Industrial application of ammonia-assisted cyanide leaching for copper-gold ores

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ABSTRACT

A description of the agitation leach plant used for treating gold-copper ores at Anglo Asian Mining's Gedabek mine in Azerbaijan is given. The plant employs a unique resin-in-pulp system for gold recovery. The integration of the agitation leach plant into the existing heap leach/ADR/SART operation is described. Operational problems caused by increasing amounts of copper in the Gedabek gold ores are outlined. Previous work on the use of ammonia in copper-gold cyanide leach systems is reviewed. The test work leading to the introduction of ammonia in the industrial leaching plant is described. In the plant it was found that the extraction of copper was halved by the use of ammonia, thus leading to significant reductions in the amount of cyanide consumed in the full-scale leaching system.

1. Introduction

In July 2013, Anglo Asian Mining inaugurated a new agitation leach and resin-in-pulp gold ore processing plant at its Gedabek gold mine in western Azerbaijan. A few months after the start-up of this plant, changes in the ore mineralogy resulted in higher copper grades in the feed than originally envisaged. Although the resin-in-pulp plant coped adequately with the resulting high copper concentrations in the leach liquors, so that gold production was not adversely affected, consumption of cyanide by the soluble copper caused unacceptably high operating costs. In order to alleviate this situation, ammonia was introduced into the leaching circuit to reduce the cyanide consumption. This paper describes both the design and operation of the new leach-RIP plant and the successful incorporation of ammonia into the process.

2. Agitation leach-RIP plant

2.1. Background

In May 2009, Anglo Asian Mining started open pit mining for gold on its Gedabek property in the Lesser Caucasus mountains in western Azerbaijan. The ore body at Gedabek is a complex copper-gold porphyry deposit. Ore from the Gedabek mine, which initially was mainly oxidic in nature, was treated by cyanide heap leaching to produce a pregnant leach liquor, from which gold was extracted by fixed-bed resin ion exchange. Resin ion exchange was used at Gedabek, instead of conventional activated carbon, because of the elevated copper concentrations in the leach liquors from the heaps, which typically contained about 1000 mg/L Cu. In spite of the high Cu/Au ratios in the Gedabek leach liquors, which usually ranged from 500 to 1000, the excellent selectivity of the Minix resin used in the process was such that the Cu/Au ratio on the loaded resin was about one. In order to prevent copper from building up in the recirculating leach liquors, a SART plant (Sulphidisation-Acidification-Recycling-Thickening) was incorporated in the circuit to remove copper from solution as a copper-silver sulphide precipitate. A full description of the operations at Gedabek, prior to the introduction of the agitation leach plant, has been published (Hedjazi and Monhemius, 2014).

As mining progressed at Gedabek, the ore became less oxidised and harder in nature, with the result that gold recoveries in the heaps started to decrease. In 2012, it was decided that agitation leaching should be introduced to operate in parallel with heap leaching, with high grade ore (> 1 g/t Au) going to agitation leaching and low grade ore (< 1 g/t Au) going to heap leaching. Accordingly, a 100 t/h stand-alone agitation leach plant, including resin-in-pulp solution processing, was designed, built, and commissioned by July 2013. In the next section, the design of this plant and its incorporation into the existing heap leach process circuit is described.

2.2. Leaching-RIP plant

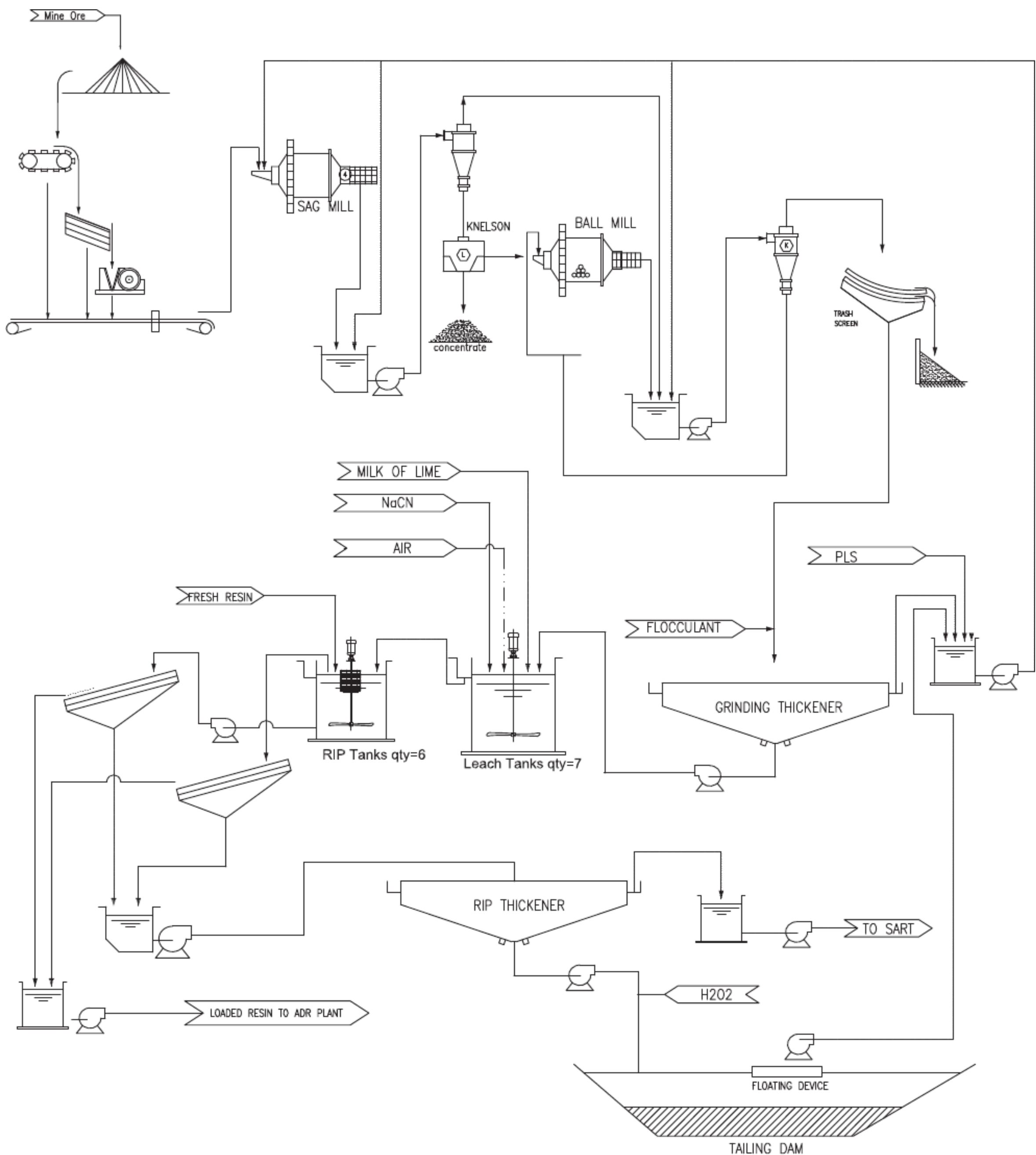The flow sheet of the agitation leach plant at Gedabek is shown in

