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Investigation into the mineralogy and flotation performance of oxidised PGM ore

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ABSTRACT

The 2.05 Ga Bushveld Complex in South Africa, host to many lucrative ore deposits, is surprisingly pristine and unaltered given its geological age. In some areas, however, there is evidence of low temperature weathering, alteration and oxidation, most commonly observed when the ore is near surface. The Pilanesburg Platinum Mines (PPM) operation in South Africa treats ore from an open pit and routinely suffers from low and erratic platinum group element (PGE) flotation recoveries. This study investigates the effect of oxidation on the mineralogy and flotation performance of PPM "silicate reef" ore and evaluates the effect of alkyl hydroxamate (AM 28) and controlled potential sulfidisation (CPS with NaHS) as a means to improve the poor flotation performance of the oxidised ore. Oxidised PPM ore is characterised by high contents of alteration minerals resulting in abundant naturally floating gangue (NFG), high contents of Fe-oxides/hydroxides and negligible base metal sulfides. Small improvements in PGE recovery with the addition of the hydroxamate co-collector with CPS or without it are more due to the high froth stability and increased water recovery rather than any selective action of the collector. The distinctly higher Pt recovery relative to Pd recovery is linked to the mobilisation and redistribution of Pd during the oxidation of the ore.
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1. Introduction

The 2.05 Ga Bushveld Complex in South Africa is host to some of the world's major platinum group mineral (PGM) ore deposits such as the Merensky reef, UG2 reef and Platreef. Although the majority of the mines exploiting these deposits have underground workings, several open cast operations do exist, most notably the Platreef deposit. Once extracted from the ground, the ore is crushed, milled and concentrated by flotation to recover the valuable platinum group elements (PGE) that are hosted either in discrete platinum group minerals (PGM) or in solid solution with the base metal sulfides (BMS), namely chalcopyrite, pentlandite, pyrrhotite and minor pyrite (e.g. solid solution Pd in pentlandite, Osbahr et al., 2013). In some instances the ore recovered from near surface has been altered by weathering and oxidation and if the ore is treated by flotation instead of being left in situ, stockpiled or discarded, then very poor flotation recoveries are achieved.

Although few descriptions of the effect of alteration and oxidation on the mineralogy of ores from the Bushveld Complex exist, e.g. Hey (1999), numerous accounts exist for ores from the Massive Sulfide Zone (MSZ) from the Great Dyke of Zimbabwe e.g. Locmelis et al. (2010), Oberthür et al. (2003a,b, 2013). The Great Dyke is located just north of the Bushveld Complex and is similarly enriched in the PGM and BMS. Pristine unaltered PGE ore is mined from several underground operations whereas the ore derived from the top 20–30 m from surface is extensively oxidised and represents a resource close to 400Mt (Prendergast, 1988). Oxidised ores are generally characterised by a loss of Pd relative to Pt due to the increased mobility of Pd in the supergene environment that has likely been removed through surface and ground waters. The PGE themselves occur as (i) relict primary PGM, (ii) in solid solution with relict BMS, (iii) as secondary PGE alloys, (iv) as PGM oxides/hydroxides and (v) as PGE hosted by secondary oxides/hydroxides (e.g. Fe, Mn) or silicates (e.g. smectite, chlorite) (Evans et al., 1994; Hey, 1999; Oberthür et al., 2013, 2003b).

The processing of these oxidised PGE ores is not simple especially since the remobilisation of Pd alters the Pt:Pd ratio which is a key metric used in the mineral processing business case. Processing these ores by conventional flotation techniques typically results in very poor recoveries, therefore various hydrometallurgical, pyrometallurgical, and future technologies may provide alternate solutions (Evans, 2002; Oberthür et al., 2013). This negates the fact though that for some of the Bushveld operations treating oxidised ores, the

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infrastructure of flotation concentrators are already in place and therefore it is viable to explore alternative flotation reagent schemes in an effort to improve PGE recovery when oxidised ore is processed. This is the case for the Pilanesburg Platinum Mines (PPM) operation, located just west of the Pilanesburg Alkaline Complex. The PPM operation treats "silicate reef" ore (comprising the Merensky reef, Merensky footwall, Upper Pseudo reef, Pseudo reef and Lower Pseudo reef; see Viljoen et al., 1986; Viring and Cowell, 1999 for more details of the stratigraphy) and "chromite reef" (UG2 Chromitite) ore derived from an open pit and routinely suffers from low (<40%) and erratic flotation recoveries.

Preliminary studies by Becker and Ramonotsi (2012) on alternative flotation strategies showed the promise of the hydroxamate co-collector AM 28 for improving the flotation recovery of oxidised PPM platinum ore. Alkyl hydroxamate collectors have been on the market for several decades. Their use in industry has been well documented (Nagaraj, 1992). More recently improved alkyl hydroxamate collectors have found application in industry as it has become economical to process a large number of oxide ore deposits. Hydroxamates are classified as powerful collectors with the ability to selectively chelate at the surface of minerals. The product formed on the mineral surface is the result of a reaction between the collector and the metal cations emanating from the mineral (Hope et al., 2010).

AM 28, a hydroxamate collector produced by Ausmelt, is a potassium hydrogen *n*-octanohydroxamate. It is a non-hazardous product consisting of a white paste which is used in a dilute potassium hydroxide solution, is effective at pH values above 6, and forms a relatively thick layer on mineral surfaces (Hope et al., 2010). Lee et al. (2009) using AM 28 in a mixture with a traditional xanthate collector in laboratory-scale batch flotation tests achieved copper recoveries from a sulfide/oxide ore blend (containing chalcopyrite, bornite, chalcocite, malachite) which were superior to those obtained using xanthate only after controlled potential sulfidisation (CPS).

