

Recovery of PGMs from Flotation Concentrate Derived from an Oxidized PGM Ore

Tshiamo Pheko, Willie Nheta

Abstract—Platinum Group Metals (PGMs) are critical minerals with wide-ranging industrial applications, yet significant quantities remain locked in oxidized ore bodies, particularly in South Africa and Zimbabwe. Conventional metallurgical methods fail to achieve high recoveries from these complex ores due to the lack of sulphide associations and the presence of refractory mineral phases. This study proposed a hydrometallurgical approach to recover PGMs from a flotation concentrate derived from oxidized ore using a two-stage leaching process. Initially, base metals were removed using sulphuric acid, followed by PGM extraction in the second stage using hydrochloric acid and hydrogen peroxide as an oxidizing agent. The leaching parameters were optimized using Response Surface Methodology (RSM). The sample was characterised using XRF for chemical composition, XRD for mineralogy, SEM for surface morphology and Fire assay coupled with AAS for quantification of PGMs. XRF results revealed that the sample had high concentrations of Fe(28.34%), Si(26.83%), Cr(18.51%) and trace elements like Mg, Al, Ca, Ti and Ni. XRD analysis identified minerals such as magnesiochromite, magnesioferrite, quartz, enstatite, aluminosilicates (anorthite, akermanite), pyrrhotite, magnetite, rutile and periclase. SEM analysis revealed PGM embedded in a matrix of quartz, Iron oxide, magnesiochromite and magnesioferrite. The highest recoveries of PGM obtained were 72.1% Pt, 65.2% Pd and 27.5%Au.

Keywords— Oxidized PGM ores, Flotation, Leaching, PGM recovery.

I. INTRODUCTION

Platinum elements (PGEs) consist of six metals: ruthenium, rhodium, palladium, osmium, iridium, and platinum that share similar physical and chemical properties. These metals, along with silver and gold, are classified as precious metals due to their rarity and value [1]. Russia and South Africa are widely recognized as the leading producers of platinum group metals (PGMs) and possess the largest global reserves of PGM-bearing minerals. South Africa's reserves and resources of PGEs are estimated at approximately 85,000 metric tons. In Russia, the Norilsk region alone contains around 2,500 metric tons of platinum (Pt) and palladium (Pd) reserves [2].

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South Africa's Bushveld Igneous Complex (BIC) has more than 80% of the world's known Platinum group metals (PGMs) deposits, which are found in three major seams of the BIC, namely: Merensky and Upper Group 2 (UG2) reefs, as well as the Plat reef. PGMs have a variety of applications, including catalysts, fuel cells, metal-winning electrodes, dental alloys, and even semiconductors [3]. The methods used to beneficiate and recover PGE minerals from ores are influenced by various factors, including mineralogical composition, the nature of mineralization, and the characteristics of the host rock. Typically, the processing of mined PGE ores involves a sequence of comminution, gravity separation, and flotation [4].

South Africa and Zimbabwe have an estimated reserve of more than 500 metric tonnes of oxidized platinum group metal (PGM) ores. These reserves include both unmined deposits and material that has been mined, stockpiled, or discarded as overburden waste [5]. Oxidized PGM ore deposits located at or near the Earth's surface offer promising opportunities for future PGM production due to their substantial resource estimates, relatively high PGM grades, and the feasibility of low-cost open-pit mining. However, these ores remain largely untapped and untreated because conventional metallurgical techniques fail to achieve sufficient recoveries. The intricate mineralogy and polymodal distribution of PGMs in oxidized ores present significant challenges for metallurgists, particularly in terms of mineral liberation and extraction [6]. Processing oxidized PGE ores is particularly difficult due to (1) the highly variable extent of oxidation, (2) significant variations in PGE mineralogy and host minerals, (3) the absence of sulphide mineral associations required for flotation, and (4) the formation of mineral coatings on PGMs because of weathering [7].

Research on the extraction of PGMs from oxidized ores has identified various methods to enhance recovery, including flotation and leaching techniques. Studies have demonstrated that flotation reagent schemes, such as hydroxamate oxide collectors, can significantly improve PGM recovery rates, achieving up to 74.7% [5]. However, direct leaching with acids like hydrochloric acid and nitric acid yielded relatively low extraction rates, necessitating alternative approaches like leaching flotation concentrates and utilizing hybrid processes involving chlorinating agents. Additionally, innovative methods such as biogenic siderophore leaching have shown promise in extracting Pt and Pd from oxidized ores. Despite these advancements, challenges remain, particularly with refractory minerals like sperrylite (PtAs_2), which exhibit poor leaching kinetics and resistance to cyanide solutions [8].

