REPORT ON MINING METHODS, EQUIPMENT AND BENEFICIATION PROCESSES:

KLEIN AUB COPPER MINE
OTJIKOTO GOLD MINE (B2GOLD)
ROSH PINAH ZINC MINE
TSUMEB SMELTER (DUNDEE PRECIOUS METALS)

by

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Enhancing decision making for sustainable development

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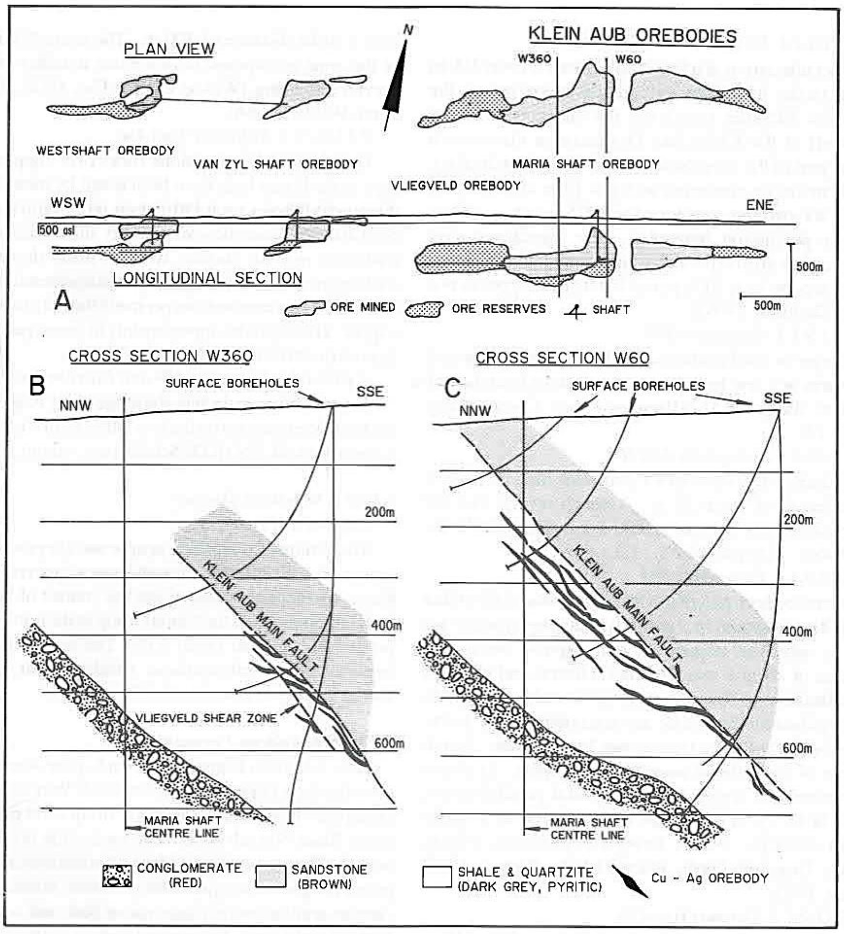
1. Klein Aub Copper Mine

The occurrence of copper on the farm Klein Aub 350 was first recorded by Rimann (1915). In 1927, the mineralised zone was explored by test pits, trenches and a 10 m inclined shaft under the direction of Dr. Hans Merensky, however with disappointing results. In the late 1950s, most of the known strike of the copper- bearing horizon had been pegged. After an intensive diamond drilling campaign in 1959 to 1960, which established a potential of 1 million t of ore, the deposit was investigated by various exploration companies (Erongo Exploration, 1964a; 1964b). In 1966 the property was brought into production by the Klein Aub Copper Company, having proved ore reserves to maintain an initial 450 tons per day mill.

The occurrence of copper over a strike length of about 7.5 km is confined to 7 argillite beds in the Kagas Member of the Klein Aub Formation. These units are intercalated in a stratigraphic succession of approximately 100 m, and range in thickness from a few centimetres to a few metres. They maintain a fairly constant distance from one another, and dip southwards at approximately 45^{o}, subparallel to a prominent breccia zone. In depth the units flatten out and are eventually cut off by the breccia (Schneider & Borg, 1988).