The classical method used to concentrate oxide copper-bearing ore is CPS. Sulfidisation is a process whereby a non-sulfide mineral surface is converted to a sulfide-like surface. Commonly used sulfidising agents include: sodium hydrogen sulfide (NaHS), sodium sulfide (Na_2S) and ammonium sulfide ($(NH_4)_2S$) (Lee et al., 2009). NaHS addition, for example, is used to reduce the redox potential of the pulp to between −450 and −550 mV versus a standard hydrogen electrode (Jones and Woodcock, 1978; Soto and Laskowski, 1978). Sulfidisation is usually done only after the primary sulfide minerals have been recovered, since NaHS can inadvertently depress sulfide flotation. Sulfidisation of the oxide minerals works extremely well under carefully controlled conditions in a laboratory, but on plant scale is variable due to its sensitivity to amongst other factors: conditioning time, type of collector, preparation procedures and the presence of slimes in the ore (Bulatovic, 2010).

The objective of this study is to characterise the mineralogy and flotation performance of PPM "silicate reef" ore and to evaluate the effect of oxidation on the ore from both a mineralogical and processing perspective. The performance of two different collector schemes generally used in the processing of oxidised copper ores (mixed sulfide/oxide) is also evaluated to determine their efficacy on oxidised PGM ores: alkyl hydroxamate (AM 28) co-collector and controlled potential sulfidisation. This is done in conjunction with the interpretation of the pulp and froth phases in flotation as well as the interpretation of mineralogy, focussing on the behaviour of both the valuable and gangue minerals.

2. Analytical methods

"Silicate reef" ore was sourced from a stock pile of oxidised ore at Pilanesburg Platinum Mines and approximately 500 kg of the ore with a topsize of 80 mm was sent to the Centre for Minerals Research at the University of Cape Town (UCT) for the experimental programme. The bulk sample was crushed to a topsize of 3 mm, blended, riffled and split using a rotary splitter into 3 kg samples prior to batch flotation tests. Batch flotation tests were conducted in a modified 8L Leeds flotation cell using the standard procedure as outlined in Wiese et al. (2005) at a grind of 80% passing 75 μm. Four successive timed concentrates were collected. The reagents used in the batch flotation tests are given in Table 1. Reagents were prepared by hydrating the dry product in distilled water as follows: sodium isobutyl xanthate (SIBX) 1%, AM 28 1% in dilute KOH solution, NaHS 5% and Depramin 267 1%. DOW 200 frother was dosed as required in its concentrated form. Hydroxamate collectors possess frothing properties (Basilio and Mathur, 2007), so frother was not added to tests evaluating the performance of AM 28. Each test condition was conducted in triplicate in order to produce sufficient concentrate mass for PGE assay. Error bars displayed in Figs. 1–4 represent the standard error between triplicate tests. 4E PGE assays (Pt, Pd, Rh, Au) were conducted at PPM using fire assay and gravimetry.

Based on the interpretations of metallurgical performance derived from the 4E assay data, a subset of tailings samples were selected for further elemental analysis to determine whether there was any preferential loss or recovery of Pt versus Pd. This subset of samples was submitted for Pt and Pd analysis using fire assay and ICP–OES at SGS, Johannesburg.

Mineralogical characterisation was performed on the feed and the same subset of tailings samples (as described above) by QEMSCAN at UCT to determine the bulk mineralogy; and MLA at ALS, Johannesburg to determine the PGM mineralogy. QEMSCAN samples were wet and dry screened into the following fractions; −10; +10/−25; +25/−53 and +53–75; +75/−106 and +106 μm, split and prepared into polished blocks for QEMSCAN analysis (Table 2). Samples were analysed using the Bulk Mineralogical Analysis (BMA) routine in QEMSCAN (Goodall et al., 2005; Gottlieb et al., 2000). MLA samples were prepared into unsized polished blocks and analysed using the Sparse Phase Liberation (SPL) measurement routine (Fandrich et al., 2007). Data validation was performed based on the correlation of the QEMSCAN results with chemical assays of the feed obtained using ICP–OES and a Leco Sulfur Analyser at UCT. Bulk samples were analysed using a Bruker D8 XRD with Vantec detector and quantified using the Bruker Topas Rietveld Refinement software for further validation of the QEMSCAN bulk mineralogy.

3. Results

3.1. Flotation

The total solids and water recovered from batch flotation tests for all conditions evaluated are presented in Fig. 1. The figure clearly illustrates the frothing properties of the hydroxamate collector, AM 28 (Basilio and Mathur, 2007), in that its addition resulted in increased froth stability as indicated by increased water recovery compared to equivalent tests conducted in the presence

Table 1
List of flotation reagents used in this study, and their dosages.