Direct leaching of oxidized PGM ore with different acids (aqua regia, HCL, HNO₃, and H₂SO₄) has been conducted, in which aqua regia yielded the highest PGM recoveries (<50%) compared to other lixivants. However, due to their high lixiviant consumption, hydrometallurgical processing of ROM ore with nonselective solvents such as aqua regia is not recommended [5]. Furthermore, the concentrated lixiviant's harsh and corrosive properties are a disadvantage of pure aqua regia [9]. As a result, aqua regia is typically utilized in laboratories, but it is not frequently used as a hydrometallurgical process medium in industry.

There is a need to develop more selective and cost-effective alternative processing methods for treating oxidized ores that will yield high recoveries of PGMs.

II. MATERIALS AND METHODS

A. Materials

The study utilized oxidized PGM ore sourced from the western limb of the BIC as the primary feed material. A Denver flotation cell was used to separate valuable minerals from gangue, with Sodium Isobutyl Xanthate (SIBX) serving as the primary collector for sulphide minerals and AM810 hydroxamate as a co-collector for oxidized PGMs. M98B depressant was used to suppress gangue in flotation, copper sulphate acted as an activator, and FLOMIN frother stabilized the froth. During leaching, hydrogen peroxide (H₂ O₂) was used as an oxidizing agent, sulphuric acid (H₂ SO₄) as a base metal lixiviant, and hydrochloric acid (HCl) as a PGM lixiviant. Finally, material characterization was performed using Rigaku ZSX Primus II XRF, Rigaku Ultima IV XRD, TESCAN SEM, and AAS to determine elemental composition, mineral phases, surface morphology, and PGM concentration.

B. Sample preparation and characterisation

The bulk sample (10 Kg) was blended, rifled and split using a rotary splitter into approximately 1 kg samples prior to batch flotation tests. Approximately 1 Kg of the sample was screened to determine the PSD of the feed. In preparation for head analysis, a representative sample of roughly 200 g of the oxidized PGM ore was separated from the bulk and pulverized to a fineness of 90% passing 75µm. Sample characterisation was conducted using Rigaku ZSX Primus II XRF, Rigaku Ultima IV XRD, TESCAN SEM, and Fire assay coupled with AAS to determine elemental composition, mineral phases, surface morphology, and PGM concentrations, respectively.

C. Flotation experiments procedure

The sample was concentrated using the flotation process and the flotation conditions shown in Table 1. The sample underwent milling for size reduction and mineral liberation in which the targeted P80 was 75 µm. A slurry level of roughly 30% w/w solids was used for batch flotation testing. A Denver machine running at an impeller speed of 1200 r/min was employed for the tests. The optimal reagent scheme for flotation recovery of PGMs into the rougher concentrates was

obtained from literature. For 10 minutes of cumulative flotation, concentrates were manually collected by scraping off the froth into collection pans every 10 seconds [5].

TABLE I: FLOTATION REAGENT CONDITIONS

Sequency of reagent dosage	Reagent	Function	Dosage (g/t)	Conditioning time(min)
1	copper sulphate	Activator	50	5
2	SIBX	Primary collector	150	2
3	AM810	Oxide co-collector	50	2
4	M98B	Depressant	50	3
5	FLOMIN	Frother	50	1

D. Base metals leaching experiments.

The base metal leaching experiments were prepared by adding 200 mL of sulfuric acid (H₂ SO₄) solution at a concentration determined through Response Surface Methodology (RSM). For each run, 10 grams of the sample were accurately measured using a watch glass and placed into a 400 ml beaker. The prepared acid solution was then added to the beaker, and the mixture was stirred at a consistent speed of 250 rpm using a magnetic stirrer. Each experiment was run for 60 minutes at the specified temperatures. Once the leaching process was complete, the solution was filtered using filter paper and a funnel. The filtrate (leachate) was transferred into a test tube, labelled according to the respective experimental conditions, and analysed using Atomic Absorption Spectroscopy (AAS). This procedure was repeated for each experimental run as defined by the RSM design.

