	7.7	6.6
% Al_2O_3	9.1	3.2	5.2	4.7
% Fe_2O_3	28.6	25.7	29.7	31.5
Acid				
Added (kg/t)	260	375	518	400
g/L in Discharge	30	35	35	20-35
Acid Production				
S burner	mostly	-	-	100%
Purchased H_2SO_4	some	100%	100%	-

Table I. Acid Consumptions in the Four PAL Plants

EXPERIMENTAL

SET UP

All the pressure leaching experiments were conducted in a 2-liter titanium Parr autoclave fitted with a dual axial-radial impeller assembly operating at 600 rpm. Under PAL conditions, acid was injected through a high-pressure diaphragm pump once the desired temperature was reached. Under pressure oxidation (POX) conditions, reagent grade oxygen was sparged into the pulp between the two impellers. For the co-treatment tests, the limonitic laterite and the sulphur-bearing sources were blended prior to being introduced into the reaction vessel, and the test was carried out as a typical POX test.

Reagents. Reagent grade acid and sulphur were used where required. Other sulphur sources were a commercial product obtained from a Canadian mill, and a concentrate produced during a pilot plant at SGS Lakefield. They are described in the next paragraphs.

Feed Description. Two samples of limonitic laterite were used during the testwork. Chemical analyses of the two samples are presented in Table II.

The limonite samples were composed primarily of Fe-oxyhydroxides (limonite/goethite) and amorphous alumino-silicates, with minor magnetite, chromite, talc and manganese wad.

Analyses of the pyrrhotite sample and sulphide concentrate used in part of the program are presented in Table III.

One of the sulphide samples was pyrrhotite flotation tailings from Falconbridge's Strathcona Mill in Sudbury, Ontario, Canada; the other was a low grade Cu, Ni, PGM concentrate from the Northmet property in Minnesota, USA.

It is important to note that the sulphide concentrates contained economically important levels of base and precious metals.

SAMPLE #	R-21	RT-B
ORIGIN	INDONESIA	PHILIPPINES
Assays (%)		
Ni	1.01	1.36
Co	0.098	0.110
Fe_2O_3	59.92	60.20
MgO	2.56	2.61
SiO_2	8.95	9.81
Al_2O_3	7.52	2.70
Cr_2O_3	1.28	2.83
MnO	1.99	0.84
CaO	0.10	0.08

Table II. Limonite Sample Assays

SAMPLE #	SM-R	DR-C
ORIGIN	ONTARIO, CANADA	USA
Assays (%)		
Fe	46.1	32.9
S^{2-}	25.5	26.7
MgO	1.91	1.99
Al_2O_3	3.31	-
SiO_2	13.02	-
Ni	0.94	3.05
Co	0.012	0.14
Cu	0.09	14.7
Au (g/t)	0.05	1.41
Pt (g/t)	0.30	2.22
Pd (g/t)	0.14	9.9

RESULTS

Two series of tests were conducted. The first series used elemental sulphur to generate heat and acid during the co-treatment; the second series used pyrrhotite or the sulphide concentrate.

CO-TREATMENT WITH ELEMENTAL SULPHUR

Proof of Principle Tests. To confirm the concept of generating sulphuric acid and heat by oxidizing elemental sulphur in an autoclave, 80 grams of sulphur were added to 1.5 liters of water and kept at 250°C for 60 minutes, while maintaining

690 kPa oxygen overpressure. Lignosol was added as a sulphur dispersant. Results indicated that only 15% of the sulphur had reacted, to produce a 25 g/L acid solution. Although the conversion was far from complete, the result was deemed positive and it was anticipated that sulphur dispersion and, as a consequence, the extent of the oxidation, would be much higher in a pulp than in water.

The next test involved adding elemental sulphur to a 25% solids pulp of limonite R-21 and oxidizing it for 60 minutes at 250°C under 690 kPa oxygen overpressure. The amount of sulphur added (171 kg/t) corresponded to 513 kg H_2SO_4/tonne of laterite, if one assumes that sulphur conversion to acid was complete. Lignosol was also added, at a dosage of 17.1 kg/t laterite (10% of the elemental sulphur).

Results are summarized in Table IV.