Reagent	Type	Dosage
Depressant (CMC)	CMC	0–750 g/t
Collector (xanthate)	SIBX	150 g/t
Hydroxamate co-collector	AM 28	300 g/t
Sulfidising agent	NaHS	As required to reduce potential to ~−550 mV
Frother	DOW 200	As required

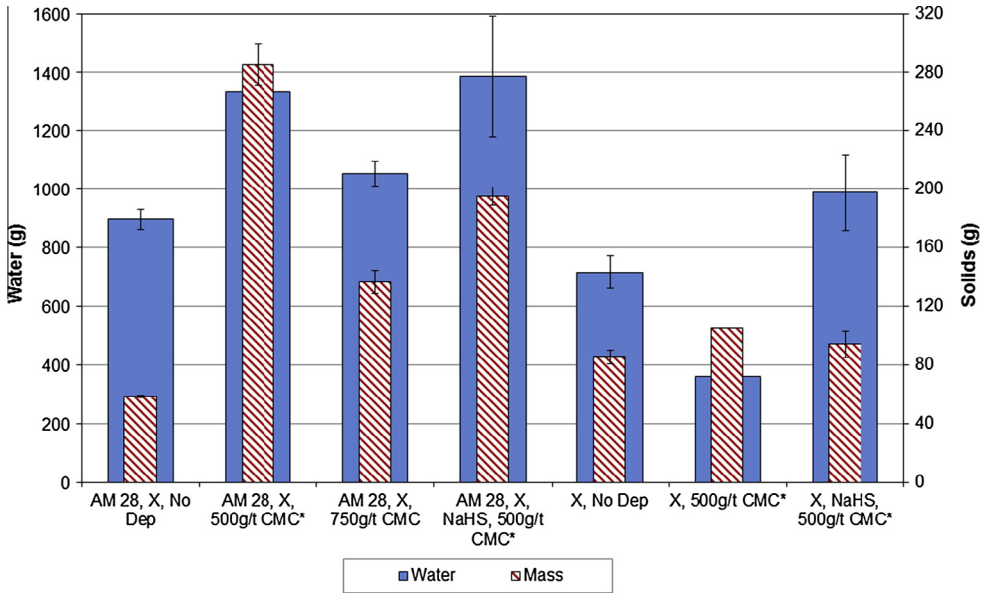


Fig. 1. Comparison of the solids – water recovery for the various batch flotation test conditions investigated. Samples highlighted with an asterisk were selected for further investigation. Error bars represent the standard error.

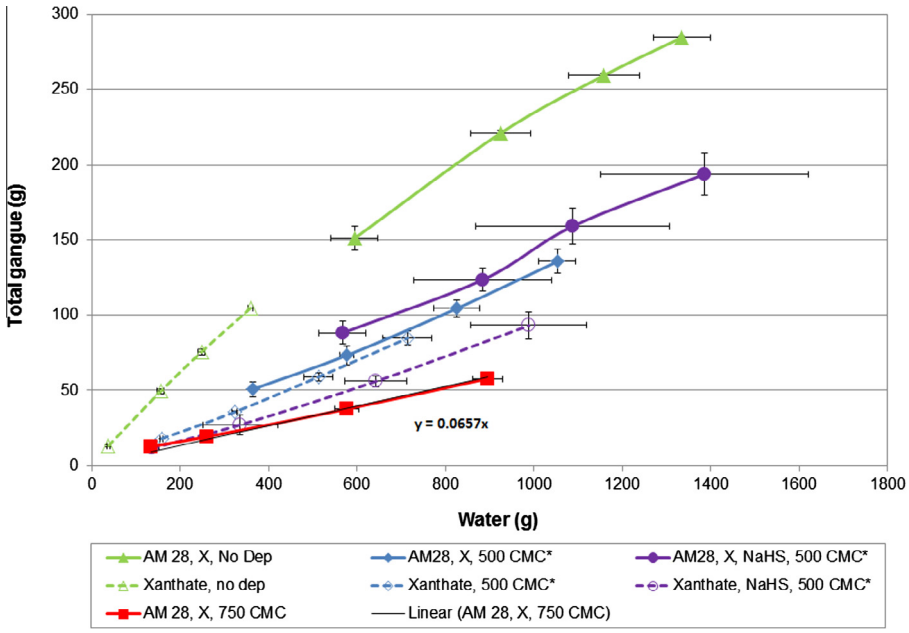


Fig. 2. Total gangue recovered as a function of water recovered. The gradient of the line for AM 28, X, 750 CMC is the entrainment function (see text for detail). Samples highlighted with an asterisk were selected for further investigation. Error bars represent the standard error.

of xanthate only. This excessive frothability, which in some tests was difficult to control, accounts for the large standard errors obtained between repeat tests. The addition of NaHS in the absence of AM 28 resulted in greater froth stability in comparison to the corresponding test without NaHS addition as indicated by increased water recovery. The impact of increased depressant addition is evident, in that as depressant dosage was increased from 500 to 750 g/t the amount of both solids and water recovered was reduced. This is due to the depression of the froth stabilising NFG present in the ore. The addition of NaHS to tests in which xanthate was used as the only collector showed the same trend as the corresponding test using AM 28 in that water recovery increased. There was, however, no corresponding increase in the amount of solids recovered.

In order to quantify entrainment and to determine the amount of NFG present in the ore, the method developed at UCT based on high depressant concentrations was used (Wiese, 2009). Tests were conducted at increasing depressant dosage to determine the dosage required to ensure the complete depression of NFG. Total gangue was determined by subtracting the amount of valuables present in the ore from the total mass of solids recovered. The gradient of the line for total gangue as a function of the amount of water recovered was determined to be the entrainability factor for the ore. This was determined to be 0.0657, and was used to calculate NFG under all other conditions evaluated. This equates to the recovery of 0.0657 g of entrained material per ml of water recovered. Valuable mineral grade and recovery are strongly dependent on the stability of the froth since the recovery