Essentially three separate, sigmoidal ore bodies can be distinguished, the West-shaft ore body, the Van Zyl-shaft ore body and the Maria- shaft ore body. The latter links up with the “Vliegveld ore body” which is only known from exploration drilling and is situated between the Van Zyl- and Maria-shaft ore bodies. The only orebodies mined were the Van Zyl lode in the west and the Maria ore body in the east. The grade of the ore varies considerably throughout, the higher copper concentration being found in the Van Zyl lode (Borg, 1987).



The Klein Aub ore bodies (after Borg, 1987)

The mineralogy of the copper ore at the Klein Aub Mine is quite complex. Chalcocite is the most abundant copper sulphide and accounts for probably more than 85% of the total copper sulphides. It is accompanied by djurleite, digenite, bornite, chalcopyrite, covellite, cuprite, native copper, malachite, wittichenite (klaprotholite), native silver, pyrite, galena, hematite and magnetite. In the oxidised zone of the ore body, which stretches from surface to a depth of about 20 m, malachite and chrysocolla are predominant. Up to 50 g/t silver as well as some gold are mainly associated with chalcocite.

The minerals are disseminated within coarser laminae of fine sand and silt within shale, or occur as nodular and lenticular aggregates in quartzite, as cement to detrital grains, as replacement of early pyrite, as cleavage parallel lenticles, as fillings in brittle fractures and as concentrations along slickensided shale layers.

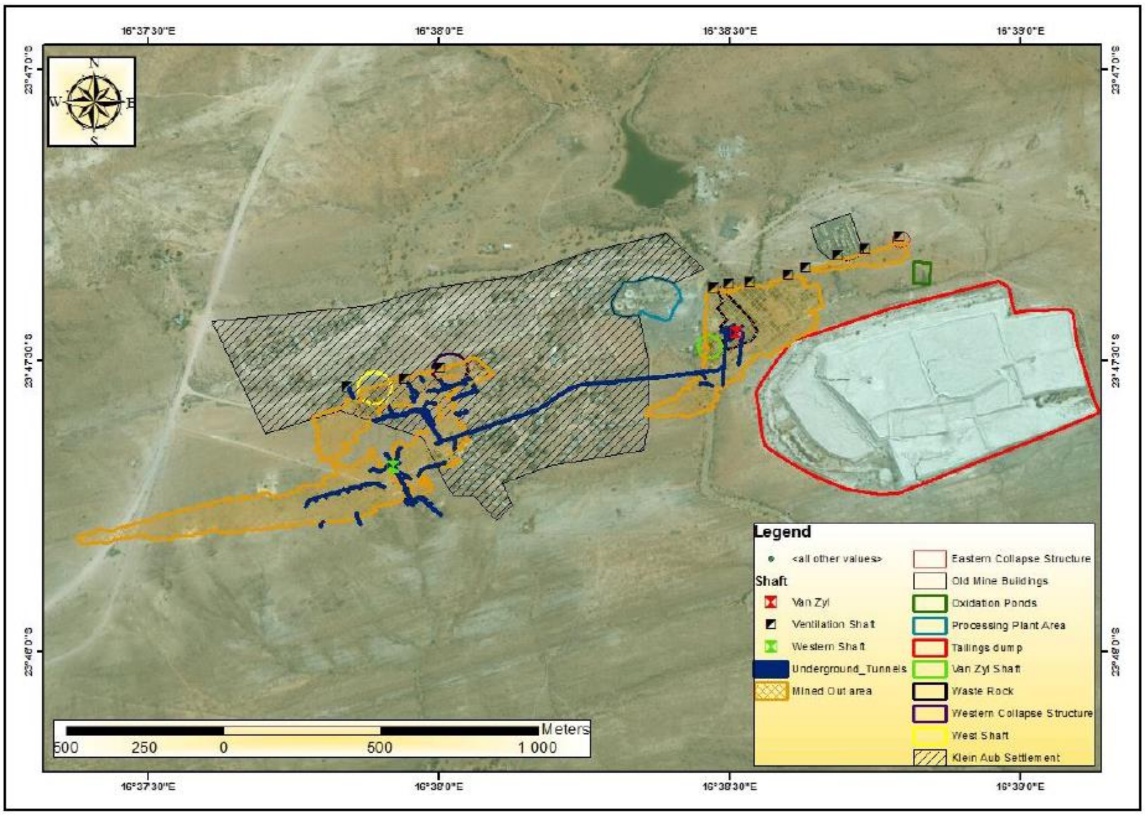
Essentially two different styles of mineralisation can be distinguished. Disseminated mineralisation accounts for approximately 55% of the total mineralisation. Some 45% of the mineralisation is hosted by fractures or other tectonic features such as breccia zones and cleavage planes. Minor ore mineral zonation is developed within the Klein Aub ore bodies, mainly up-dip, and with increasing distance from the Klein Aub Fault, chalcocite mineralisation gradually develops into a narrow bornite and chalcopyrite zone before grading into unmineralised pyrite- bearing sediments (Schneider & Borg, 1988).