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Fig. 1. Agitation Leach Plant flow sheet.

Fig. 1. The design is conventional, except for the resin-in-pulp section, which as far as is known, was unique at the time for this type of application.

2.2.1. Comminution circuit

Mined ore from the stockpile is fed into a silo by a front-end loader. The broken ore is moved from the silo by an apron feeder over a vibrating grizzly, with the oversize passing through a jaw crusher (1100 × 900 mm). Undersize and crushed ore at minus 15 cm size is conveyed to the SAG mill (5000 × 2800 mm, 750 KW), which reduces the ore to minus 1 mm. Return water from the tailings dam is added to the SAG and ball mills to maintain the correct pulp densities. The SAG mill discharge is cycloned, with a cut at 75 μm (P$_{80}$), with part of the underflow passing to a continuous Knelson concentrator (CVD64) to remove any particles of free gold. After the Knelson, the cyclone underflow goes through a ball mill (4000 × 6000 mm, 1.4 MW), with the cyclone overflow joining the ball mill discharge. The combined ball mill discharge is then cycloned again, also with a cut at 75 μm. The overflow from these cyclones is the main feed to the leach tanks, which is first thickened to 40 wt% solids in the leach thickener, before passing into

the first leach tank. The underflow from these cyclones is recycled to the ball mill. The Knelson concentrate is leached in a separate batch tank with a strong cyanide solution (> 2000 mg/L CN) for sufficient time to fully dissolve all the free gold. Once dissolution is complete, the contents of the batch leach tank are added into the main grinding circuit.

2.2.2. Leaching

Underflow from the leach thickener (40 wt% solids) is pumped to the first leach tank at a rate of 160 m^3/h. There are seven leach tanks in series; each tank is 11.5 m in diameter and 12 m in height, with a working capacity of 1100 m^3. The tanks are constructed from mild steel and each tank is agitated by a double stage 4-blade propeller, diameter 3.8 m, driven by a 55KW electric motor. The residence time of the pulp in the leaching train is 48hr. Cyanide and milk of lime for pH control are added in Tank 1 and further additions of cyanide are made in Tanks 3, 5 and 7. Air is blown into all tanks.

2.2.3. Resin-in-Pulp (RIP) circuit

Leached pulp from leach tank 7 flows directly to the first tank in the resin-in-pulp section, together with the PLS solution from the heap leach operations. There are six tanks in this section, each of 150 m^3 working capacity, 5.5 m in diameter by 8.5 m in height, and each is equipped with a Kemix AAC Pumpcell contactor, MSP(P) type. Each contactor holds 6 m^3 of resin (DOWEX XZ-91419, particle size 760–1200 μm). The RIP tanks are operated in series in a carousel-fashion: fresh leach pulp enters the first RIP tank and barren pulp exits from the sixth tank, after a total residence time of two to two-and-a-half hours. When the resin in the first tank is fully loaded, that tank is taken off-line by directing the fresh leach pulp to the second tank in the series, it consequently becoming the number one tank, of five then on-line. The loaded resin in the off-line tank is discharged and pumped to the elution column for stripping and subsequent gold bullion production and then the tank is recharged with freshly-eluted resin. This procedure takes about four hours and when completed, the recharged tank is brought back on-line in position number six in the carousel. As soon as the resin in the new number one tank is fully loaded, this procedure is repeated, and so on, so that at any given time, either five or six RIP tanks are on-line. Barren pulp from the last tank in the series flows into the tailings thickener, where it is thickened to 50–52% solids, with the underflow being pumped to the tailings dam, while the overflow goes to the SART plant for recovery of copper and silver.

2.3. Integrated process circuit

The agitation leach plant had to be integrated into the existing heap leaching and solution processing operations in a manner that minimized the use of extra fresh water, because of the positive overall water balance experienced at the Gedabek property due to the prevailing weather conditions at its elevated (1600 m ASL), mountainous site.

