

Capital-efficient upgrading of PGM concentrates

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Platinum group metals (PGM) production will continue to be driven by demand from the automotive industry. Their use is changing from auto-catalysts for internal combustion engine driven vehicles to fuel cell catalysts in electric vehicles. Furthermore, higher PGM loadings are seen in the auto-catalysts within hybrid vehicles and in vehicles complying with new emission standards being implemented in different jurisdictions around the world.

Existing and new platinum producers are thus investing in greenfields and brownfields projects to increase production at improved energy and capital efficiencies. The ISASMELT™ and Albion Process™ technology has been successfully applied to base metal feeds and specifically to nickel-copper sulphide smelting in Western Australia, with the Radio Hill plant achieving design capacity within three months of commissioning. The ISASMELT™ process can produce low iron nickel-copper matte directly from concentrates, suitable for a PGM base metal refinery, and thus obviate the need for a primary electric arc smelting furnace. This is particularly applicable to PGM concentrates with high sulphide contents.

The Albion Process™ is a simple, robust atmospheric leach process which is used to oxidise sulphide minerals for subsequent recovery of valuable metals. The Albion Process™ plant at Sable, Zambia has been used to successfully leach copper concentrate containing refractory chalcopyrite and carrollite minerals. The Albion Process™ technology uses ultra-fine grinding to prevent surface passivation in the oxidative leach, and highly efficient oxygen gas injectors for maximising mass transfer, to ensure reliable operation able to meet production goals with variable feed inputs.

This paper considers a flowsheet in which these technologies are used alongside widely installed technologies such as precipitation and copper electrowinning to process a typical high sulphide PGM concentrate to produce a high grade PGM alloy suitable for processing at an existing precious metals refinery, LME grade copper cathode and nickel mixed hydroxide product. The capital and operating costs for the flowsheet are considered for concentrate feed rates of 250,000 tonne per year and 800,000 tonne per year and is evaluated against existing and emerging flowsheets.

INTRODUCTION

With an increasing awareness of the impacts that mining and processing of primary ores has on the environment and society, the industry is rapidly adopting methods and technologies to prevent or minimise these impacts. In producing PGMs, the largest impacts occur from the use of high carbon electricity, fossil fuel transport of ores, concentrates and intermediates, the production of unstable residues, complex and toxic reagents and aggressive processes leading to safety concerns. In order to remove or minimise these in the PGM production process, new technologies and solutions are required. These can be adapted from other commodities and processes or through new innovations.

Due to the similarities with other sulphide ores, it is expected that a number of technologies and innovations will be able to be adapted from these other commodities. The South African PGM-bearing reefs are Merensky, UG2 and Platreef with the PGMs associated with base metal sulphides. The beneficiation steps for South African producers are crushing, grinding and flotation. This is followed by smelting, base metal refining and PGM refining. Merensky ore contains up to 3% base metal sulphide and 0.1% chromia (Cr_2O_3) and UG2 ore contains 0.3% base metal sulphide and up to 30% chromia. Platreef ore is a high sulphide, low-chromite ore, with similar sulphide and chromia contents to the Merensky ore (Jones, 2005). The PGM content of the reefs differ and hence the grade of the flotation concentrates differ. Merensky concentrate contains 100 to 150 g/t, UG2 concentrate contains up to 300 g/t (Jones, 2005) and Platreef concentrate contains 85 g/t (Ivanhoe, 2020).

In this paper, a flowsheet is presented to treat the lower-grade, base metal-rich, low chromite Platreef concentrate. A typical feed composition to the process is provided in I. The flowsheet seeks to minimise some of the negative impacts associated with the processing of primary ores. Specifically, the flowsheet seeks to reduce the consumption of high carbon electricity and use simple technologies and reagents. This is achieved by the inclusion of the ISASMELT™ top submerged lance (TSL) for smelting and the Albion Process™ for base metal refining.

Table I. Chemical composition of feed¹

3PE+Au (g/t)	Cu (%)	Ni (%)	Total S (%)	MgO (%)	Fe (%)
85	4.7	7.4	18	13	24

1. Chemical data obtained from locked cycle flotation tests, Ivanhoe 2020

The ISASMELT™ (TSL) technology was developed at Mount Isa Mines (now part of Glencore) during the early 1980s. It was identified that the smelter was running old technology, with new technology required to significantly reduce the energy consumed in the smelter. The furnace technology was first tested in a 250 kg/h test rig in the 1980s (Arthur and Alvear, 2011). The furnace technology was subsequently scaled up with the operation of a demonstration plant and finally a full-scale furnace. The technology resulted in 93% less energy being consumed in the primary furnace (Arthur *et al.*, 2003) and the smelter became one of the lowest cost operations globally (Arthur *et al.*, 2003). Due to the success of the ISASMELT™ at Mount Isa Mines, the technology has been installed at 25 sites around the world, with smelters using the technology to process nickel (Bakker *et al.*, 2011), lead (Farrington *et al.*, 2005) and copper (Hunt and Arthur, 2005 and Alvear and Nikolic, 2005) concentrates and secondary materials. Of importance to this work, the Radio Hill ISASMELT™ plant in Western Australia was successfully applied to nickel-copper sulphide smelting, achieving design capacity three months after commissioning.