Locally within the ore beds there may be a maze of quartz-ankerite stringers in highly sheared and tightly folded argillite. The distribution of the mineralisation is clearly structurally controlled within the stratabound framework. Regional folding coupled with weak metamorphism caused partial recrystallisation and internal adjustments of the copper-bearing sediments. The remobilised elements concentrated within the newly formed structures, although part of the original mineralisation remained unaffected in its original disseminated form.

Borg (1987) developed an ore genesis model for the Klein Aub deposit. It is suggested that the ore formed mainly during an epigenetic multi-phase event. During Doornpoort Formation times the deposition of coarse red alluvial fan sediments by braided river systems forming a thick sequence of oxidised red beds was accompanied by the extrusion of basaltic lava flows. The Klein Aub Formation represents a major environmental change. Due to a marine transgression, fine-grained sediments were deposited in a shallow marine or lacustrine environment. Minor precipitation of copper sulphides in a reducing environment probably took place. Sediment compaction occurred due to continuing burial under accumulating clastic sediments, followed by basin dewatering. The fluids migrated upwards, therefore leaching copper from all available source rocks, especially from permeable zones of basaltic flows. During the peak of the Damaran age metamorphism fluids altered portions of the Doornpoort Formation basalts and leached major amounts of copper and some silver from them. The by now strongly metal-enriched fluids then migrated upwards from the volcanic units and percolated through the overlying sediments. A major fault system at Klein Aub focussed the fluid flow upwards along faults. The mineralising fluids then precipitated sulphides in the reducing environment of the Klein Aub Formation, upgrading the earlier, diagenetic mineralisation considerably.

Ruxton (1986) suggested a period of copper release and concentration of a basement copper source during semi-arid to arid weathering. Subsequent uplift and erosion led to the transport of copper initially as copper sulphate and later as fine malachite particles in the suspended sediment load of alluvial fan distributaries. The deposition of detrital malachite at lake margins adjacent to zones of major sediment and water discharge is envisaged, with further concentration by lacustrine currents. Bacterial action and the breakdown of organic matter during diagenesis led to the formation of sulphides and subsequent copper fixation.

The Van Zyl and Maria ore lodes were each served by a vertical production shaft and a subinclined auxiliary shaft. Horizontal development per level extends over 2500 m in the Maria area and for some 1900 m in the Van Zyl area. By the end of 1980 down-dip mining had advanced to approximately 350 m below the surface. The Van Zyl lode was mined by a panel method (80% extraction) followed by pillar extraction at a daily production rate of 160 t. The Maria ore body was mined by a wide raise method (+ 85% extraction) and pillar recovery which accounted for a further + 8% extraction. The production from the Maria area approximated 900 t of ore per day (Malone, 1985).



Klein Aub tunnels and mined out area (after Moses et al. 2015)

The ore was processed in a plant applying crushing, milling and floatation. The resulting chalcocite concentrate was with up to 56% copper very lucrative. It was sold to the Tsumeb smelter, where it attracted a bonus payment because of the up to 0.2 g/t gold contained in the concentrate. There were also up to 155 ppb PGMs in the concentrate, however, the majority of the PGMs reported to the tailings, and it is estimated that the tailings dam material at Klein Aub contains around 1 ppm/t PGMs.

The Klein Aub Copper deposit contained 7.5 million t of ore averaging 2% copper and 50 ppm silver. Of these, 5.5 million t were mined in the period 1966 to 1987. Due to low copper prizes and the comparatively low grade the mine became subeconomic in 1987. At the time of mine closure the ore reserves amounted to 2 million t (Schneider & Borg, 1988).