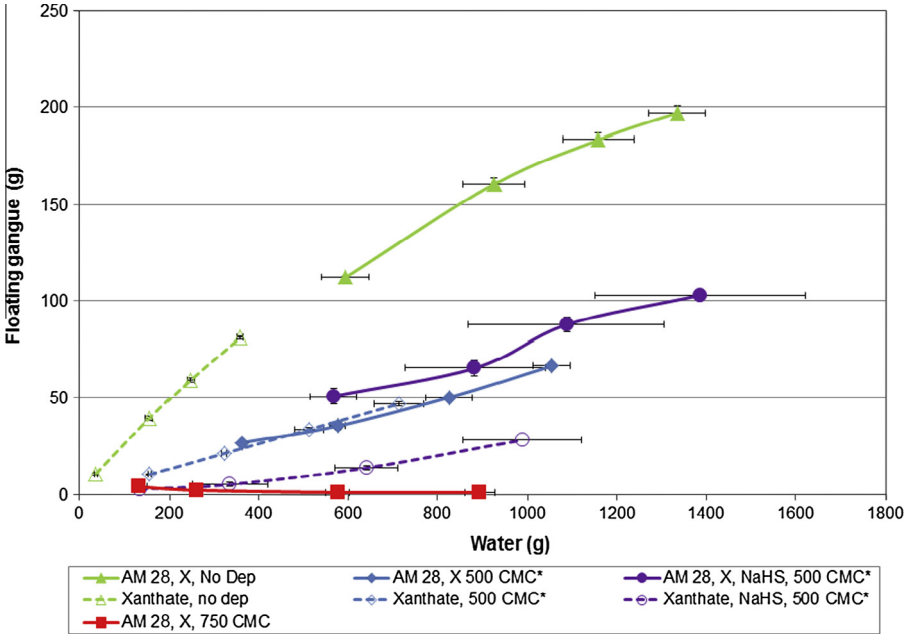


Fig. 3. Floating gangue recovered as a function of water recovered for the various batch flotation test conditions investigated. Samples highlighted with an asterisk were selected for further investigation. Error bars represent the standard error.

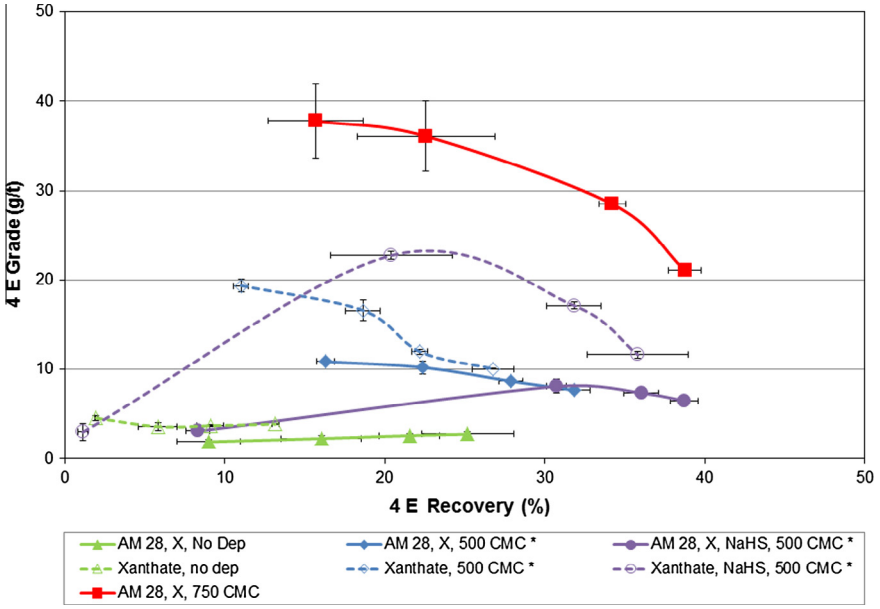


Fig. 4. 4E grade versus recovery for the various batch flotation test conditions investigated. Samples highlighted with an asterisk were selected for further investigation. Error bars represent the standard error.

of entrained gangue is directly proportional to the amount of water recovered (Engelbrecht and Woodburn, 1975; Neethling and Cilliers, 2002; Zheng et al., 2006a,b). At a depressant dosage of 750 g/t it was assumed that all NFG was depressed and gangue recovered was due to entrainment alone. Fig. 2 shows total gangue versus water recovered depicting the entrainment function. Fig. 3 compares floating gangue as a function of water recovered for all batch flotation conditions evaluated, illustrating that a depressant dosage of 750 g/t NFG recovery was by definition zero. As expected, the highest mass of NFG was recovered from the flotation test with no depressant addition (in the presence of AM 28).

The results obtained for 4E recovery as a function of 4E grade for the concentrates from all batch flotation conditions evaluated are compared in Fig. 4. The lowest 4E recovery (13%) was obtained from tests conducted using xanthate only in the absence of a depressant. Under these conditions there was maximum dilution of the concentrate grade by NFG. The addition of the AM 28 co-collector in conjunction with xanthate yielded a similar grade, but higher recovery (25%) which has been attributed to the increase in frothability accompanying the use of AM 28. Improved recoveries of the 4E were also obtained from tests conducted at a depressant dosage of 500 g/t in the presence (32% recovery) and absence of AM 28 (27% recovery). The addition of NaHS in the presence of 500 g/t depressant resulted in higher recoveries (39%) as a result of enhanced frothability due to the addition of NaHS. The very poor grade and recovery achieved with the collection of the first concentrate in these two tests with NaHS suggests that under these conditions, the valuables were less able to attach to bubbles due to the presence of large amounts of NFG which could lead to competitive bubble loading, with NFG substituting the valuable minerals on the

Table 2
Mineralogy of the feed and selected tailings samples of selected batch flotation tests in wt.% (all at 500 g/t CMC).