The Albion Process™ technology was developed as a low-cost way of leaching sulphide minerals. It was initially applied to bulk zinc concentrates which could not be processed in a roaster. It was then applied to gold and silver hosted in pyrite, arsenopyrite and chalcopyrite at GPM, Armenia (Voigt *et al.*, 2015) and subsequently copper sulphides at Sable, Zambia (Voigt *et al.*, 2019). The technology is based on an ultrafine grind of the sulphide concentrate, followed by atmospheric oxidation in agitated leach reactors. The GPM plant was commissioned and ramped up in six months and after two years could be considered to have followed a Series 1 McNulty ramp up curve (Voigt *et al.*, 2018). The Sable plant in Zambia was commissioned and ramped up in six weeks (Voigt *et al.*, 2019). The fast commissioning and ramp up represent significant cost savings for the project. The technology has been shown to have a capital cost 20% lower and an operating cost 30% lower than other technology options (Aylemore and Jaffer, 2015).

PGM processing chemistry

The processing of a PGM concentrate containing approximately 100 g/ton PGM to a high-grade PGM concentrate containing a minimum of 50% PGM consists of two main steps: the production of a Ni-Cu matte and the recovery of base metals.

Smelting of PGM concentrates to a Ni-Cu matte

The smelting of sulphide concentrates from a PGM mine and concentrator has a number of unique chemistry challenges. There are three process challenges in the smelting of PGM concentrates: gangue components; separation of nickel and copper from iron and sulphur while maintaining a high PGM recovery and recovery of entrained PGM matte prills in the slag. Each of these lead to a smelting flowsheet very different from other base metal operations.

The main gangue oxides of concern are chromia (Cr_2O_3) and magnesia (MgO). The chromia stabilises the spinel phase at temperature, commonly described as the end-members chromite or magnetite. The magnesia in the concentrate stabilises olivine and pyroxene at temperature in the furnace. Due to the high magnesia concentrations present in the PGM concentrates, furnaces operate at higher temperatures than other base metal operations. The chromium spinel has a high density and can accumulate at the matte/slag interface or form hearth build-up. This results in furnace containment, accretion and tapping challenges.

In order to separate the nickel and copper from the iron and sulphur, while achieving a high level of PGM recovery, two stages of smelting are performed. In the first stage, the concentrate is smelted with minimal oxidation of the concentrate in the primary furnace to produce a low grade matte. This results in a large matte/slag ratio to improve collection of the PGMs into the matte phase. In the second stage (the converting stage), the low-grade matte is oxidised to remove the sulphur from the matte as $\text{SO}_{2(g)}$ and the iron reports to the slag phase. Due to the high melting point of a Cu-Ni alloy, the converting stage ends with the production of a Cu-Ni-S matte phase typically named Bessemer matte. The converter endpoint (wt% Fe in matte) is dictated by the subsequent nickel-copper separation step.

Due to the melting and reactions occurring in separate furnaces, slag cleaning is performed to recover the PGMs contained in the matte droplets entrained in the slag. A generic flowsheet of the smelting flowsheet is shown in Figure 1.

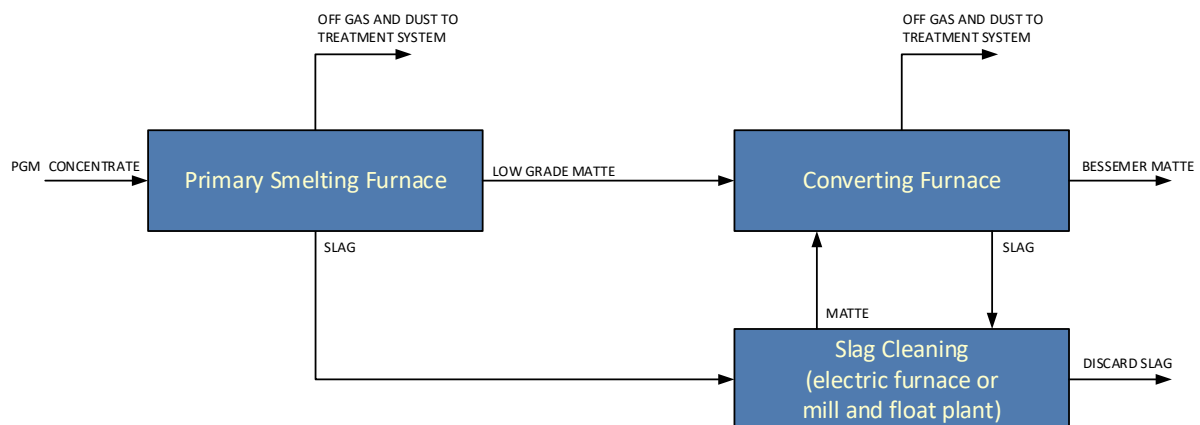


Figure 1. Generic PGM smelting flowsheet.