The solution flow sheet of the integrated plant is shown in Fig. 2. Return water is pumped from the tailings dam to the process water holding tank, where it joins the overflow from the leach thickener. Water from this tank is pumped to the grinding circuit where it is mixed with fresh ore by adding it to the SAG mill and ball mill feeds. After agitation leaching and gold extraction by RIP, water is recovered from the barren tailings in the tailings thickener. Part of the overflow from this thickener is passed through the SART plant (100 m^3/h), where dissolved copper and silver are recovered as a copper-silver sulphide precipitate. The underflow from the tailings thickener is pumped to the tailings dam. Water leaving the SART plant rejoins the remainder of the tailings thickener overflow and flows to the barren leach solution pond (BLS). Water from this pond is used as the feed to heap leaching. Pregnant solution from the heaps is collected in the pregnant leach solution pond (PLS), from where it is pumped back to enter the agitation leach system just ahead of the RIP tanks, so that both heap-leached

and agitation-leached gold is recovered by the RIP system.

2.4. Operational problems

The Gedabek ore body is a very complex, highly altered and faulted, copper-gold porphyry, which is difficult to model accurately. Copper grades in particular vary widely across the ore body, from zero to 1.5 wt % Cu, or even higher. The copper mineralization comprises the usual suite of copper oxides, carbonates, simple and complex sulphides. The mining method used at Gedabek involves drilling on a 5 m × 5 m grid pattern ahead of blasting to define blocks of mineral as either, waste, low Au grade ore, high Au grade-low copper ore, or high Au grade-high copper ore. In the early years of operation, distinction was made between oxide and sulphide ore blocks, but currently there is little true oxide ore left, except for isolated pockets that appear amongst the now mainly sulphidic ore. Once identified, the blocks are mined selectively and then sent to various destinations, depending on their classification: waste to the dumps; low grade ore (< 1 g/t Au) to heap leaching; high grade-low Cu ore (> 1 g/t Au) to agitation leaching; and high grade-high Cu ore to the sulphide ore stockpile. The latter stockpile will be treated in due course through a flotation plant, which is currently under construction at Gedabek.

Before the introduction of the agitation leach plant, when only heap leaching was used at Gedabek, the copper content of the ore was not a serious problem. In spite of the fact that much of the copper in the upper levels of the mine was in the form of oxide and carbonate minerals, which are highly soluble in cyanide solutions, the concentration of copper in the heap leach solutions was typically around 1 g/L Cu. While this is much higher than in most other gold heap leach operations, the combination of gold-selective ion exchange and SART technology used at Gedabek enabled these high copper solutions to be treated with relative ease, with comparatively low overall cyanide consumptions, which were typically in the range of 1–2 kg CN/t ore. However, since the advent of the agitation leach plant, copper has become a much more important issue in the operation of the Gedabek site. Grinding the ore to −75 μm for agitation leaching, instead of crushing it to −25 mm for heap leaching, exposes much more of the copper minerals to direct contact with the cyanide leach solutions, thus increasing the rate and extent of dissolution of copper. Furthermore, as the mine goes deeper, the copper grades of the ore are tending to increase. The result of these factors was that the consumption of cyanide and hence the overall operating costs for producing gold became unacceptably high, so that steps had to be taken to modify the process to bring down the operating costs. To do this, a little-used technique that was invented at the beginning of the 20th century in the early years of cyanide leaching was revisited. The technique involves the addition of ammonia into the cyanide leaching system to suppress the dissolution of copper, without adversely affecting the extraction of gold, as described in some detail in the next section of this paper.

3. Ammonia assisted cyanide leaching

3.1. Previous work

The use of ammonia to reduce the consumption of cyanide during the leaching of gold from copper-bearing ores was patented in 1902 by Hunt (1902), during the first period of intense R&D activity in gold metallurgy that followed the invention of the cyanide process by J.S. MacArthur and the Forrest brothers in the late 1880s. The ammonia-cyanide process was used commercially by Hunt to treat tailings from the Comstock Lode in Nevada and cupriferous gold ore at Dale, California. After the First World War, interest in the Hunt process died and was not revived again until the 1990s, during the second very active period of gold R&D that began nearly a century after the first period, following the steep rises in the price of gold in the early 1980s.