	Feed	Tail: AM 28, X, CMC	Tail: AM 28, X, NaHS, CMC	Tail: X, CMC	Tail: X, NaHS, CMC
Base metal sulfides	0.2	0.2	0.2	0.3	0.2
Olivine	3.8	2.1	2.3	2.4	2.3
Orthopyroxene	24.6	30.5	31.6	31.9	33.2
Clinopyroxene	10.3	11.2	10.6	10.7	10.8
Serpentine	10.6	9.2	9.4	9.4	8.8
Talc	18.0	16.6	15.3	14.8	14.5
Chlorite	5.1	5.7	5.3	5.1	5.3
Amphibole	6.8	5.5	6.1	6.1	5.9
Plagioclase	4.4	1.8	2.3	2.4	2.0
Epidote	1.2	2.0	1.9	1.9	2.2
K-feldspar	0.1	0.1	0.1	0.1	0.2
Mica	0.9	1.0	1.0	1.0	0.9
Calcite	1.0	1.0	0.8	0.8	0.9
Quartz	0.3	0.4	0.4	0.5	0.4
Chromite	0.8	0.9	0.8	0.8	0.8
Fe oxides/hydroxides	11.0	10.8	10.9	11.0	10.6
Other	0.9	1.0	0.9	1.0	0.9

pulp-phase bubbles. An increase in depressant dosage from 500 to 750 g/t CMC results in the highest concentrate grades (37.8 g/t, AM 28, X at 750 g/t CMC) due to the full depression of NFG. Given that the highest recovery was also obtained in this test provides further support for the argument that competitive bubble loading by the NFG prevents the attachment of the valuables to the bubbles.

Further Pt and Pd assays from selected flotation tests show that for the conditions investigated (Fig 5), very poor Pd recovery (⩽20% recovery) was obtained. Final Pt recovery was however slightly better (35–52% recovery). Flotation tests conducted using CPS with NaHS resulted in the highest Pt, Pd recovery both in the presence (52% Pt, 17% Pd recovery) and absence of AM 28 (52% Pt, 20% Pd recovery).

3.2. Mineralogy

The bulk mineralogical analysis of the feed and tailings samples given in Table 2 and illustrated in Fig. 6. The PPM feed ore is dominantly comprised of orthopyroxene (24.6 wt.%), clinopyroxene (10.3 wt.%), the alteration minerals (10.6 wt.% serpentine, 18.0 wt.% talc, 5.1 wt.% chlorite, 6.8 wt.% amphibole) and the Fe-oxides/hydroxides (11.0 wt.%). The BMS content of the ore is only 0.2 wt.%. The significant enrichment of the alteration minerals and the Fe-oxides/hydroxides, as well as depletion of the feed ore in the BMS are characteristics typical of altered and oxidised PGE ores (Oberthür et al., 2003b). The plagioclase content of this ore is also particularly low (4.4%) compared to other Merensky ores (e.g. Solomon et al., 2011; Vogeli et al., 2011), but this may well be due to the mining cut and the processing of "silicate reef" compared to pure Merensky reef rather than the effect of oxidation. Comparison of the bulk mineralogy of the feed with that of the tailings samples shows little variation for the different flotation conditions investigated (Table 2, Fig. 5). A slight reduction in the proportion of alteration minerals occurs from the feed (40.4 wt.%) to the tailings (35.3–37.0 wt.%), most likely due to the recovery of these minerals to the concentrate as NFG. No significant differences are however noted in the amount of alteration minerals for the different flotation test conditions that can be correlated with water recovery or NFG recovery (Figs. 1 and 3).

The PGE mineralogy (Fig. 7) of the feed sample is dominated by the PGE alloys (21% ferroplatinum, 10% PtRuFeNi) and PGE sulfides (19% laurite, 7% braggite), although significant proportions of the PGE arsenides (8% sperrylite, 7% arsenopalladinite) and tellurides (7% kotulskite) also occur. The PGM are typically very fine grained with a d50 of 7 μm. At the target grind (80% < 75 μm) only 59% liberation was achieved (Fig. 8). The majority of the unliberated PGM (Fig. 9) are locked in gangue (16%), attached to gangue (13%) or associated with the Fe-oxides (11%). The association of unliberated PGM with the BMS was negligible (<1%).

The dominant PGM reporting to the tailings are the PGE alloys, PGE arsenides, and PGE sulfides, which broadly mimics the major PGM phases present in the feed (Fig. 7). There does not appear to be any significant correlation between reagent type and the species of the PGM losses. The majority of the PGM reporting to the tailings for all the conditions investigated occur as middlings and locked particles indicating that inadequate liberation is one of the reasons for the poor performance of the PPM ore. However, there is still evidence of flotation losses of liberated particles which are of concern; e.g. loss of 42% liberated PGMs from the flotation test with AM 28, xanthate and CMC.