Refining of nickel-copper matte

The process for the separation of nickel and copper in a Bessemer matte in a sulphate lixiviant was first developed by Outokumpu and was implemented at the Harjavalta nickel refinery in 1960 (Burkin, 1987). Similar patents by Sherritt Gordon followed in the 1970s and the 'Sherritt Gordon' base metal refining process has subsequently been implemented in various forms at PGM operations at Stillwater, Lonmin

(Steenekamp and Dunn, 1999, Steenekamp and Turner-Jones, 2012), Impala (Plasket and Romanchuk, 1978) and Anglo American (Hofiek and Kerfoot, 1992 and Bryson *et al.*, 2008).

The key to this flowsheet is the metathesis reaction in the nickel leach, where nickel sulphides in the matte precipitate Cu^{2+} from solution and in doing so release Ni^{2+} . This allows for the primary separation of nickel from copper without the need for sulphide precipitation and/or solvent extraction which would be required if complete extraction of nickel and copper was targeted in a single leaching stage (Evans *et al.*, 1973). The primary leach may be an atmospheric or pressure leach or both. The copper-free, nickel-rich solution is suitable for nickel recovery as nickel precipitates, nickel briquettes or nickel cathode.

The copper-rich solids remaining after nickel leaching are directed to an oxidative secondary leach and copper are generally recovered as copper cathode. In the existing refineries, using technology from the 1970s, the secondary leach is performed in pressure oxidation vessels at approximately 145°C and 6 bar(g). These leaching processes face challenges with respect to safety and reliability. The solids from the copper leach are now rich in PGMs and can be sent for further upgrading.

A key difference in the market of the 1970s when these flowsheets were developed and now, is the establishment of the mixed hydroxide precipitate (MHP) industry. MHP is a mixed nickel cobalt hydroxide which is a feed to the battery markets. MHP is now a traded commodity at a discount of 5 to 10% of the LME nickel price (Milewski, 2021). As such, this is the most likely product for future PGM refineries processing small to medium quantities of concentrates and mattes.

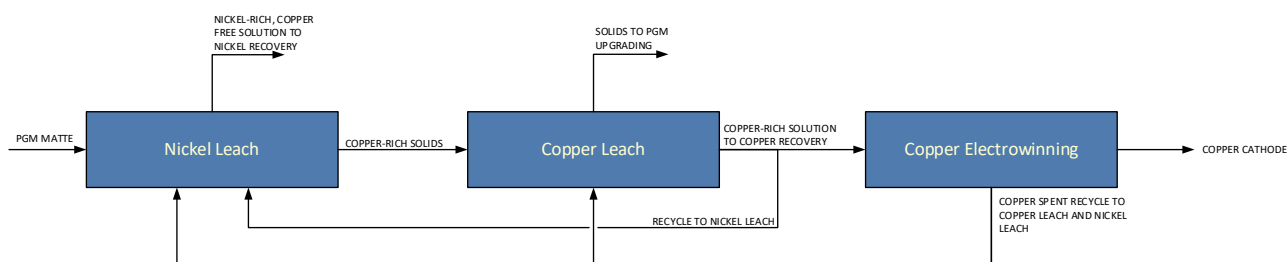


Figure 2. Generic PGM base metal refining flowsheet.

Flowsheet for processing Platreef concentrates

The block flow diagram for the process is given in Figure 3. Broadly, the flowsheet consists of three areas: Area 4000 smelting and offgas handling, Area 5000 base metal recovery and Area 6000 PGM upgrading. Each area is described and the basis for the design is provided.