	Ni	Co	Fe	Al	Mg	Mn	S	H_2SO_4
Final PLS (g/L)	3.4	0.33	13	8.2	2.1	2.7	46.5	35
Final Residue (%)	0.05	0.003	44.2	1.66	1.06	0.02	2.86	-
% Extraction	95.8	97.3	9.3	62.1	40.1	97.7	-	-

Table IV. Nickel and Cobalt Extraction During Co-Treatment of Laterite R-21 and Elemental Sulphur (Temp = 250°C; 690 kPa oxygen; 60 minutes; 171 kg/t S; 17.1 kg/t Lignosol)

Co-Treatment Process Optimization. Based on the results presented above, the major parameters of the process were optimized. These include: elemental sulphur addition, Lignosol addition and oxygen overpressure. Temperature and retention time were kept constant at 250°C and 60 minutes, respectively. The effect of sulphur addition (equivalent to acid addition) on metal extractions is presented in Figure 1, while the effects of Lignosol addition and oxygen partial pressure are presented in Figures 2 and 3, respectively.
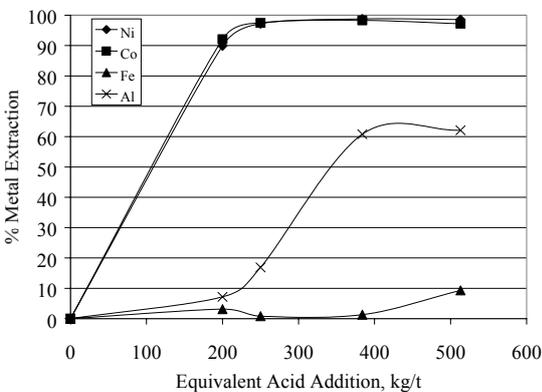
Figure 1. Effect of elemental sulphur addition on metal extraction during co-treatment (690 kPa oxygen; Lignosol = 10% of elemental sulphur)

The results in Figure 1 indicated that the optimum (equivalent) acid addition was between 200 and 250 kg/t: too low additions (200 kg/t) led to incomplete Ni/Co extractions, whereas too high additions led to excess acid consumptions due to aluminium being only partially hydrolyzed.

For the remainder of the test program, an (equivalent) acid addition of 236 kg/t laterite was selected.

It is well known that the presence of elemental sulphur in a pressure oxidation vessel (i.e. for refractory gold ores) can lead to incomplete sulphide oxidation. Because of that, it had been assumed that the addition of a sulphur dispersant would be necessary. The results presented in Figure 2 indicated that complete sulphur oxidation could be achieved in a batch autoclave at 250°C without Lignosol.

Results in Figure 3 clearly show that conversion of elemental sulphur to acid is strongly favored at 250°C, even with only 175 kPa oxygen overpressure.

CO-TREATMENT WITH PYRRHOTITE
Pyrrhotite, nominally FeS, is found in significant quantities in pentlandite deposits. Its rejection during mineral processing to produce nickel concentrates is a major concern for all nickel operators that treat pentlandite concentrates in smelters. On the one hand, complete rejection of pyrrhotite to the tailings always results in lower nickel recovery to the concentrate. On the other hand, pyrrhotite in the concentrate is undesireable because it generates additional SO_2. Pyrrhotite rejects are produced at all the major nickel sulphide producers, such as Inco, Falconbridge, Norilsk, and WMC.

Pyrrhotite can contain significant quantities of nickel (0.5-1%) and cobalt in solid solution, in addition to PGM values. Such high values have justified the development by Norilsk of a hydrometallurgical process to treat their pyrrhotite.

Because they are generally waste material that often contain significant nickel (and PM) values, pyrrhotites would seem to be an obvious target for co-treatment of limonites.

Preliminary Tests. A few preliminary tests were conducted to determine the acid requirement for the PAL treatment of limonite RT-B, and to confirm the pressure oxidation response of pyrrhotite sample SM-R.