4. Discussion

The poor overall flotation performance of the PPM ore can be attributed to two major factors: poor liberation and the effect of

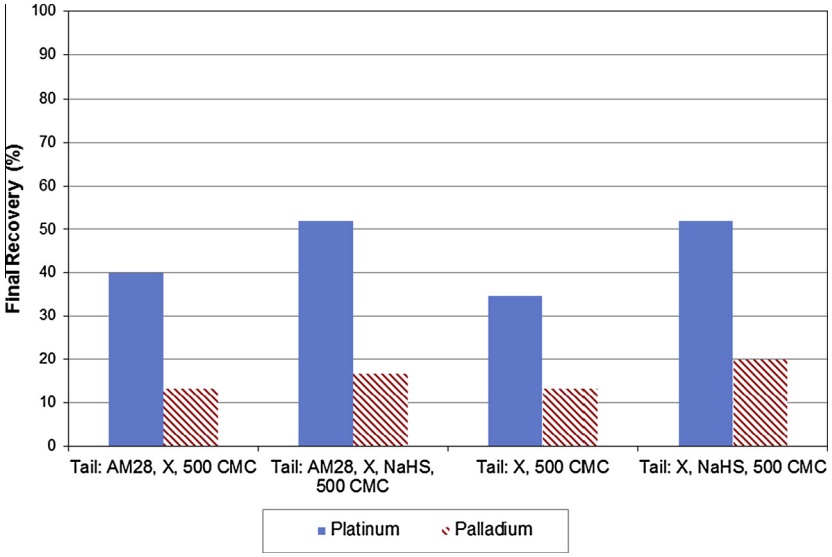


Fig. 5. Pt and Pd recovery from selected batch flotation tests (all at 500 g/t CMC).

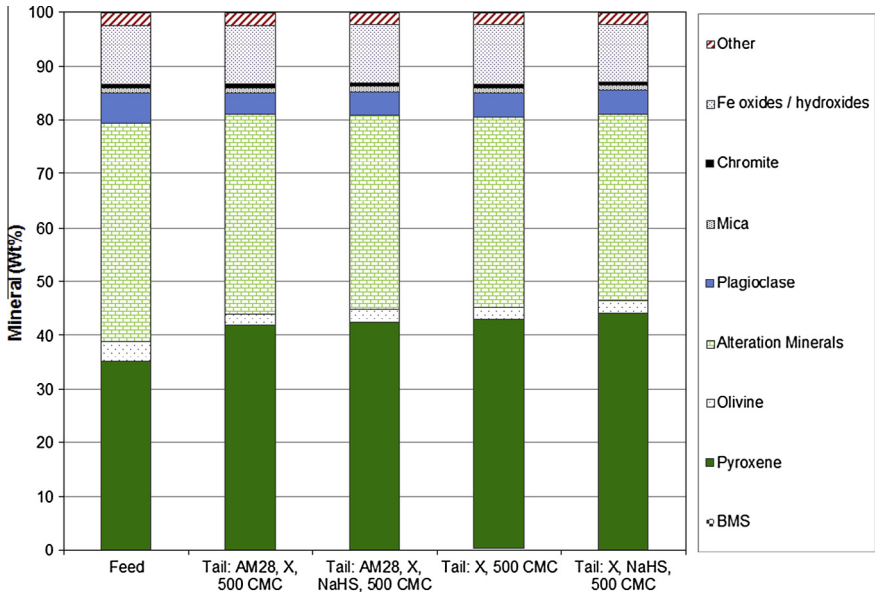


Fig. 6. Simplified bulk mineralogy of the PPM feed and selected tailings samples (from flotation tests all at 500 g/t CMC). Note that the BMS content for all samples is negligible (0.2 wt.%) and too small for representation on the scale of the graph.

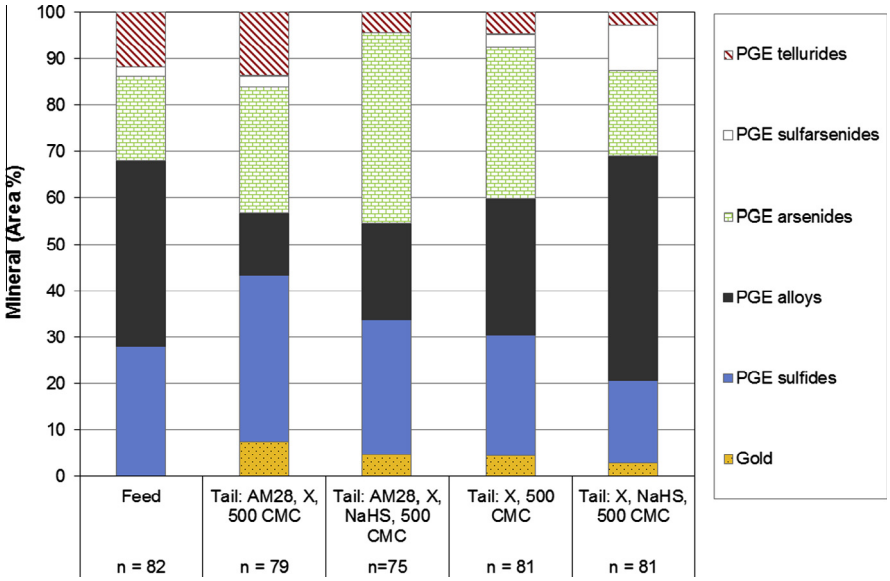


Fig. 7. Relative PGM abundance in the PPM feed and selected tailings samples (from flotation tests all at 500 g/t CMC).

alteration/oxidation. A typical Merensky PGM ore consists of PGM that show a strong association to the BMS (e.g. Schouwstra et al., 2000) and hence flotation concentrators target the recovery of composite PGM and BMS particles. Under these circumstances poor PGM liberation (defined by area% PGM in a particle) does not necessarily mean poor PGE recovery, if the PGM are locked in or associated with the coarser grained BMS. For the PPM ore however, where there are both negligible BMS in the feed (Table 2), and negligible association of PGM with BMS, then poor PGM liberation may well mean poor PGE recovery. If one assumes a good correlation between PGM liberation and PGE recovery, then for the PPM "silicate reef" ore which is 59% liberated, a PGE recovery ~60% could be expected. In this study, the highest 4E PGE flotation recovery obtained was 39%. The highest Pt recovery was 52%, and the highest Pd recovery was 20%.