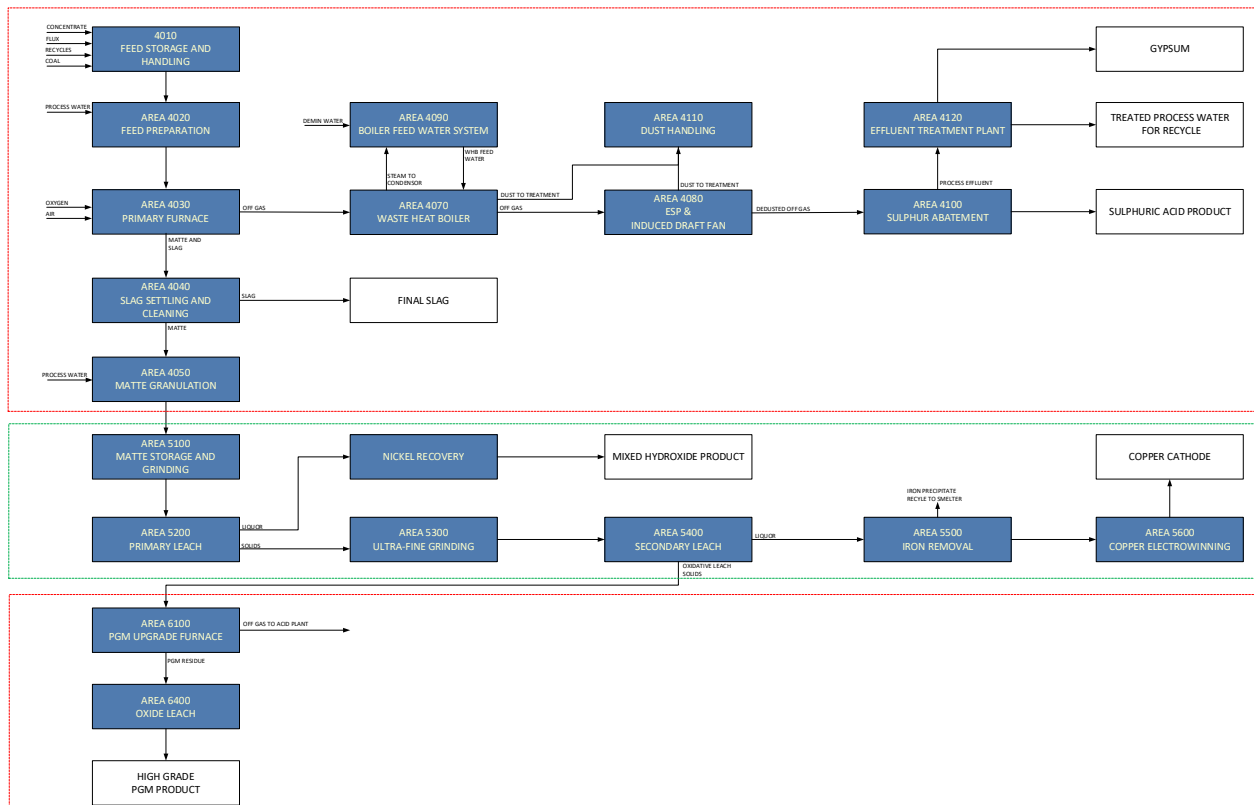


Figure 3. Flowsheet for processing Platreef concentrates.

Smelting

Due to the high sulphide content of the Platreef concentrate, smelting is conducted in a single stage in an ISASMELT™ furnace. The ISASMELT™ furnace, shown in Figure 4, is a cylindrical vessel with a flat roof. The vessel lining is either brick or a mixed refractory/copper cooler arrangement (Hogg *et al.*, 2018). The wet feed is introduced to the furnace by a series of conveyors, without any feed drying required. A central lance injects air, oxygen and fuel into the molten slag bath. The air, oxygen, and fuel injected down the lance violently agitates the liquid, ensuring a very rapid reaction between the raw materials and the slag bath. Importantly, the ISASMELT™ furnace is able to suspend spinel solids (chromia-rich solids) that form in the bath. They are tapped concurrently with matte and slag through a single taphole, reducing the issues associated with higher levels of chromia in the feed.

A frozen layer of slag on the outside of the lance protects it from the aggressive environment in the furnace. Due to the extensive agitation of the ISASMELT™ furnace bath, the molten material is directly tapped into a settling furnace.

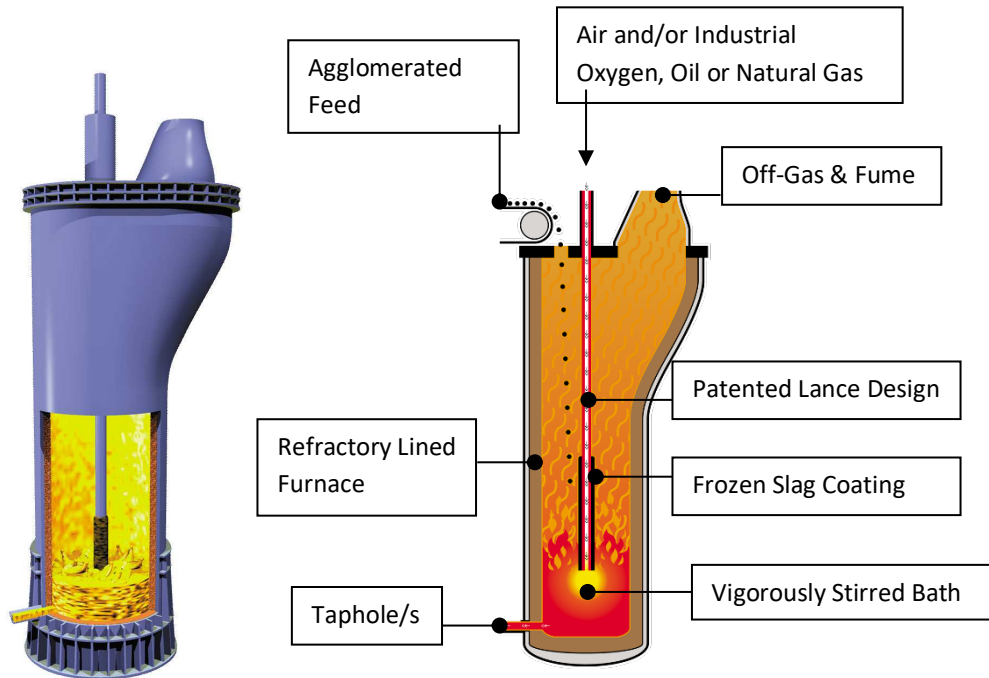


Figure 5. ISASMELT™ technology.