PAL results for the limonite with the addition of various amounts of sulphuric acid are presented in Figures 4 and 5, while oxidation results for the pyrrhotite sample (treated alone) are presented in Figure 6.
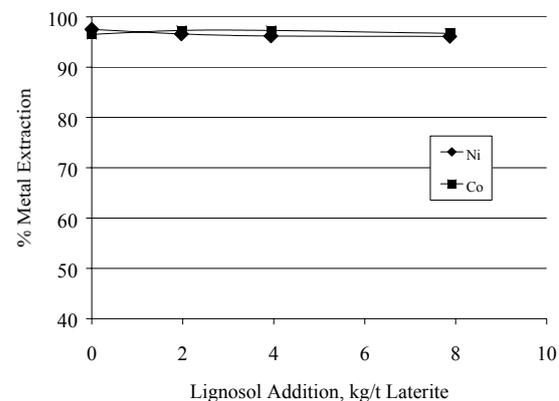
Figure 2. Effect of Lignosol addition on metal extraction during co-treatment (690 kPa oxygen; 236 kg/t acid equivalent)
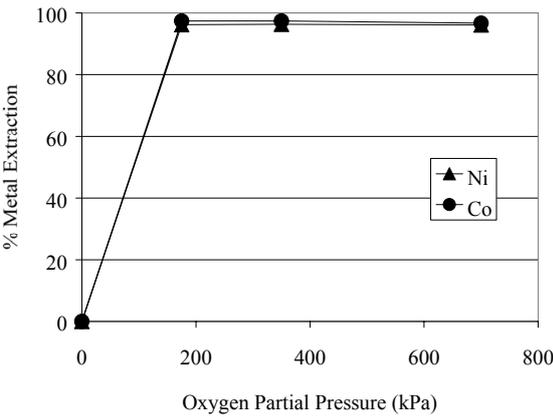
Figure 3. Effect of oxygen partial pressure on metal extraction during co-treatment (236 kg/t equivalent acid; 7.87 kg/t Lignosol)
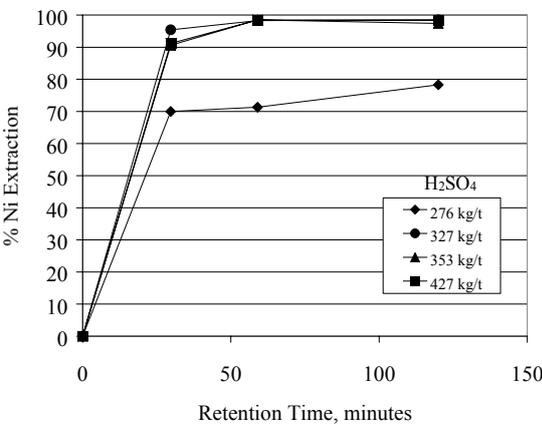
Figure 4. Effect of H_2SO_4 on nickel extractions during PAL treatment of limonite RT-B (Temp = 250°C, 27% solids)

The results in Figures 4 and 5 showed that a residence time of 60 minutes is sufficient for maximum dissolution of nickel (cobalt results were very similar), while an acid addition of 327 kg/t yielded maximum nickel dissolution and very efficient aluminium hydrolysis (Figure 5).
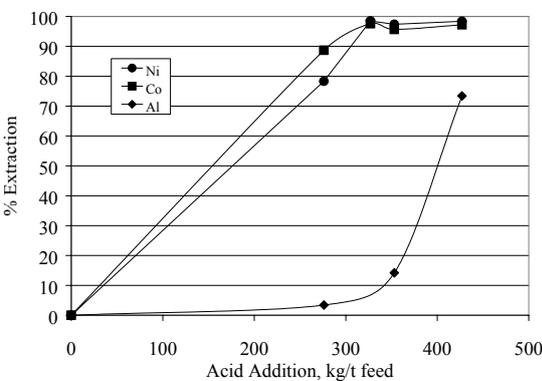
Figure 5. Effect of acid addition on nickel, cobalt and aluminium dissolutions during PAL treatment (Temp = 250°C, retention time = 120 minutes)

As expected, the oxidation of pyrrhotite (as measured by nickel extraction) under POX conditions at 250°C is extremely rapid, and sulphide oxidation for all practical purposes was complete in about 30 minutes.

Several co-treatment tests were thereafter conducted to confirm its technical feasibility. For these tests, temperature, retention time and oxygen overpressure were kept constant at 250°C, 120 minutes and 690 kPa, respectively. The main variable was the amount of pyrrhotite in the blend, hence the equivalent acid addition. Results are presented in Figure 7.

Overall nickel and cobalt extractions as a function of equivalent acid addition are presented in Figure 8.

Several comments can be made about the results presented above. First of all, the co-treatment was shown to be effective, and acid generated by the oxidation of SM-R pyrrhotite was effective in dissolving the nickel and the cobalt from both the limonite and the pyrrhotite. Comparing the results from Figures 6 and 8, it appears that the optimum equivalent acid addition is slightly higher than the optimum actual direct acid addition. This implies that pyrrhotite conversion under co-treatment conditions is slightly lower than with pyrrhotite alone. This fact was shown by analysis of residual sulphide sulphur in the final autoclave residue, which confirmed that sulphide sulphur oxidation was in the 95-97% range.