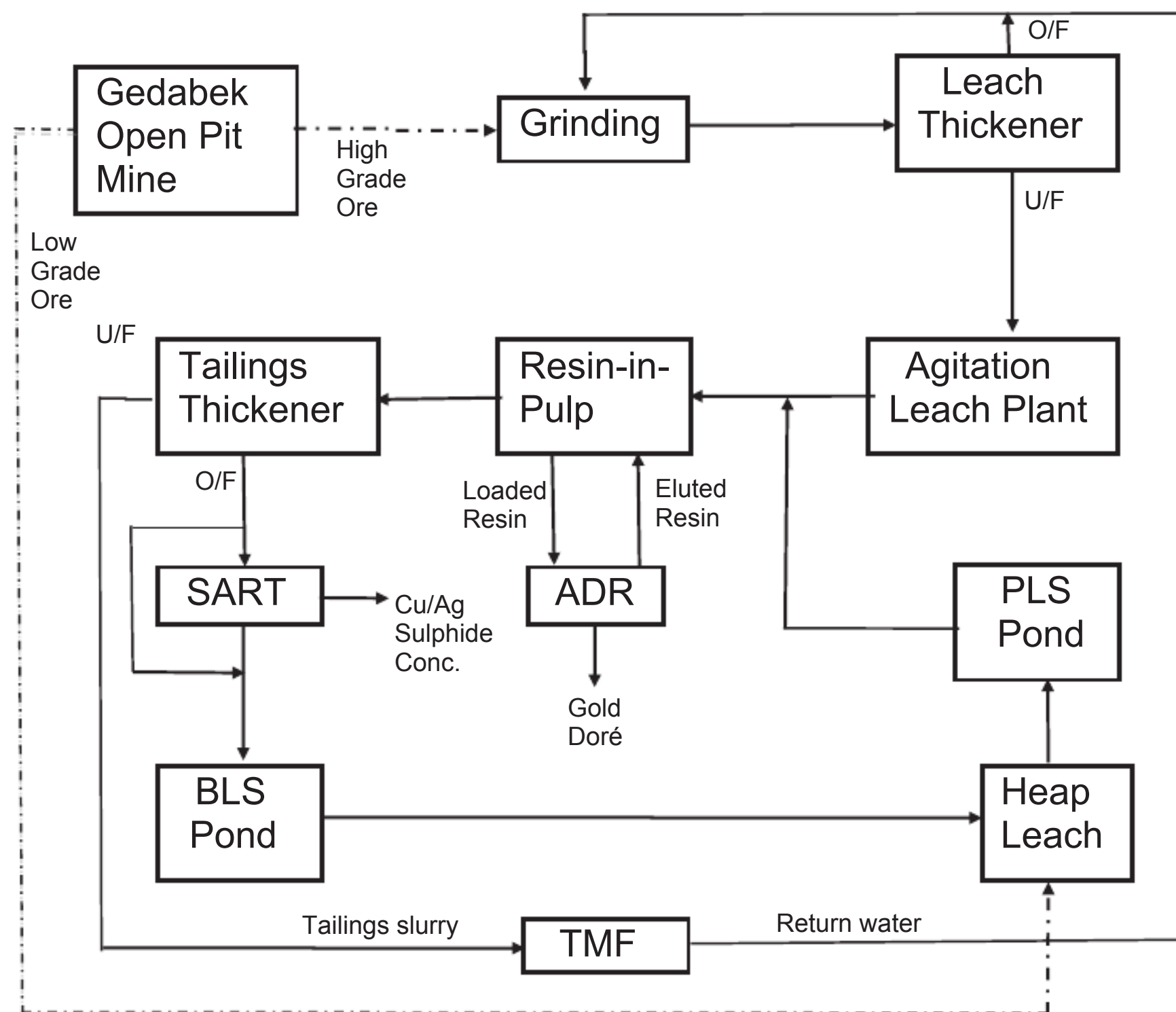Much of the recent research work to elucidate the mechanisms of

Fig. 2. Integrated HL/AGL/ADR flow sheet.

the behaviour of ammonia in the cyanidation of copper-gold ores was carried out by the late Dr David Muir and his colleagues at the Parker Centre in Perth, Western Australia and the current state-of-the-art was admirably summarized in a comprehensive review by Muir published in 2011 (Muir, 2011). More recently, further experimental work on the copper-gold-ammonia-cyanide leaching system has been published by Bas et al. (2012, 2015). As reported by Muir, two commercial plants using the Hunt process to treat gold-bearing copper tailings were in operation in the early 1990s - one in Western Australia and the other in Mauritania (Costello et al., 1992). Both these plants closed down after a few years of operation and then there was another hiatus in the industrial application of the Hunt process until 2014, when ammonia was introduced into the plant at Gedabek, as described in this paper.

3.2. Chemical principles

It is well established that oxidised copper minerals and secondary copper sulphide minerals readily dissolve in cyanide solutions, whereas primary copper sulphide minerals are much less soluble in cyanide (Hedley and Tabachnick, 1968, Deschênes et al., 2012). The most stable copper species in gold cyanide leach solutions is the tri-cyano-cuprous complex, $Cu(CN)_3^{2-}$, as illustrated in the Eh-pH diagram in Fig. 3, where X marks typical solution conditions used for leaching gold.