E. PGMs leaching experiments.

Following the initial leaching process, the solid residue was subjected to a secondary leaching experiment to recover PGMs using hydrochloric acid (HCl) and hydrogen peroxide (H₂ O₂). The residue was first carefully collected from the filter paper and dried at room temperature or in a drying oven set to approximately 50 °C to remove any remaining moisture. Once dried, the residue was transferred into the reactor. Stock solutions of HCl and H₂ O₂ were prepared according to the concentrations extrapolated from RSM and added to the reactor. The mixture was continuously stirred mechanically to ensure uniform distribution of the ore and acids within a period of 2 hours for each run. The leaching process was carried out at predicted temperatures extrapolated from RSM, for a duration of 2 hours. After leaching, the slurry was filtered using a funnel and filter paper to separate the solid residue from the leachate. The filtrate was analysed using Atomic Absorption Spectroscopy (AAS) to quantify PGM content.

III. RESULTS AND DISCUSSION

A. Chemical composition of the as-received sample.

The XRF analysis of as received sample showed high concentrations of Fe(28.34%), Si(26.83%), Cr(18.51%) occurring with valuable elements such as Cu(0.05%), Ni(0.36%) and Zn(0.07%) associated with sulphur(0.23%) and trace elements like Mg, Al, Ca and Ti. XRF analysis of flotation concentrate showed high concentration of Si(37.51%), Fe(28.40%), Mg(17.67%) occurring with valuable elements such as Cu(0.20%), Ni(0.54%) and Zn(0.06%) associated with sulphur(0.62%) and traces of Cr, Al, Na, and Ca. The elevated Fe and Cr contents suggested that the feed sample was rich in ferrochromite or other Fe–Cr-bearing spinels, which are typical in PGM-bearing ores and chromititic materials. The high silicon content indicated a significant presence of silicate gangue minerals, likely quartz or aluminosilicates, which tend to host Mg.

B. Mineralogical analysis of the as-received oxidized ore

The mineralogy of the as-received sample was performed using the XRD and the results are displayed in Fig. 1 below. The XRD analysis identified minerals such as magnesiochromite, magnesioferrite, quartz, enstatite, aluminosilicates (anorthite, akermanite), phyrrotite, magnetite, rutile and periclase. The presence of chromite and magnetite confirms an oxide mineral assemblage, supporting the XRF findings of high Fe and Cr. The quartz and silicate peaks indicate gangue minerals like serpentine or talc, which can affect flotation by entrainment rather than true mineral recovery. The absence of peaks related to sulfide minerals (such as pentlandite or chalcopyrite) reinforces that the ore is oxidized. The sharpness of the peaks shows good crystallinity, meaning these minerals are well-developed and stable, which is typical of weathered oxide ores. This mineralogical composition aligns with XRF results and supports the classification of the material as an oxidized PGM-bearing ore.

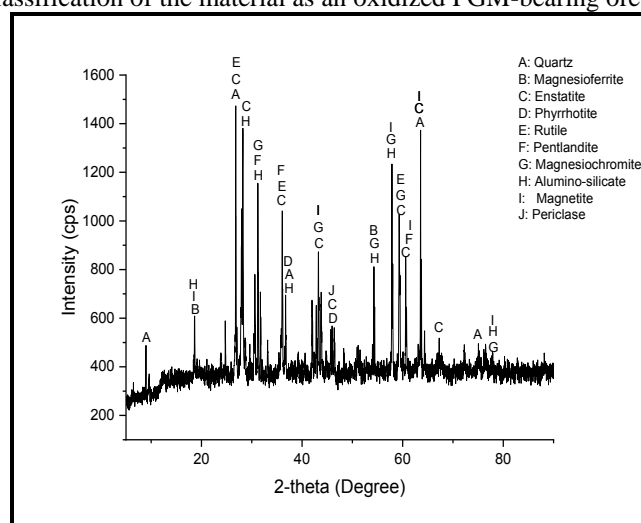


Fig. 1. Mineralogical composition of the as-received sample

C. PGM Assay of the as-received sample and the flotation concentrate

The PGM assays of the as-received sample and the flotation concentrate obtained from the fire assay, coupled with AAS are shown in Table II.