Co-Treatment under Platsol® Conditions.

The Platsol process has been well described in the open literature [19]. It is basically a high temperature pressure oxidation process that allows simultaneous extraction of base and precious metals, promoted by the addition of small amounts of chloride during oxidation. Therefore, if a co-treatment process is applied under Platsol conditions, the possibility exists to also dissolve the precious metals (gold + PGMs) present in the autoclave feed during the leaching stage.
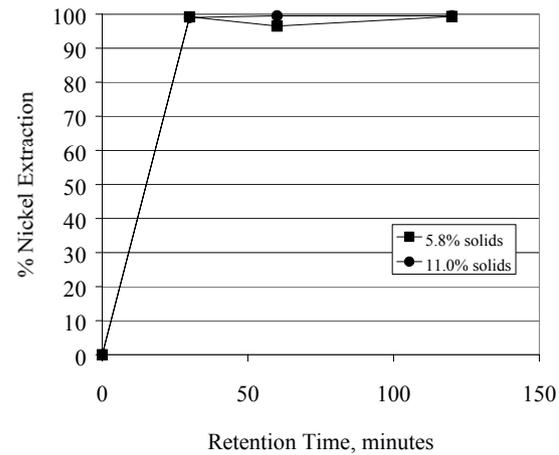
Figure 6. Nickel extraction during POX treatment of pyrrhotite SM-R (Temp = 250°C; 690 kPa oxygen)
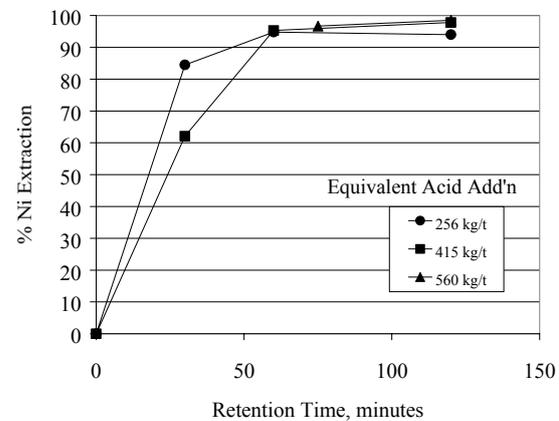
Figure 7. Nickel extraction during co-treatment of limonite RT-B with SM-R pyrrhotite (Temp = 250°C, 690 kPa oxygen, 26% solids)
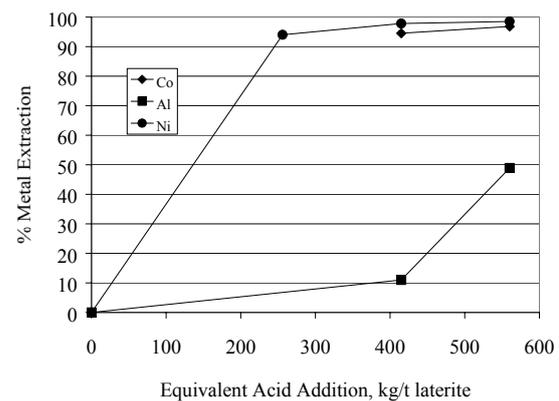
Figure 8. Nickel, cobalt and aluminum extractions during co-treatment of limonite and pyrrhotite (Temp = 250°C, 120 minutes, 690 kPa oxygen, 26% solids)

Pyrrhotite contains nickel and cobalt and copper in solid solution, in addition to fine inclusions of chalcopyrite and pentlandite. Pyrrhotite may also often contain various amounts of PGMs; for example at Norilsk [20]: rhodium can range from 0.11 to 1.6 g/t, iridium from 0.05 to 0.25 g/t, ruthenium from 0.10 to 0.45 g/t, osmium from 0.07 to 0.25 g/t,

palladium up to several tenths of a gram per tonne and platinum can reach up to 1 g/t. At an average value of US$350/oz of PGM, a concentration of 1 g/t PGM is equivalent to 0.22% Ni (at US$3.00/lb), and is, therefore, quite significant.