Dissolution of the various oxide and secondary sulphide copper minerals during cyanide leaching can be represented as follows:-
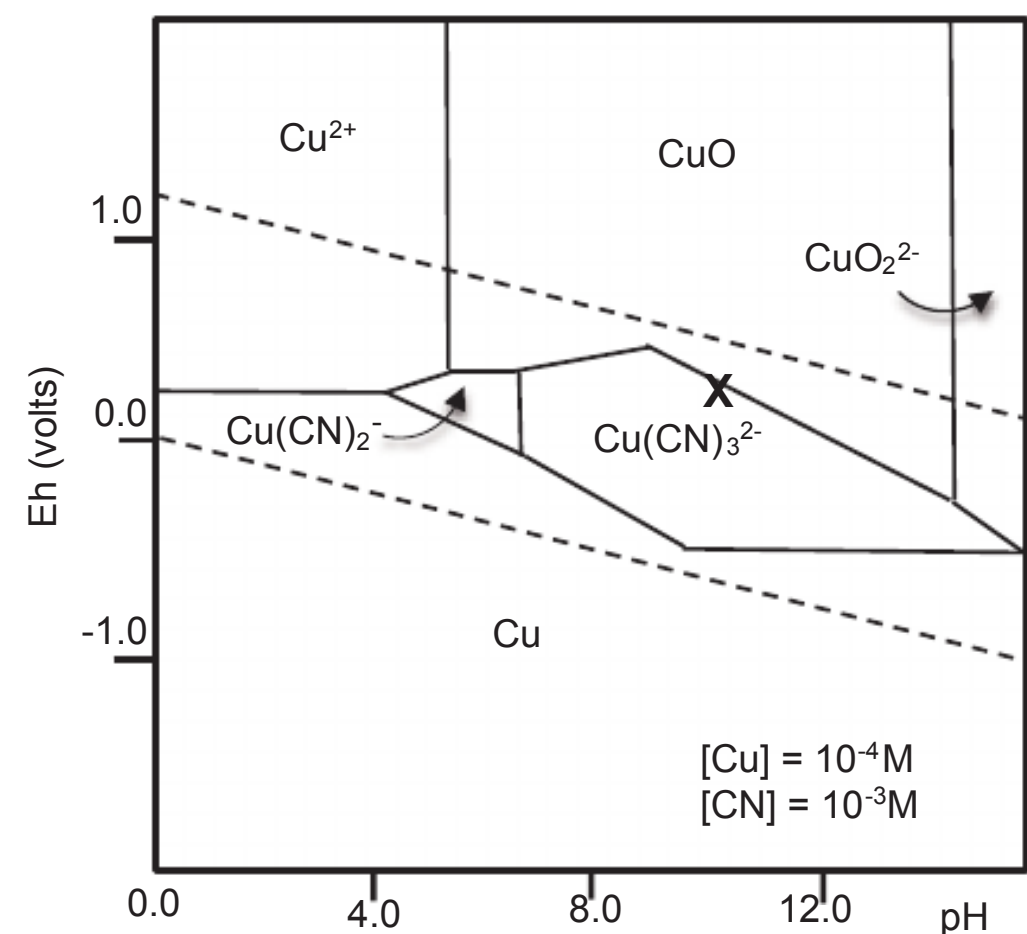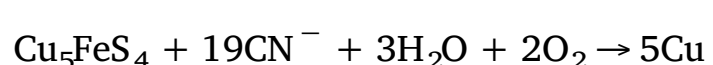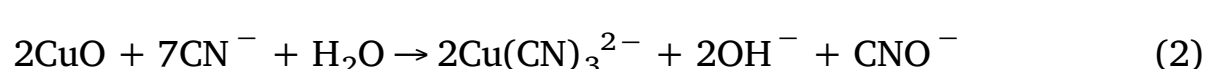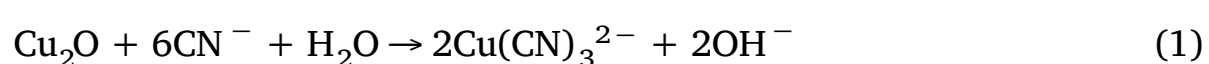
$$Cu_2O + 6CN^- + H_2O \rightarrow 2Cu(CN)_3^{2-} + 2OH^- \tag{1}$$

$$2CuO + 7CN^- + H_2O \rightarrow 2Cu(CN)_3^{2-} + 2OH^- + CNO^- \tag{2}$$

$$Cu_2S + 7CN^- + 0.5O_2 + H_2O \rightarrow 2Cu(CN)_3^{2-} + 2OH^- + CNS^- \tag{3}$$

$$2CuS + 8CN^- + 0.5O_2 + H_2O \rightarrow 2Cu(CN)_3^{2-} + 2OH^- + 2CNS^- \tag{4}$$

$$Cu_5FeS_4 + 19CN^- + 3H_2O + 2O_2 \rightarrow 5Cu$$

Fig. 3. Cu-CN-H$_2$O Eh-pH diagram (After Osseo-Asare et al., 1984).

$$(CN)_3^{2-} + 5OH^- + 4CNS^- + FeOOH \tag{5}$$

Hence, each mole of soluble copper (Cu(I)) will consume at least three moles of cyanide, or, in concentration terms, 1 g/L Cu will consume a minimum of about 1.2 g/L cyanide.
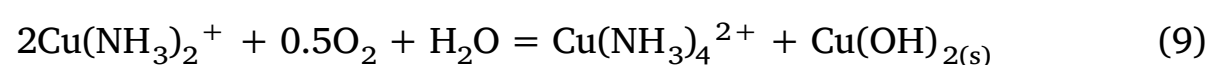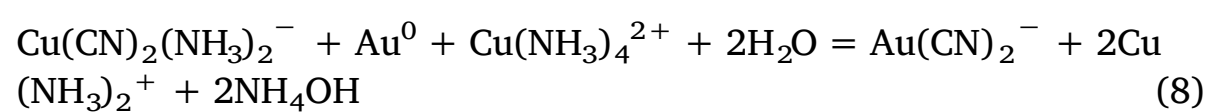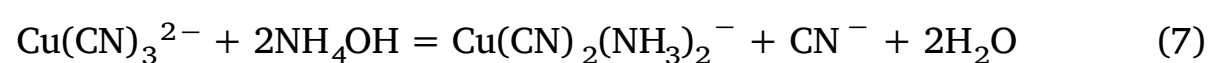
The effects of adding ammonia during the cyanidation of copper––gold ores are complex, but under the right conditions, the consumption of cyanide can be substantially reduced. This is generally ascribed to the oxidation of Cu(I) in the cuprous cyanide complexes and the stabilisation of Cu(II) in solution, due to the formation of the tetra-ammino-cupric complex ion, $Cu(NH_3)_4^{2+}$, thus releasing all the cyanide that would otherwise be bound into the cuprous cyanide complex ions.
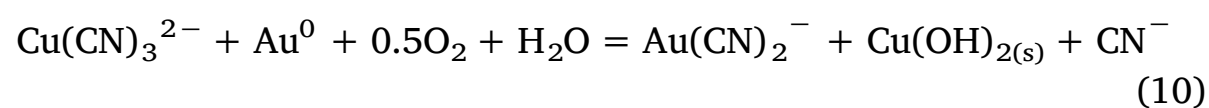
Depending on the pH and the ammonia concentration, some of the copper may be precipitated from solution as copper(II) hydroxide, as shown in Eq. (6):-