When processing PGM concentrates, the furnace operates at a temperature of 1400°C to 1450°C, depending on the concentrate blend. At these temperatures, water cooling elements in the bath zone are required, due to the superheated matte and a high slag temperature.

Operating at matte grades of 4 wt% Fe results in the slag containing less dissolved copper, cobalt and nickel, thus simplifying slag cleaning, reducing recirculating loads in the smelter and increasing cobalt recovery in the smelter. In addition, nickel dissolved in the slag stabilises the spinel phase with high concentrations resulting in the formation of the nickel ferrite end-member. As a result, the low concentrations of nickel in the slag simplify the fluxing and slag chemistry control.

The matte and slag, tapped concurrently from the ISASMELT™ furnace, are settled in a dedicated settling furnace, rotary holding furnace or matte settling electric furnace. As these settling furnaces are dedicated to this duty, a high level of phase separation is achieved. This results in a very low levels of entrained matte in the discard slag. The matte is granulated for subsequent processing and the slag is then reduced prior to being discarded. When compared to slow-cooled matte, granulated matte is easier to leach in the secondary leach of the base metal refinery.

A summary of the operating conditions able to be achieved with the ISASMELT™ technology is given in Table II.

Table II. ISASMELT™ operating conditions

Parameter	Value
Lance Total Flowrate	200 - 71,000 Nm ³ /h proven
Furnace Fuel/Reductant	Coal, Plastic, Coke, Secondary Scrap
Furnace Trim Fuel Supply	Natural Gas, Diesel, Pulverised Coal, Waste Oil, Hydrogen
Lance Oxygen Enrichment	21 (air) to 95 vol% O ₂ proven
Availability	92%
Taphole Types	Combined or Separate
Furnace Campaign	4+ yrs

Parameter	Value
Ramp-Up to Design Capacity	3 months
Furnace Operation	Continuous
Feed Moisture	0.5 wt% to 14 wt% – based on material properties
Feed Delivery System to Furnace	Vibrating or Belt-Style for Coarse Feed Pneumatic Injection for Fine, Volatile, or Low-Density Feed

Off-gas handling

The furnace off-gas from the ISASMELT™ furnace is hot, dust laden and rich in SO_{2(g)}. The furnace off-gas is first cooled in a waste heat boiler, recovering energy in the off-gas to produce saturated steam. This steam can be upgraded to superheated steam and used in a steam turbine to produce power, making the smelter energy positive. This is practised at Mount Isa Mines, Australia where steam generated in the ISASMELT™ powers a 10 MW power station.

The dust is captured in a baghouse or ESP and is able to be directly recycled back to the flowsheet. The off-gas is processed in an acid plant to produce a sulphuric acid product, difficult or costly with other furnace technologies.

Importantly, when compared to batch converting and other furnace technologies, the SO_{2(g)} generated in the primary off-gas is recovered as sulphuric acid and fugitive emissions of SO_{2(g)} are captured. In a conventional converter, up to 5 wt% of the sulphur in the feed is emitted into the surrounding environment as fugitive emissions while in the ISASMELT™ technology this is reduced to less than 0.02 wt% of the sulphur in the feed (Jones *et al.*, 2019).

In addition to the sulphuric acid product, the acid plant also produces a weak acid stream. This stream is treated in a high density sludge water treatment plant. This plant also treats liquor from the mixed hydroxide precipitate (MHP) plant. Lime is used as the neutralising agent to form a stable gypsum residue. The cleaned water can be recycled back to the process.

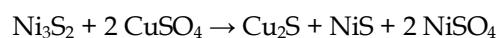
Primary leach

The granulated converter matte from the ISASMELT™ furnace area is received in the matte storage and grinding area. A ball mill in closed circuit reduces the particle size distribution from approximately 2 mm to 80% passing 45µm. A matte slurry storage tank with a residence time of 12 hours receives the ground matte. Matte is dosed to the primary leach which consists of a non-oxidative section (four hours residence time) and an oxidative section (eight hours).

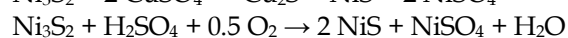
In the non-oxidative section, copper advance solution is contacted with the ground nickel-copper matte and copper is removed from solution via metathesis (Reaction 1) and nickel is released to solution. In the oxidative section, oxygen is supplied and oxidative leaching of heazlewoodite and metallic nickel (Reaction 2 and 3) occurs. As acid is consumed, the pH rises and copper and then iron hydrolysis occurs as per Reaction 4 and 5 (Hofirek and Kerfoot, 1992).