Production of copper concentrates (45- 56% copper, 700-1100 g/t silver, 0.2 g/t gold) of the Klein Aub Mine (Schneider & Seeger, 1992)

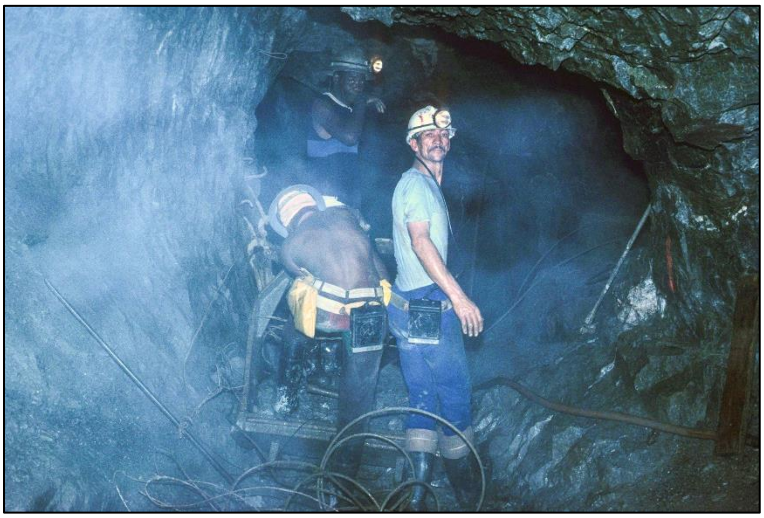
Year	Ore mined (t)	Cu concentrate (t)
1966	41 138	460
1967	186 519	7 428
1968	250 787	9 983
1969	258 315	10 965
1970	225 037	8 926
1971	222 353	9 590
1972	179 193	7 617
1973	163 769	6 460
1974	234 282	9 575
1975	253 839	8 674
1976	289 334	12 441
1977	279 650	12 025
1978	310 329	13 389
1979	318 400	13 576
1980	321 549	16 405
1981	306 200	14 155
1982	294 600	11 464
1983	259 400	10 043
1984	249 357	9 917
1985	214 120	10 064
1986	230 000	7 501
1987	38 347	1 265

Following are some photographs from the time when the mine was in operation.

Klein Aub overview from tailings dam ca. 1984 (Photo: G Borg)

Van Zyl Shaft and plant ca. 1984 (Photo: G Borg)

Maria Shaft and tailing dam ca. 1984 (Photo: G Borg)



Klein Aub underground workings ca. 1984 (Photo: G Borg)

2. Otjikoto Gold Mine (B2Gold)

The first regional gold-focused exploration activity commenced in 1995. Work included geological reconnaissance, airborne magnetic surveys, ground magnetic, electromagnetic and induced polarization surveys, rotary air-blast (RAB), reverse circulation (RC) and core drilling, and Mineral Resource estimation. Since acquisition in 2011, B2Gold has completed additional drilling, a feasibility study in 2012, and has updated Mineral Resource and Mineral Reserve estimates. Commercial production started in 2015. Mining is currently conducted from the Otjikoto and Wolfshag pits.

The mining operations use conventional open pit mining methods and equipment. Mining is based on a phased approach with stockpiling to bring high-grade forward and provide operational flexibility. Ten geotechnical domains have been defined in two oxidation domains (calcrete and oxidized; fresh rock), and pit slope angles vary by geotechnical domain. Inter-berm angles range from 30–60º. Beginning at the 1465 RL, 15 m wide geotechnical berms are included in the design on 60 m intervals. These criteria were the basis for the OSA applied in the optimization analysis.

Groundwater is actively extracted ahead of mining from a single dewatering borehole situated between the Otjikoto and Wolfshag pits. Excess water that accumulates in the pit due to groundwater seepage and rainwater accumulation is collected in sumps located in low spots in each pit and pumped to the return water dam. A phased development strategy was applied in the LOM to smooth the mine production schedule by deferring waste stripping, and to bring high-grade material forward. Tabulations were developed for each of the phases based on the Mineral Reserve gold cut-off grade of 0.45 g/t. The mineralization was then subdivided based on the stockpiling strategy of delaying the processing of most of the low-grade material until the end of the mine life. Development is based on the Otjikoto and Wolfshag deposits each being mined in four phases for a total of eight phases.

Phase 1 has already been completed for both deposits. Wolfshag Phase 4 represents the final expansion to the ultimate pit. The current LOM plan assumes processing of approximately 5.5 Mt from the Mineral Resource low-grade stockpile when higher grade feed is not available. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The stockpile has an average grade of 0.43 g/t Au, which is similar to the break-even processing cut-off grade, so processing of this stockpile will be determined when processing capacity is available.