4.1. Effect of oxidation on the PPM ore

The very low BMS content of the PPM ore (0.2 wt.%) and the lack of association of the BMS to the PGM are just some of the typical features of oxidised ores. The depletion of BMS in the PPM ore and the formation of Fe-oxides/hydroxides (11.0 wt.%) are also typical features of oxidised PGM ores. The PPM ore is also characterised by high amounts of phyllosilicate minerals such as talc, serpentine and chlorite which are formed through the low temperature alteration of anhydrous minerals such as orthopyroxene (Nesbitt and Bricker, 1978; Viti et al., 2005). During the alteration process, hydrous phyllosilicate minerals such as talc form along grain boundaries and cleavage planes of orthopyroxene. This results in the occurrence of composite orthopyroxene and talc particles which have been linked to the occurrence of naturally

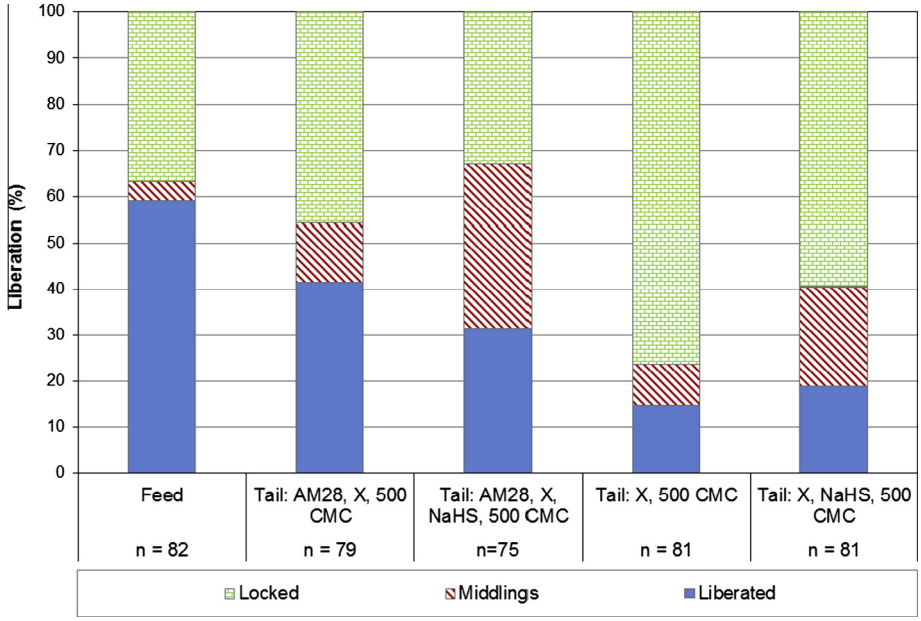


Fig. 8. Liberation of PGM in the feed and selected tailings samples (from flotation tests all at 500 g/t CMC) shown as locked (0–30 area% PGM), middlings (30–80 area% PGM) and liberated (>80 area% PGM).

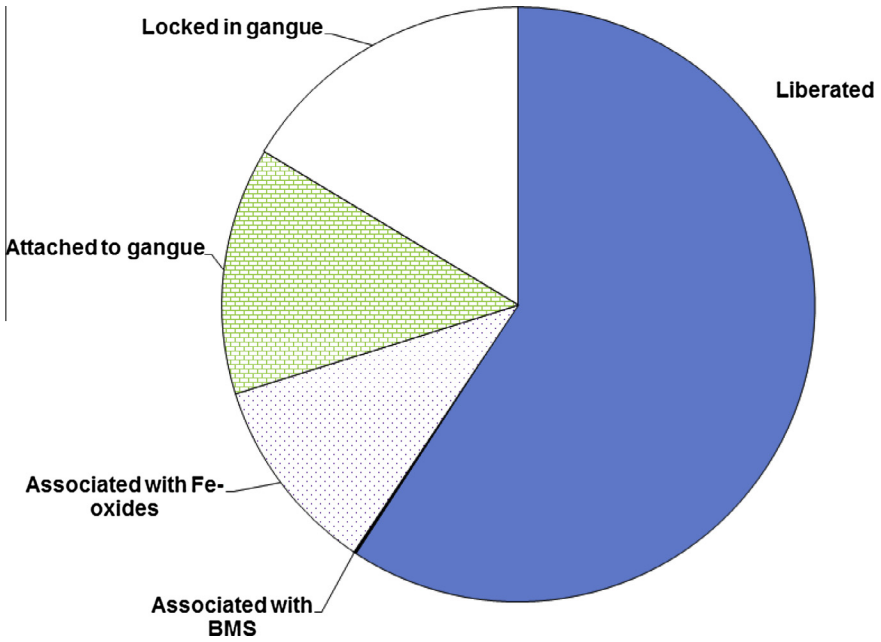


Fig. 9. Liberation and association of PGM in the feed ore, where liberated is defined as greater than 80% PGM by area. Association is given for unliberated PGM.

floating gangue (NFG: Becker et al., 2009; Jasieniak and Smart, 2009). Previous work by Wiese et al. (2007) has shown that a depressant dosage of ~300 g/t is sufficient to depress all the NFG in Merensky ore. In this study, a depressant dosage of 750 g/t was required to fully depress all the NFG (Figs. 2 and 3). If the NFG is not managed appropriately, this may lead to competitive bubble loading that ultimately prevents the attachment of valuable PGM to the bubbles (13% 4E recovery with no depressant). The improved 4E recovery at 750 g/t provides further support for the possibility of competitive bubble loading (39% recovery). The increase in recovery may also be due to the high dosages of CMC which can "slime-clean" the surfaces of the valuable minerals (Nagaraj and Ravisahnkar, 2007). This may have led to improved hydrophobicity and enhanced attachment of PGM to bubbles under these conditions.