$$2Cu(CN)_3^{2-} + 4NH_3 + 0.5O_2 + H_2O \rightarrow Cu(NH_3)_4^{2+} + Cu(OH)_2 + 6CN^- \tag{6}$$
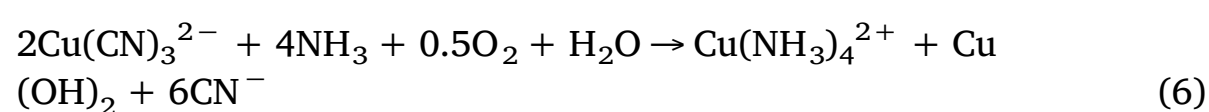
Eq. (6) is adequate to illustrate the overall beneficial effect of ammonia, but in reality, the situation is considerably more complicated, due to the many intermediate and mixed complex ions that can be formed in the Cu-CN-NH$_3$-H$_2$O system (Muir et al., 1993). Fundamental studies of the speciation, kinetics and electrochemistry of this system indicate that mixed copper(I) cyanide-ammonia complexes react with gold and provide the cyanide for gold complexation, whilst either O$_2$ or Cu(NH$_3$)$_4^{2+}$ act as oxidants (Muir, 2011). A possible sequence of reactions resulting in the dissolution of gold in the ammoniacal system is shown below in Eqs. (7)–(9). Eq. (7) illustrates the formation of a cuprous dicyano-diammine complex, which can react with gold and solubilise it as the familiar aurous dicyano complex ion, as shown in Eq. (8). The oxidant for this reaction is shown as Cu(NH$_3$)$_4^{2+}$ which itself can be regenerated by oxidation with atmospheric oxygen and thus it can act as an oxidation catalyst for O$_2$, see Eq. (9). The overall effect of the chain of reactions caused by the addition of ammonia can be illustrated by summing Eqs. (7)–(9), to give Eq. (10), which shows that ammonia effectively acts as a catalyst, enabling the oxidation of cuprous copper by oxygen, which subsequently precipitates as cupric hydroxide, thus removing copper from solution, whilst gold is also oxidised and reacts with cyanide released from the cuprous complex ion to form the soluble aurous dicyano complex ion.

$$Cu(CN)_3^{2-} + 2NH_4OH = Cu(CN)_2(NH_3)_2^- + CN^- + 2H_2O \tag{7}$$

$$Cu(CN)_2(NH_3)_2^- + Au^0 + Cu(NH_3)_4^{2+} + 2H_2O = Au(CN)_2^- + 2Cu(NH_3)_2^+ + 2NH_4OH \tag{8}$$

$$2Cu(NH_3)_2^+ + 0.5O_2 + H_2O = Cu(NH_3)_4^{2+} + Cu(OH)_{2(s)} \tag{9}$$

Summing (7)–(9):

$$Cu(CN)_3^{2-} + Au^0 + 0.5O_2 + H_2O = Au(CN)_2^- + Cu(OH)_{2(s)} + CN^- \tag{10}$$

3.3. Use of ammonia in the plant

3.3.1. Bottle roll tests
3.3.1.1. Cyanide consumption by copper. The results of some bottle roll tests on typical ore samples from the Gedabek mine are shown in Table 1. As can be seen in the penultimate column of this table, the cyanide consumption varies from 2 to 9 kg/t ore. There is a direct relationship between cyanide consumption and copper dissolution, which is illustrated in Fig. 4, where the molar concentration of dissolved copper is plotted against the quantity of cyanide consumed.

It can be seen that there is a linear relationship between these molar variables with a slope of 3, which is as expected if the major copper
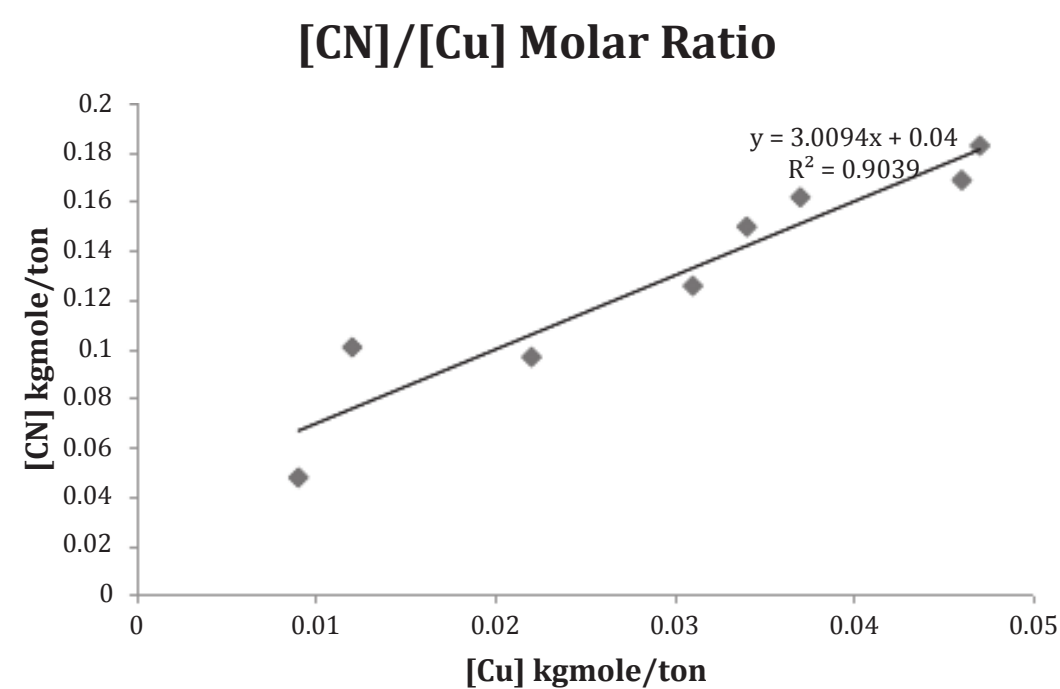