Average assay(ppm)	Pt	Pd	Au	3E
Head sample	0.91	0.88	0.12	1.91
Flotation concentrate	1.15	1.1	0.19	2.44

The average PGM assay in the head sample is 1.19 ppm for the 3E (Pt, Pd and Au). After flotation, the concentrate showed slightly higher values of 3E, yielding 2.44 ppm. Despite the increase, the absolute PGM grades in the concentrate remained low, confirming the refractory nature of the oxidized ore. The limited upgrade suggested that most PGMs were finely disseminated or associated with oxide and silicate minerals rather than sulphides, which are more amenable to conventional flotation. The weak response to flotation could also be attributed to poor liberation of PGM-bearing phases or entrainment of fine gangue minerals in the froth. Overall, these findings demonstrated that flotation alone was insufficient for significant PGM recovery from oxidized ores.

D. Base metal leaching experiments results.

The base metal leaching results presented in Table III indicate the recovery efficiencies of the base metals following the leaching process. The highest recoveries were 9.03% Fe, 5.74% Cu, and 3.94% Ni, respectively. These values are considerably low, suggesting that the leaching process was not highly effective in dissolving and recovering the base metals from the flotation concentrate. One of the main reasons for the low recoveries is likely the consumption of the lixiviant by gangue minerals before it could effectively react with the target base metal phases.

Temp. °C	[H ₂ SO ₄]	% Wt Fe	% Wt Cu	% wt Ni
80	3	9.03	5.74	3.94
90	2	3.09	3.67	1.54
90	3	3.39	2.97	1.27
90	3	4	4.13	1.47
90	3	3.25	5.26	2.02
90	3	6.67	4.25	1.22
80	4	4.27	2.49	1.63
100	2	3.09	5.51	1.39
90	4	4.35	1.77	1.48
100	4	4.17	2.19	1.02
90	3	3.47	4.98	1.38
80	2	3.76	2.43	0.97
100	3	3.71	7.4	1.59

The flotation concentrate contained a significant proportion of non-sulphide gangue material and a notable amount of chromite, both of which tend to consume acid or other leaching

reagents without contributing to metal dissolution. This results in reduced reagent availability for attacking the desired metallic phases.

E. PGMs leaching experiments results

The results presented in Table III indicate the recovery efficiencies of the PGMs following the leaching process. The highest recoveries achieved were 72.1% for Pt, 65.2% for Pd, and 27.5% for Au. These results demonstrate that the applied leaching conditions were effective in dissolving a substantial portion of the PGMs, particularly platinum and palladium. The highest recoveries were obtained under elevated temperatures exceeding 90 °C, using hydrochloric acid concentrations greater than 4.5 M in combination with 5 vol% hydrogen peroxide (H₂O₂) as an oxidizing agent. These conditions enhanced the redox potential of the leaching medium, promoting the oxidative dissolution of PGMs into their soluble chloro-complexes, such as [PtCl₆]²⁻ and [PdCl₄]²⁻.

The relatively high recoveries of Pt and Pd suggest that the leaching lixiviant composition and process parameters were well-suited for their dissolution. The use of H₂O₂ likely improved metal solubilization by maintaining a strong oxidizing environment, enabling effective breakdown of any residual PGM-bearing oxide or hydroxide phases. In contrast, gold exhibited a lower recovery of 27.5%, which can be attributed to its noble character and lower chemical reactivity. Gold in oxidized ores often occurs as fine inclusions or is encapsulated within silicate and chromite gangue phases, which hinder reagent access and limit dissolution under the applied leaching conditions.

Notably, the presence of excess gangue material in the flotation concentrate did not significantly hinder PGM recoveries, unlike the case with base metal sulfides (BMS), which were more sensitive to reagent consumption by gangue minerals. This suggests that PGMs were present in more leachable mineral forms or that the oxidative leaching system was sufficiently strong to overcome partial gangue interference.

TABLE III: PGMs LEACHING RESULTS.