Nickel extractions across primary leach of up to 75% can be achieved (Steenekamp and Dunn, 1999). Solution tenors of 80 to 100 g/L Ni, 0 to 2 g/L of copper and 0 to 2 g/L H₂SO₄ can be expected (Crundwell *et al.*, 2011).

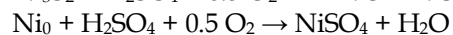
In the primary leach thickening and filtration stage, nickel-rich liquor is separated from the copper-rich solids. The filtration step is essential as it prevents carryover of nickel to the secondary leach.



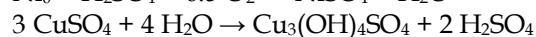
Reaction 1



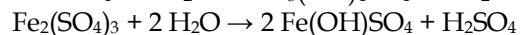
Reaction 2



Reaction 3



Reaction 4



Reaction 5

Nickel recovery

In the nickel recovery area, nickel sulphate in solution is precipitated as an MHP product. The nickel-rich liquor from primary leach is directed to nickel recovery where two pH adjustment steps are performed. In the first stage, the pH is raised to 3.2 using low cost quicklime to remove residual acid, copper and iron from solution. For this stage, a residence time of eight hours is typically used and the temperature is controlled to 80°C by steam heating coils.

In the second stage the pH is successively raised to 5.5 and then 8.8 using magnesia to precipitate a mixed nickel and cobalt hydroxide product (Reactions 6 and 7). The magnesium sulphate-rich supernatant is treated in the water treatment plant and process water is recovered. The precipitation kinetics are rapid and a residence time of one hour is typically required. The temperature is controlled to 80°C by steam heating coils. The slurry is filtered to recover the MHP cake and the liquor is sent to the water treatment plant for magnesium removal and water recovery.



Secondary leach

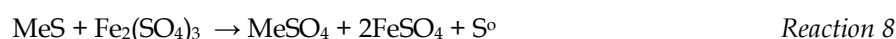
The copper-rich solids from the nickel leaching area are directed to ultra-fine grinding in an IsaMill, where the p80 is reduced to 80% passing 12µm. At this grind size, surface passivation from elemental sulphur formation is prevented as the mineral will leach prior to the elemental sulphur layer becoming thick enough to passivate the mineral. The rate of leaching is also enhanced, due to the increase in the mineral surface area. The IsaMill technology has been widely used since the early 2000s, first in ultra-fine grind applications and then in mainstream inert grind (MIG) applications in the South African platinum industry (Buys *et al.*, 2004, Rule *et al.*, 2008).

The finely ground solids are then treated in the secondary leach. In traditional Sherritt Gordon flowsheets, the secondary leach is conducted at 6 bar and 145°C. In this flowsheet, the benefits of ultrafine grinding are realised in the Albion Process™, an atmospheric oxidative leach.

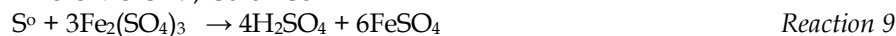
The oxidative leach is carried out in leach reactors operating close to the boiling point of the slurry. Leaching is carried out autothermally, in that the temperature of the leach slurry is set by the amount of heat released by the leaching reaction. Heat is not added to the leaching vessel from external sources. Temperature is controlled by the rate of addition of oxygen, and by the leach slurry density, with excess heat removed by direct evaporative cooling. There is no need for cooling coil installation, flash recycle or flash-thicken-recycle systems.

Oxygen is injected into the leach tanks using Hypersparge technology. The supersonic injection creates a jetting plume of oxygen and a localised zone of very high shear between the gas and liquid phase. This promotes gas transfer. Ultrafine bubbles of oxygen peel off from the jet, increasing the surface area for the oxygen dissolution. Down pumping, high solidity agitators are used to maximise the gas hold up and impact shear to the slurry.

Sulphide leaching in the Albion Process™ occurs via ferric iron intermediates, in two steps as shown in Reactions 8 and 9. In the first step, the sulphide is oxidised to a soluble sulphate and elemental sulphur. In the second step, the elemental sulphur is oxidised to sulphuric acid. 5 to 20% of the elemental sulphur is oxidised, which reduces acid demand. Based on the Albion Process™ operation at Sable, the residence time for the secondary leach is 24 hours, which will achieve a copper extraction of 99% and a nickel extraction of 98%.



where Me is Ni, Cu or Co



The residue containing the elemental sulphur and PGMs is separated from copper-rich liquor using filtration only. Thickening is not recommended as elemental sulphur is known to produce a floating scum which could entrap PGMs. The elemental sulphur and PGM residue is sent to the PGM upgrade furnace and the liquor is directed to the iron removal step.

The Albion Process™ requires an iron tenor of 5 to 10 g/L for effective and rapid conversion of the sulphides. The tolerance for Fe in direct electrowinning circuits is 3 to 5 g/L Fe (Ryley, 1983). Iron removal is therefore performed after the Albion Process™. By increasing the pH to 3.5 using magnesia, a basic ferric sulphate is precipitated. Thickening and filtration steps separate the iron solids from the copper-rich solution, now referred to as copper advance solution. Some of the iron solids are recycled back to the secondary leach to establish the requisite 5 to 10 g/L Fe and the rest is recycled back to the smelter. Any PGMs that leach under secondary leach conditions are co-precipitated with the iron and recovered in the ISASMELT™ furnace. Copper advance solution is recycled back to the primary leach to provide the Cu²⁺ for the metathesis reactions.