Fig. 4. Molar ratio of cyanide consumption vs. copper dissolution.

species in solution is Cu(CN)$_3^{2-}$, as shown in Eqs. (1)–(5).

3.3.1.2. Effect of ammonia additions. An extensive series of bottle roll tests was carried out to examine the effects of adding ammonia to the Gedabek leach solutions. For each test, 500 g of ground ore taken from the grinding thickener underflow was leached with 1 L of process solution. The process solutions always contained copper, generally in the range 1000–2000 mg/L Cu, together with the equivalent amount of cyanide to form the Cu(CN)$_3^{2-}$ complex, as illustrated by Fig. 4. The amount of free cyanide in these solutions varied, ranging from zero to 1000 mg/L NaCN. The effects of adding ammonia only, ammonia plus extra cyanide, and ammonia plus hydrogen peroxide, were investigated.

The outcome of these bottle roll experiments was to show that adding ammonia at concentrations where the molar ratio of ammonia to copper was two or greater resulted in increases in gold extraction of at least 10%. The addition of a small amount of hydrogen peroxide to the ammoniacal leach solutions generally increased gold extractions by a further few percent. In contrast to the behaviour of gold, it was found that the concentrations of copper in the final leach solutions decreased when ammonia was used and generally decreased further still when hydrogen peroxide was used in addition to ammonia. These effects are broadly in line with the reactions shown in Eqs. (6)–(10). Thus the addition of ammonia facilitates reactions (7) and (8) and the addition of hydrogen peroxide assists the role of atmospheric oxygen in the oxidation of cuprous copper to cupric, followed by hydrolysis to give cupric hydroxide, shown in reactions (9) and (10).

3.3.2. Plant results
The agitated leach plant at Gedabek was started up in July 2013 and for the first year it was run without ammonia additions to the leach tanks. At start up, the copper concentration of the ore feed was about 0.4% Cu and the cyanide consumption was about 4 kg/t ore. During the first year, copper in the ore feed gradually increased, as did the cyanide consumption, until by July 2014, the ore was running at 0.65% Cu and

Table 1
Bottle Roll Test Results showing Cyanide Consumption vs. Copper Dissolution.

Assay			Recovery			Copper	Dissolved	NaCN	Consumed
Au (mg/L)	Ag (mg/L)	Cu (wt%)	Au (%)	Ag (%)	Cu (%)	(kg/t ore)	(kgmole/t ore)	(kg/t ore)	(kgmole/t ore)
0.36	10.44	0.359	87.4	75.7	70.9	2.140	0.034	7.35	0.150
0.36	11.36	0.483	81.2	45.8	51.9	2.966	0.047	8.97	0.183
0.79	13.92	1.138	76.5	72.8	12.9	1.388	0.022	4.73	0.097
0.90	4.04	0.430	83.4	75.9	17.7	0.747	0.012	4.94	0.101
1.04	13.00	0.084	88.6	86.2	62.4	0.594	0.009	2.33	0.048
1.13	15.04	0.791	81.0	62.5	35.3	2.932	0.046	8.27	0.169
1.18	22.56	0.663	82.9	55.3	31.6	2.366	0.037	7.96	0.162
2.00	23.98	1.352	77.5	52.2	13.7	1.959	0.031	6.16	0.126

Table 2
Agitation Leach Plant statistics for the periods before, during and after the use of ammonia in the cyanide leach.

Period	Months	1–13	14–27	28–37
Av NaCN consumption	kg/t	5.8	7.2	5.8
Av Cu in ore	wt%	0.55	0.85	0.66
Av Cu dissolution	%	41.9	21.1	24.2
Av Au extraction	%	72.6	72.5	77.4
Av NH$_3$ consumption	kg/t	0	5.9	0
Av Cu in solution	mg/L	1516	1278	907

the cyanide consumption had reached around 7 kg/t ore. By this time, it was clear that the copper concentration of the available ore would remain high for at least the next twelve months and so it was decided to introduce ammonia into the leaching system.

The addition of ammonia led to an immediate decrease in the amount of copper leached from the ore and consequently, a decrease in the concentration of copper in the recirculating leach solutions. The behaviour of copper in the plant was similar to that observed in the bottle roll experiments. However in the case of gold, the increase in extraction seen in the bottle roll experiments was not reproduced in the plant: in the first year of operation, the average gold extraction was 72.6%; over the next thirteen months, while ammonia was being added to the leach solutions, the average gold extraction remained constant at 72.5%. Thus the advantage gained by the introduction of ammonia into the plant was not due to an increase in the extraction of gold, but instead it was due to the decrease in the extraction of copper, which in turn reduced the overall consumption of cyanide, in spite of the increased copper concentration in the ore feed, as illustrated in Table 2, below. In this table, the statistics of the leach plant are given for three periods of time, namely: the first thirteen months of operation before ammonia was introduced; secondly, the next thirteen month period when ammonia was used; and finally, the nine month period after the cessation of ammonia use.