It is also well known that some laterite deposits also contain significant platinum values, and a platinum recovery circuit was included in the Syerston pre-feasibility study [21]. The Platsol process was shown to be very effective in recovering platinum from the Syerston laterite in Australia, and 97% of the platinum was dissolved from a limonite assaying 0.76 g/t Pt, when treated under Platsol conditions [19].

One co-treatment test was conducted on a RT-B limonite blended with the Cu-Ni sulphide concentrate containing significant PGM values (see Table III). The co-treatment test was conducted at 250°C and 27% solids, for 120 minutes under 690 kPa oxygen overpressure, with 10 g/L NaCl added. Results are presented in Table V.

	Ni	Cu	Pt	Pd	H_2SO_4
Feed (%, g/t)	1.84	5.1	0.83	3.68	-
PLS (mg/L)	6,160	17,700	0.23	1.16	23
Residue (%, g/t)	0.06	0.08	0.22	0.46	-
% Extraction	97.3	98.7	78.3	89.7	-

Table V. Co-Treatment of Limonite/Sulphide Concentrate under Platsol Conditions

Results confirmed the excellent extraction of nickel from both the laterite and the sulphide concentrate, as well as the other base metals, copper (and cobalt) from the sulphide concentrate. The extractions of PGMs from the sulphide concentrate during Platsol co-treatment, were also good considering this was a once-off test without optimisation.

DISCUSSION

The results presented in this paper confirm that various sulphur-containing materials blended with limonitic laterites can readily be oxidized to completion at 250°C. The oxidation of the sulphidic portion of the blend generates in-situ the heat and the acidity required for the desired reactions.

One should now consider the various practical and safety implications of the co-treatment process. On the operation itself, a major factor is that the PAL process has to operate under oxygen overpressure. At 250°C, the water vapor pressure is 3880 kPa. The results of tests with elemental sulphur co-treatment indicated that, at this temperature, 175 kPa oxygen overpressure was sufficient to fully oxidize elemental sulphur in one hour, and provide all of the heat/acid required to dissolve the nickel and cobalt. This would bring the total autoclave pressure to 4055 kPa. One of the WA laterite plants autoclaves is rated to operate at 260°C (4595 kPa pressure), and Inco's Goro autoclaves will be operated at 270-280°C (5410-6320 kPa), and these are the pressures at which the demonstration plant (one fiftieth of the full size plant) was operated at length in New Caledonia. Based on that, it can be assumed that the engineering aspects of operating a co-treatment autoclave around 4055 kPa are manageable.

As far as materials of construction are concerned for the Platsol option, there are two choices for handling oxygen, chlorides and acid simultaneously, at temperatures of up to 250°C. The one is titanium clad vessels, and the other is the use of high temperature polymer/elastomer membranes, which are positioned between a conventional steel shell and the brick lining (as a substitute for lead lining). All three Western Australian laterite operations are using Ti-clad autoclaves because of the high chloride content of the ground water in the region, although these plants were not designed for oxygen service. In this regard, from a safety perspective, it should be borne in mind, that the

percent oxygen in the gas phase at 175 kPa overpressure and 250°C is only 4.3%, under non-upset conditions. As to the membrane option, there has been significant progress and commercial development of these materials recently. Such types of membrane have already been in commercial use in the Macraes autoclave in New Zealand, and the Phelps Dodge autoclave in Bagdad, (although at lower temperatures, 225-230°C), and have been tested in PAL autoclaves in WA.

On the reagent side, oxygen consumption under the co-treatment scheme could be significant. If one considers the three oxidation reactions for complete conversion of elemental sulphur, pyrrhotite and pyrite to sulphate:

$$S + \tfrac{3}{2}O_2 + H2O \rightarrow H_2SO_4 \qquad (10)$$

$$2FeS_2 + \tfrac{15}{2}O_2 + 4H_2O \rightarrow Fe_2O_3 + 4H_2SO_4 \qquad (11)$$

$$2FeS + \tfrac{9}{2}O_2 + 2H_2O \rightarrow Fe_2O_3 + 2H_2SO_4 \qquad (12)$$
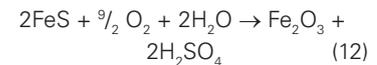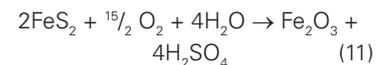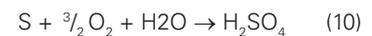
Based on the stoichiometry, these reactions correspond to oxygen requirements of 1.5 kg/kg S, 1.875 kg/kg S and 2.25 kg/kg S, respectively. Oxygen consumption for pyrite and pyrrhotite is higher, because sulphide has to be oxidized to the plus 6 state from the minus 2 state (FeS), or from an average minus 1 state (FeS.S); also, iron has to be oxidized from the plus 2 to the plus 3 state. Oxygen is the major reagent consumed in the co-treatment flowsheet, and its cost would need to be offset against the costs of acid and heat in the conventional process.