The effect of oxidation on the PGM mineralogy itself is less clear. Due to the very low grade of the ore, the number of PGM grains analysed is low (<100) and hence interpretations based on the presence or absence of specific Pt or Pd bearing minerals is not viable. The high proportion of PGM alloys in the feed (Fig. 7), and association of PGM to Fe-oxides (Fig. 9) is however, a likely result of the oxidation of the ore. The disparity in the flotation recovery between Pt and Pd (Pt recovery was ~2.5 times higher than Pd; Fig. 5), however, can be attributed to the effect of oxidation of the ore causing mobilisation of Pd. The use of specific mineralogical techniques (e.g. EPMA, LA–ICP–MS, FEG–SEM,

synchrotron XRFS – Oberthür et al., 2013) to further characterise the PGE deportment of this ore and determine the contribution of PGE oxides/hydroxides, PGE hosted by Fe oxides/hydroxides and PGE hosted by phyllosilicate minerals is recommended to understand PGE behaviour.

4.2. Evaluation of alternate collectors

The interpretation of the relative effect of using the AM28 alkyl hydroxamate co-collector and/or performing controlled potential sulfidisation with NaHS is done by comparison with the baseline flotation conditions: xanthate with 500 g/t CMC depressant (26% 4E recovery). Even though the use of AM28 as a co-collector, NaHS as a co-collector, or AM28 combined with NaHS as co-collectors resulted in improved 4E recovery, this is attributed more to the increased frothiness and water recovery (Fig. 1) than any apparent selective action of the co-collector on the surfaces of oxidised PGM. For the different collectors to show selective action, a threefold increase in water recovery (e.g. 360 to >1000 g, Fig. 1) due to the frothiness of the co-collectors would not be expected, and losses of liberated PGM to the tailings would be minimal. The results from this study show that the losses of liberated PGMs to the tailings were actually the least for the baseline flotation test (xanthate with 500 g/t CMC, 15% liberated PGMs). The loss of liberated PGMs to the tailings (Fig. 8) for the flotation tests with AM28 as a co-collector, NaHS as a co-collector, and AM28 combined with NaHS as co-collectors is therefore attributed to the unselective recovery of PGM to the concentrate caused by the increased froth stability and water recovery.

Although CPS sulfidisation has previously been investigated as a potential remedy when treating oxidised Merensky ore by (Newell et al., 2006, 2007), their study focused on the BMS minerals, namely pentlandite, chalcopyrite and pyrrhotite rather than the PGM. In addition, the study used microflotation as a means to investigate flotation performance which serves as a useful indication of single mineral floatability, but does not take into account the effect of the froth phase (Bradshaw and O'Connor, 1996). Nagaraj (1992) have also reported the effectiveness of alkyl hydroxamate collectors in PGM flotation, but all the ore types investigated contained significant BMS (e.g. pentlandite, pyrrhotite) and no indication of the PGM mineralogy is given. The possibility remains that the success of the alkyl hydroxamate in some cases is related to the recovery of PGM based on their association to the BMS. For PGM ore types comprising a suite of PGM species; sulfides, alloys, tellurides, arsenides; the nature of the interaction of the collector with the mineral surface will be highly variable and dependent on the crystallography, and both the anionic non-metal (S) or semi-metal (As, Te) components, as well as the metal (PGE) component. A clearer understanding of the effect of oxidation on the surfaces of these different PGM species is therefore needed before appropriate treatment strategies to recover oxidised PGM become a real possibility.

5. Conclusions

The objective of this study was to investigate the effect of geological oxidation/alteration on the mineralogy and flotation performance of PPM "silicate reef" ore. Oxidised PPM ore is characterised by high contents of alteration minerals (talc, serpentine, chlorite), Fe-oxides/hydroxides and negligible BMS. The PGM were 59% liberated with PGE sulfides (27.7%) and PGE alloys (40.1%) occurring as the dominant species present in the feed. Overall, poor flotation performance of the ore was obtained (27% 4E recovery), which is attributed to a combination of poor liberation, the effect of oxidation on the PGM, and the lack of BMS association to the PGM (recovery as composite particles). Pt recovery was 2.5–3 times higher than Pd recovery due to the mobilisation and redistribution of the Pd during the oxidation process.

The high concentration of alteration minerals (40.4 wt.%) in the PPM ore was manifested as abundant naturally floating gangue (NFG) requiring depressant addition up to 750 g/t (CMC) during flotation. If the NFG is not satisfactorily controlled, PGE recoveries are severely affected (13% 4E recovery) due to competitive bubble loading preventing the attachment of the valuable PGE to the bubbles.

The use of alkyl hydroxamate with CPS and without CPS as co-collectors resulted in improved PGE recovery (up to 39% 4E recovery) which is attributed more to the increased froth stability and water recovery rather than the selective recovery of PGM.

The use of an integrated approach in the interpretation of the flotation performance involving consideration of the behaviour of the pulp and froth phases, and the valuable and gangue mineral phases has been beneficial to the overall interpretation of the performance of the PPM ore.

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