Where possible, ramps were located in the east wall of both pits to mitigate potential geotechnical hazards associated with planar and wedge-forming structures present in footwall structures along the west wall. A nominal ramp and road width of 27 m, including drainage and safety berms, was used for dual lane truck operation. Ramp widths were reduced to 20 m in the lower levels of the phase designs to allow for single lane haulage on the final benches. Ramp grades were designed to a maximum of 10%. Temporary ramps will be used, as needed, for initial access to stages.

Ore is hauled by truck from the pit to a stockpile, the ROM pad or direct-tipped into the crusher. The highest-grade material is direct-tipped or placed on the ROM pad depending on the plant feed requirements. Production drilling and blasting is done on10 m benches with patterns and powder factors varying by material type and geological conditions. Mining operations are scheduled for 365 days per year with a 15% decrease in the production rate during the rainy season, December through March. Vertical advance is limited to two operating benches per pit phase. This will involve the movement of an average of 38.5 Mt/a of material to sustain processing of 3.5 Mt/a. Mine and mill production are scheduled for eight years with the mining rate dropping the last two years with material from the low-grade stockpiles supplementing the process feed. Haul truck numbers will increase from the current 23 to 26 as the haul distances increase due to the deepening of the pits and distances from the pits to the WRSF.

Two Caterpillar 6018 shovels are generally operating on 10 m benches to remove waste. Ore is selectively mined with Liebherr 984 excavators using three flitches of 3.33 m each. These excavators are also used to mine difficult areas. The two R9250 excavators alternate between ore and waste, as needed. Caterpillar 992 and 990 wheel loaders offer flexibility and are used to supplement the mine production and for stockpile reclaim.

The metallurgical testwork results and information in the 2012 feasibility study provided the data to finalize the process design criteria and the Otjikoto mill flowsheet. The process recovery uses conventional designs and equipment.

The original design of the Otjikoto mill was based on a gravity/whole ore leach flow sheet with a nominal treatment rate of 2.5 Mt/a and a plant availability of 94%. A 25% design factor was included for sizing the primary crusher, conveyors, ball mill, thickeners, cyanide destruction circuit, reagent systems and mainstream pumps which would facilitate a future expansion. A pebble crusher was installed in the SAG mill circuit and two leach tanks were added to the leach circuit in the second half of 2015 to expand the mill capacity from 2.5 to 3.1 Mt/a.

Gold is recovered by gravity concentration/intensive leaching and by a cyanide leach/CIP process for treatment of gravity tailings. The Otjikoto mill design is robust and able to process the three major ore types (XR1 – oxide, XR2 – pyrite-dominant, XR3 – pyrrhotite-dominant) and now Wolfshag over the range of ore grades mined, and with variable materials handling and metallurgical characteristics. The process flow sheet consists of the following: crushing; grinding; gravity concentration and intensive cyanidation; cyanide leaching of gravity tailings; carbon-inpulp (CIP); cyanide destruction; tailings disposal; acid wash and elution; electrowinning and gold room; carbon regeneration; reagents make-up and distribution; air services and plant water service.

The mill fresh water consumption was 1 Mm^3/a at the mill design throughput of 2.5 Mt/a. Fresh water consumption is now permitted for 2 Mm3/a with the expanded mill throughput. Fresh water is supplied from wells for both potable and process needs. Average overall plant power consumption during steady state mill operation is approximately 25–26 kWh/t of ore processed. Electrical power is generated on site using heavy fuel oil generators, and by a new solar power plant that was commissioned in 2018. Reagents are conventional for gold operations, and reagent consumptions for Wolfshag ore are similar to Otjikoto ore (Garagan et al., 2019).

The Otjikoto Gold Mine Plant (Photo: G Schneider)

3. Rosh Pinah Zinc Mine

The Rosh Pinah mineralization was discovered during regional geological mapping in 1963. Initial drilling revealed the presence of 2.9 million t of ore containing 5.8% zinc, 1.2% lead, and 0.15% copper. Subsequent exploration established additional ore of much higher grade. The deposit has been mined uninterruptedly since 1969 (Wartha & Genis, 1992).

The current mining method used at Rosh Pinah is lughole open stoping (LHOS) without backfill, and mining primary, secondary, and, where appropriate, tertiary stopes, in an underhand (top-down) extraction sequence. Mining without backfill since 1969 has resulted in significant voids within the historical and currently mined areas.