Co-treatment of sulphur compounds will also have an effect on the percent solids in the autoclave feed and discharge. When using elemental sulphur, complete oxidation of the sulphur generates only acid, which will not affect the amount of solids in the autoclave discharge. When using other sulphur compounds, such as pyrite or pyrrhotite, the amounts of solids in the autoclave feed will increase, and so too will the final autoclave discharge solids, because of the hematite generated from the oxidation. Although the rheological properties of

	PAL		Co-treatment		
	H$_2$SO$_4$ (PURCHASED)	S BURNER	S^o	FeS$_2$	FeS
Equipment					
Autoclave	yes	yes	yes	yes	yes
Acid Plant	no	yes	no	no	no
Boiler	yes	no	no	no	no
Steam Injection and Heat Recovery	yes	yes	no	no	no
Acid Injection	yes	yes	no	no	no
Oxygen Plant	no	no	yes	yes	yes
Downstream Plant	larger	larger	smaller	smaller	smaller
Additional Recovery Plant (Cu, PMs)	no	no	no	yes	yes
Power Co-generation	no	yes	no	no	no
Consumables					
H$_2$SO$_4$ (kg/t laterite)	333	0	0	0	0
S (kg/t)*	0	123-159	111	0	0
Steam	yes	no	no	no	no
O$_2$ (kg/t laterite)**	0	0	208	260	312
FeS or FeS$_2$ Concentrate	no	no	no	yes	yes
Revenue Generation⁕					
Ni (lb/tonne laterite)	26.4	26.4	26.4	34.15	34.15
Cu (lb/tonne laterite)	0	0	0	7.7	7.7
PM (oz/tonne laterite)	0	0	0	0.012	0.012

* Assume 70-90% co-generation heat recovery for acid plant

**Assume 80% oxygen utilization

⁕ Assume 1.2% Ni limonite (95% extraction) and 1% Ni, 1% Cu, 1g/t PM with 95% extractions for the FeS, FeS$_2$ cases

Table VI. Various Options: PAL versus Co-treatment

the co-treatment blends were not measured precisely, it was observed during testing that the addition of sulphur compounds to the limonite did not qualitatively worsen the behaviour of the blend; nevertheless, if the rheology of the blend becomes an issue, the two components of the blends could be added separately into the autoclave, since the rheological properties improve significantly at higher temperatures. This will likely not be an issue.

There are also implications on the downstream processing plant and unit operations. In the conventional process, heating of the autoclave to 250°C is by steam injection and, because of this, the PLS coming out of the autoclave is significantly more dilute then the case where heat would be provided via indirect heating or in-situ generation. Therefore, the volume of all the downstream unit operations (CCD, precipitation, solvent extraction, final tailing) are increased (22). The effect of pulp dilution because of steam injection on the size of the autoclave itself, the feed pumps and the heat exchange equipment has been recently quantified (23). Pulp dilution also increases acid consumption, if one operates at a given final acidity.

With the co-treatment process, the amounts of solids discharged would be the same (if elemental sulphur is used) or only slightly increased (with FeS or FeS$_2$). In any case, the increased volume due to additional solids would be far less than the increased volume due to dilution of the PLS because of steam injection; the increased solids volume only slightly affects the CCD circuit and the tailings, but not the precipitation and the SX circuits.

Any value metals (Cu, Ni, Co, PMs) in the sulphur compounds added, except for the elemental sulphur, will require additional recovery capacity (Ni, Co), or additional unit operations such as, for example, sulphide precipitation/thickening/filtration (for Cu, PMs), or possibly ion exchange directly from pulp. On the other hand, significant additional revenues would be generated. Table VI compares qualitatively the various process options.