It can be seen from Table 2 that, in the first period of operation, the average copper grade of the ore was 0.55 wt% Cu and the average cyanide consumption was 5.80 kg/t ore. In the second period, during which ammonia was used, the ore grade had increased to an average of 0.85 wt% Cu and the cyanide consumption also increased to 7.15 kg/t ore. However the cyanide consumption would have been even greater had it not been for the introduction of ammonia into the system, which reduced the average extraction of copper from 41.9% without ammonia, to 21.1% with ammonia. The dissolved copper concentrations in the recycled leach liquor dropped from an average of 1516 mg/L before ammonia to 1278 mg/L with ammonia. If ammonia had not been used and the percentage copper extraction had remained similar to that in the first period, the dissolved copper concentrations would have risen to well above 2000 mg/L and the cyanide consumptions would have risen accordingly.

The third period of operation detailed in Table 2 corresponds to a change in the mineralogy of the ore, where the copper mineralisation became less cyanide soluble (i.e. more chalcopyritic). At this point, ammonia additions in the plant were ceased, but the copper extraction rose only a few percentage points to an average of 24.2% and the cyanide consumption dropped back to an acceptable average of 5.8 kg/t ore.

3.3.3. Discussion of plant results

The average saving in NaCN over the 13 month period when ammonia was being added into the leach tanks was 4.2 kg/t ore treated, whilst the average quantity of ammonia consumed was 5.9 kg/t ore. Based on the delivered costs of these products, the total saving was about US$5/ton of treated ore.

The improved gold recoveries seen in the bottle roll tests when ammonia was added were accompanied by considerable precipitation

of copper due to oxidation and hydrolysis to form cupric hydroxide, i.e. completion of the chain reactions because of sufficient oxidation (Eq. (10)). However, in the plant, it was not possible to achieve this desirable set of conditions because, in order to maintain the selectivity of the ion exchange resin for gold over copper, it is necessary to have a certain amount of free cyanide in the pulp entering the RIP section. In practice, this is achieved by adding PLS from the heap leach operation to the agitation leach pulp just before it enters the RIP tanks. However, any precipitated cupric hydroxide in the slurry would consume free cyanide by reaction and redissolution as copper cyanide complexes. In other words, reaching the highest possible gold recovery in the plant was not economically advantageous because the precipitated copper would consume additional cyanide. So in practice, there was optimum point based on a trade-off between reagent cost and gold recovery.

In actual fact, ammonium-assisted cyanide leaching is best suited for process flowsheets where there is a solid-liquid separation stage between the leaching and adsorption stages, so that any precipitated copper is removed from the system before the free cyanide concentration of the solution is adjusted ahead of adsorption. At Gedabek, because the process involves a resin-in-pulp stage, the advantage gained by adding ammonia to the leach tanks was due, not to increased gold recovery, but instead to the fact that the dissolution of copper from the ore was decreased, thus lessening the overall consumption of cyanide.

4. Conclusions

1. A description of the agitation leach plant at Gedabek is given. The plant is fairly conventional in design, except for the resin-in-pulp section, where gold is recovered from the leach pulp by contact with a gold-specific ion exchange resin, DOWEX XZ-91419, in a series of six tanks equipped with Kemix pumpcell contactors.
2. The integration of the agitation leach plant into the existing heap leach/ADR/SART operation is described.
3. Operational problems caused by increasing amounts of copper in the Gedabek gold ores are outlined.
4. Previous work on the use of ammonia in copper-gold cyanide leach systems is reviewed.
5. Bottle roll test results show that the molar ratio of cyanide consumed versus copper extracted is three, confirming that the copper is in solution largely as the Cu(CN)$_3^{2-}$ complex ion.
6. Further bottle roll experiments show that adding ammonia at molar ratios of ammonia to copper of two or greater results in increases in gold extraction of at least 10%. The addition of a small amount of hydrogen peroxide to the ammoniacal leach solutions generally increase gold extractions by a further few percent.
7. The concentrations of copper in the final bottle roll leach solutions decrease when ammonia is used and generally decrease further still when hydrogen peroxide is used in addition to ammonia.
8. When ammonia was introduced into the industrial agitation leach plant, there was no improvement in the extraction of gold, in contrast to the bottle roll tests, but the extraction of copper was halved, thus leading to significant reductions in the amount of cyanide consumed in the full-scale leaching system.
9. The overall saving in reagent costs when ammonia was used in the leach system was about US$5 per ton of ore treated.

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Glossary

ADR: Adsorption/Desorption/Regeneration plant: where gold is eluted from the loaded resin and the stripped resin is returned to RIP, while the strip solution is electrowon to produce gold doré.

AGL: Agitation Leach plant.

BLS: Barren Leach Solution: recycled process solution used as feed to heap leaching.

HL: Heap Leach plant.

PLS: Pregnant Leach Solution: gold-bearing solution from heap leaching.

RIP: Resin-In-Pulp plant: where gold is extracted from the agitated leach pulp and the PLS.

SART: Sulphidisation/Acidification/Recycling/Thickening plant: where copper and silver are precipitated from the cyanide leach liquor to produce a mixed Cu/Ag sulphide concentrate.

TMF: Tailings Management Facility.