Copper recovery

Copper electrowinning of copper advance solution arising from nickel-copper matte leaching is directly plated on to stainless steel cathodes such as Glencore Technology's ISAKIDD technology. For good copper cathode quality, nickel content in the solution needs to be controlled between 50 to 70 g/L Ni. Nickel tenors higher than 70 g/L result in nodule and dendrite formation (Ryley, 1983). Due to the efficient solid liquid separation after the primary leach, the modelling of this flowsheet suggests that approximately 45 g/L Ni is achievable in the copper advance. Current densities of approximately 200 A/m² are applicable for these solutions (Ryley, 1983). As the copper advance solution and the copper spent solution is used as a copper and acid source in the primary leach and secondary leach respectively, it is essential that large solution storage facilities are considered. For this duty, residence times of 48 hours for copper advance storage and copper spent storage are typically required.

PGM upgrading

The elemental sulphur contained in the secondary leach residue is removed in the PGM upgrade furnace. The off-gas from the furnace is directed to the off-gas handling system of the primary smelter. In this way, elemental sulphur is recovered as sulphuric acid. The remaining residue is leached in a mildly acidic sulphate medium to produce a PGM residue containing at minimum 50% PGM. The liquors from the leach are recycled back to the secondary leach to recover the nickel and copper units.

Flowsheet summary

The use of ISASMELT™ technology for the smelting of PGM concentrates provides a number of improvements when compared to conventional processing technology used in the industry. The ISASMELT™ technology uses the fuel value of the feed for reaching operating temperature as opposed to solely electrical energy and is able to recover energy from the off-gas in the waste heat boiler to produce saturated steam. In addition, the SO_{2(g)} in the off-gas is recovered as a sulphuric acid product. The saturated steam can be upgraded to superheated steam and used in a steam turbine to produce power, making the smelter energy positive. The ISASMELT™ technology is equipped with modern safety equipment, making it safe for operators and other personnel at the smelter. The furnace technology is also able to be used for both small and large throughputs, enabling the smelting to occur directly beside the PGM concentrator.

The Albion Process™ technology offers a number of significant benefits when compared to alternative secondary leaching technology. Despite the ultrafine grinding of the sulphide materials being energy intensive, the total energy consumed is more than half that of pressure oxidation technology (Aylemore and Jaffer, 2005). The leaching reactors operate at atmospheric conditions, using low cost materials and simple reagents. This results in significantly safer and more reliable operation when compared to pressure oxidation vessels.

EVALUATION OF THE FLOWSHEET

Environmental evaluation

The environmental evaluation of three PGM flowsheets are considered in Table III: this flowsheet; a traditional PGM flowsheet and a primarily hydrometallurgical flowsheet.

Table III. *Environmental evaluation*

	<i>Flowsheet presented in this paper</i>	<i>Traditional PGM producer</i>	<i>Hydrometallurgical flowsheet</i>
Sulphur emission capture	99.5+%	95+% (Peirce-Smith converter)	99+% (roaster)
Sulphur capture	Sulphuric acid product	To atmosphere, gypsum residue or sulphuric acid product (batch converting furnaces require larger sulphur capture equipment)	To atmosphere, gypsum residue or sulphuric acid product
Process safety - pyrometallurgical steps	Gas emissions, electrical risks	Gas emissions, higher electrical risks (with primary smelting electric furnace)	Gas emissions (roaster)
Process safety - hydrometallurgical steps	Low pressure oxygen, fire risk	Pressure vessels, explosion risk and oxygen fire risk	Pressure vessels, explosion risk and oxygen fire risk
Major chemicals used	Oxygen, sulphuric acid, lime and magnesia	Oxygen, sulphuric acid and caustic soda	Oxygen, sulphuric acid and chlorides
Energy recovery	From smelter off-gas	Not common	Not common
Technology maturity	Three ISASMELT™ site in Africa (x3), one Albion Process™ in Zambia (Copper Leaching)	Four operating sites in South Africa	No operating sites

Energy efficiency

The specific energy consumption of an existing producer for the smelting and base metal refining is calculated as 1.8 PJ/Moz (data from Anglo American, 2009). For the flowsheet presented in this work, the specific energy consumption drops to 1.1 PJ/Moz, a 40% reduction. The pyrometallurgical portion of both flowsheets contribute approximately 80% to the total energy demand. The specific energy reduction is due to the use of the more energy efficient technologies.

The ISASMELT™ furnace uses the fuel value of the feed to reach operating temperature. In the traditional flowsheet, electrical energy is required to obtain and maintain operating temperature in the first stage of smelting. In the hydrometallurgical area, the primary leach uses steam generated in the ISASMELT™ waste heat boiler and the Albion Process™ operates autothermally, thus negating the need for gas or coal fired boilers.