If the flowsheet involving PAL with a S burner is the base case economically, the co-treatment using elemental sulphur does not bring additional metal values, and a decision between the two flowsheets would be based on a comparison of the capital and operating costs of an acid plant versus those of an oxygen plant, with the deciding factor being possibly the size of the downstream plant, significantly smaller for the co-treatment case. For co-treatment with iron sulphides, similar comparisons can be made, but in this case, significant additional metal values can be recovered, if present in the iron sulphide product.

Finally the S burner waste heat generates power to use in the plant, while in-situ generation of acid and heat generates only low-temperature steam. The implications of implementing the co-treatment process can be shown as illustrated in Figure 9.

A. Typical PAL process (simplified)
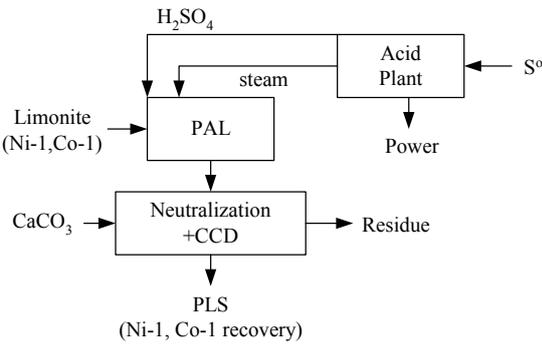
B. Proposed co-treatment process (simplified)
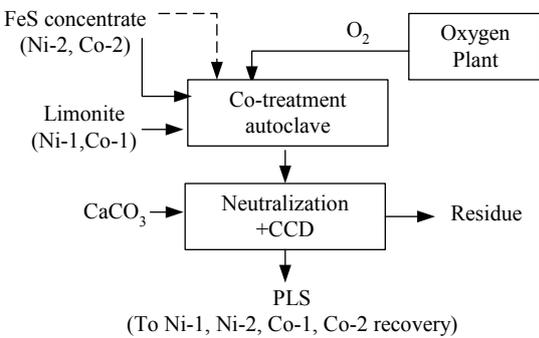
C. Proposed co-treatment process Platsol conditions (simplified)
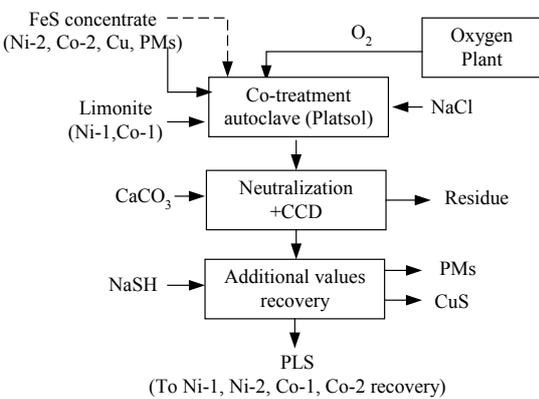
Figure 9. Schematic comparison of flowsheets for PAL versus co-treatment

As can be seen, the process flowsheets differ little, with the exception of the choice that must be made between an acid plant and an oxygen plant, and additional unit operations for copper and precious metals recovery, if the co-treatment plant is operated under Platsol conditions.

CONCLUSIONS

Co-treatment of limonitic laterites with various sources of sulphur has been shown in the laboratory to be technically feasible: sulphur oxidation was nearly complete and the acid (plus heat) generated in-situ was shown to be adequate to extract nickel and cobalt values from the limonite. At a first glance, the comparison of PAL versus co-treatment, appears to come down to a choice between an acid plant and an oxygen plant. However, a closer look indicates that there is more to it than this: a significant downstream plant size reduction will result from not having to inject steam directly into the pulp (less dilution) in the co-treatment case; moreover, any nickel, cobalt (and copper) in the sulphur compound would also be recovered, adding to the overall revenues. Finally, any precious metals (gold + PGMs) present in the sulphur compound, or in the original laterite, would also be extracted, if the co-treatment is operated under Platsol conditions.

The availability of both limonitic laterite and sulphur compounds in close proximity is unlikely and, therefore, transportation costs could be part of the overall economic equation. Similarly, local availability of cheap power or excess oxygen might have to be considered. To take into account all these project-specific factors, the economic comparison of PAL versus co-treatment must be conducted on a project-by-project basis.

Further technical development of the co-treatment process should examine the apparently beneficial effect of co-treatment on the feed pulp rheology, since this could also have a positive impact on the project economics.

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CONTACT INFORMATION

Email us at minerals@sgs.com
WWW.SGS.COM/MINERALS

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