Temp.(°C)	%H ₂ O ₂	[HCl]	%Pt	%Pd	%Au
90	3	6	60.5	46.5	13.7
100	5	4.5	72.1	64.9	27.5
90	4	4.5	58.2	62.8	7.2
80	3	4.5	34.3	29.4	5.4
90	5	3	57.8	59.2	10.6
80	4	3	30.6	25.3	7.7
90	5	6	61.2	65.2	9.3
100	3	4.5	69.3	30.2	19.4
90	3	3	51.6	23.2	6.3
100	4	3	67.4	27.5	14.8
80	4	6	35.1	31.1	5.9
80	5	4.5	32.8	30.2	6
100	4	6	70.3	31.9	8.1

IV. CONCLUSION

This project aimed to recover PGMs from a flotation concentrate obtained from oxidized PGM ore using a hydrometallurgical approach. The study successfully investigated the flotation and leaching behaviour of oxidized PGM ores through a systematic experimental design encompassing sample preparation, characterization, and process optimization. The sample was dominated by oxide and silicate minerals such as magnesiochromite, magnesioferrite, and quartz, with minimal sulphide phases, verifying its highly oxidized nature. Flotation results indicated poor upgrading of PGMs. This limited recovery was attributed to the entrainment of silicate gangue and the refractory association of PGMs with oxide minerals.

Subsequent hydrometallurgical treatment through acid leaching demonstrated that base metal recoveries were relatively low (Fe: 9.03%, Cu: 5.74%, Ni: 3.94%) due to gangue mineral interference and suboptimal flotation selectivity. In contrast, PGM leaching using HCl–H₂O₂ showed significantly improved recoveries, achieving up to 72.1% Pt, 65.2% Pd, and 27.5% Au under optimized conditions of high temperature and oxidant concentration.

Overall, the findings reveal that while conventional flotation alone is inadequate for oxidized PGM ores, a combined approach involving optimized flotation followed by oxidative acid leaching can yield improved metal recoveries.

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REFERENCES

- [1] G. M. Mudd, S. M. Jowitt, and T. T. Werner, "Global platinum group element resources, reserves and mining – A critical assessment," *Sci. Total Environ. and Sci Total Environ.*, vol. 622–623, pp. 614–625, 2018, doi: 10.1016/J.SCITOTENV.2017.11.350).
- [2] C. O'connor and T. Alexandrova, "The geological occurrence, mineralogy, and processing by flotation of platinum group minerals (Pgms) in South Africa and Russia," *Miner.*, vol. 11, no. 1, pp. 1–15, Jan. 2021, doi: 10.3390/min11010054.
- [3] J. H. Potgieter and E. van der Lingen, "Noble Metal Additions to Passive Alloys: Effect on Corrosion Resistance," *Reference Module in Materials Science and Materials Engineering*, 2016, doi: 10.1016/B978-0-12-803581-8.01649-0.
- [4] P. Sahu, M. S. Jena, N. R. Mandre, and R. Venugopal, "Platinum Group Elements Mineralogy, Beneficiation, and Extraction Practices—An Overview," *MIN. PROC. EXT. MET. REV.*, vol. 42, no. 8, pp. 521–534, Nov. 2021, doi: 10.1080/08827508.2020.1795848.
- [5] R. Sefako, V. M. Sibanda, and K. Sekgarametso, "PGM extraction from oxidized ores using flotation and leaching," *J. South. Afr. Inst. Min. Metall.*, vol. 119, no. 11, pp. 929–936, 2019, doi: 10.17159/2411-9717/287/2019.
- [6] C. N. Mpinga, J. J. Eksteen, C. Aldrich, and L. G. Dyer, "A conceptual hybrid process flowsheet for platinum group metals (PGMs) recovery from a chromite-rich Cu-Ni PGM bearing ore in oxidized mineralization through a single-stage leach and adsorption onto ion exchange resin," *Hydrometallurgy*, vol. 178, pp. 88–96, 2018, doi: 10.1016/j.hydromet.2018.03.024.
- [7] D. Kraemer, M. Junge, and M. Bau, "Oxidized Ores as Future Resource for Platinum Group Metals: Current State of Research,"

Chemie-Ingenieur-Technik, vol. 89, no. 1, pp. 53–63, Feb. 2017, doi: 10.1002/CITE.201600092.

- [8] Y. Zhang, M. Cui, J. Wang, X. Liu, and X. Lyu, “A review of gold extraction using alternatives to cyanide: Focus on current status and future prospects of the novel eco-friendly synthetic gold lixivants,” *Miner. Eng.*, vol. 176, pp. 107336, 2022, doi: 10.1016/J.MINENG.2021.107336.
- [9] S. Ilyas, R. R. Srivastava, H. Kim, and H. A. Cheema, “Hydrometallurgical recycling of palladium and platinum from exhausted diesel oxidation catalysts,” *Sep. Purif. Technol.*, vol. 248, pp. 117029, 2020, doi: 10.1016/J.SEPPUR.2020.117029.