Electricity from steam generated in the ISASMELT™ waste heat boiler has not been included in the specific energy consumption presented above. However, it does mean that there is potential to further reducing the specific energy consumption for the presented flowsheet if a steam turbine were installed.

Economic evaluation

A Class 5 or concept level (AACE, 2020) economic evaluation of the flowsheet was conducted ($\pm 45\%$ accuracy). The evaluation included the plant areas described in Section 0 and provided in

Figure . Two flowsheet sizes were considered: a feed of 250 000 tonne/year concentrate and 800 000 tonne/year concentrate.

The motivation for further processing of one's own concentrates is:

- The production of high grade PGM concentrate on site extracts more revenue from PGM units
- The treatment of concentrate on site avoids loss of value from penalty elements
- The processing facility is tailored to the concentrate
- The production of nickel and copper product increases revenue from these by-products
- Sulphuric acid is able to be produced and sold, generating extra revenue
- Mining and concentrating of PGM ores constitutes 88% of the total OPEX (Ndlovu, 2014), meaning that once capital is sunk, there is a marginal OPEX increase for smelting and refining

Table IV provides the capital and operating cost estimates for the two plant sizes. The concentrate grades and PGM pricing used in the calculation is provided in Table V and Table VI and recoveries are provided in Table VII.

From the estimated capital cost and profit, the payback period for the installation of the smelting and base metal refining complex can be estimated. A payback period of 5.5 and 4 years was calculated for the 250 000 tonne/year and 800 000 tonne/year plants, respectively.

Key assumptions used in the calculation are:

- Sale of PGM concentrates will attract 85% of metal value (Ndlovu, 2014)
- Sale of high grade PGM concentrate will attract 90% of metal value (Ndlovu, 2014)
- Sale of LME grade copper with a premium of USD 200/tonne
- Sale of MHP will attract 95% of metal value (Milewski, 2021)

Table IV. Capital and operating cost estimate

	Case 1	Case 2
Plant throughput (000 tonne/year)	250	800
Total CAPEX (million USD)	237	692
Hydrometallurgical areas ('000,000 USD)	67	154
Pyrometallurgical areas ('000,000 USD)	170	538
Total OPEX	42.3	112.5
Hydrometallurgical areas ('000,000 USD/y)	14.8	40.3
Pyrometallurgical areas ('000,000 USD/y)	27.5	72.1
Payback period (years)	5.5	4.0

Table V. Concentrate grades¹

Metal	Unit	Grade	
Platinum	g/t	38.2	
Palladium	g/t	39.0	
Gold	g/t	5.3	
Rhodium	g/t	2.4	
3PE+Au	g/t	85.0	
Copper	% Cu	3.3	
Nickel	% Ni	5.4	

1. From Ivanhoe, 2022

Table VI. PGM pricing

Metal	Unit	Price
Platinum ¹	USD/ounce	950
Palladium ¹	USD/ounce	2,000
Gold ¹	USD/ounce	1,600
Rhodium ¹	USD/ounce	5,000
Copper	USD/tonne	7,500
Nickel	USD/tonne	15,000

1. From Ivanhoe, 2022

Table VII. PGM recoveries

Metal	Unit	Recovery
Platinum	%	98
Palladium	%	98
Gold	%	98
Rhodium	%	98
Copper	%	97
Nickel	%	97

CONCLUSION

The flowsheet presented in this work makes use of the ISASMELT™ and Albion Process™ technology to process a lower-grade, sulphide-rich Platreef concentrate to a high-grade PGM concentrate. Benefits of this flowsheet compared to the traditional PGM flowsheet are:

- The specific energy consumption is 1.1 PJ/Moz compared to 1.8 PJ/Moz for a traditional flowsheet. This is primarily due to the ISASMELT™ furnace using the fuel value of the feed to reach operating temperature, whereas in the traditional flowsheet, electrical energy is required in the first stage of smelting.
- Sulphur emission capture of greater than 99.5% is possible in an ISASMELT™ compared to 95% in a Peirce-Smith converter.
- Steam is generated in the ISASMELT™ waste heat boiler which can be supplied to other plant areas. Energy recovery systems such as steam turbines can be installed, as practised at Mount Isa Mines. Energy recovery in traditional converters is not common.
- The use of the atmospherically operated Albion Process™ in the secondary leach step reduces the risks associated with pressure oxidation. These risks include pressure vessel operation and maintenance and oxygen fire risk.
- The operation of the Albion Process™ is simpler in that cooling is effected by direct evaporative cooling and there is no need for cooling coil installation, flash recycle or flash-thicken-recycle systems.

A Class 5 level economic evaluation of the flowsheet showed that the payback period is 5.5 and 4 years when processing 250,000 and 800,000 tonne/year Platreef concentrate, respectively. This provides motivation for smaller producers with a single sulphide-rich concentrate to consider installing their own processing facilities rather than selling their concentrate.

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