Access to the production areas is via multiple interconnecting declines, which also provide fresh air intake into the mine.

Ore is sourced from five steeply dipping mineralized zones, with an increasing proportion derived from the Western Orefield (WF3), as the Eastern Orefield (EOF), and the Southern and Central Orefield (SF3, SF3, and BME) are depleted.

The mine works at a 24-hour, three-shift cycle with primary blasting taking place at the end of the day shift. Material is loaded by rubber-wheeled LHDs and transported by truck either to an underground ore tip or out of the mine to a waste rock dump. The ore passes through a grizzly, equipped with a hydraulic rock-breaker, on its way to the underground primary crushing station, where it is crushed to <150mm size and transported to the surface via an inclined conveyor drift. Waste rock is trucked out of the mine and dumped on one of two designated rock dumps which are located within the Mining Licence area.

The current Rosh Pinah concentrator utilizes a conventional three-stage crushing and ball milling circuit followed by flotation. In the concentrator, the cyclone product is subjected to two differential flotation processes, namely the lead floatation circuit and the zinc floatation circuit.

In the lead floatation circuit, the lead is initially floated using sodium normal-propyl xanthate, dextrin, sodium cyanide, senfroth 20 and water, with the aid of conventional agitation flotation in the lead rougher and scavenger banks. The sodium normal-propyl xanthate ensures that the mixture is hydrophobic, so that in later steps the lead sticks to the air bubbles. The tails of the lead circuit go through to the zinc circuit. The lead rougher concentrate is floated through a series of cleaner cells consisting of a column cell, 'Tankcell' and 'Smartcell'. The overflow water of the lead thickener is pumped to the tailings dam.

In the zinc floatation circuit, the zinc is floated off in the zinc circuit using senfroth 20 and copper sulphate. The mixture is fed through a battery of conventional agitation cells in the rougher circuit. The zinc is collected in the rougher and cleaned cells. Both the lead and zinc column cells were retrofitted with Microcell spraying systems. Air is pressed into these cells with high pressure which allows for the forming of more and smaller air bubbles, to which the Pb and Zn respectively stick. The tails from the zinc circuit are first thickened using anichem 8861 and the residue pumped to the tailings dam located approximately 2 km to the south-east of the plant. The overflow of the zinc thickener goes back into the zinc circuit.

From the concentrator, the lead and zinc concentrates pass through separate belt filters that dewater the concentrate prior to being stockpiled on drying pads. The drying pads have a concrete floor and are bounded by approximately 1 m-high walls. The water from the belt filters is pumped to the thickener.

In August 2021 Trevali (the owner of RPZC until 2022) announced positive results from the independent Rosh Pinah Expansion "RP2.0" NI 43-101 Feasibility Study (FS) at its 90%-owned Rosh Pinah mine in Namibia. The FS considers the potential for ore production expansion from 0.7 Mtpa to 1.3 Mtpa. The FS is based on a scenario to expand the current throughput through the upgrading of the processing plant, construction of a paste fill plant (including a water treatment plant), surface crushing facilities, and development of a dedicated portal and ramp to the Western Orefield (WF3) deposit.

The Expansion Feasibility Study includes the following changes at the mine:

Processing Plant: The FS incorporates an upgrade to the comminution circuit to include a new single stage SAG mill and pebble crusher. The upgrade also includes primary crushing upgrades, an ore blending system, along with other circuit modifications to provide increased flotation, thickening, filtration and pumping capacity to achieve the target throughput of 1.32 Mtpa. The upgrade will also include several flowsheet modifications aimed at improving both the concentrate grade and metal recoveries.

Underground Development and Infrastructure: A dedicated portal and decline to the WF3 deposit will be constructed to support the increase to mine production levels and reduce operating costs. The trucking decline is 3.9 km in length, excluding level access and stockpiles. For construction purposes, the decline is separated into five independent legs to enable concurrent development and reduce overall construction time. Internal legs of the trucking decline will be developed off existing development, with take-off positions selected to minimize interaction with the underground operation.

The new trucking decline will act as an additional fresh air intake within the ventilation network and will enable direct ore haulage from the WF3 zone to a new surface primary crusher station utilizing large-scale (60 tonne) trucks. Ore sourced from other areas (EOF, SF3, SOF, and BME) will be transported to the existing underground crushing system using the existing 30 tonne truck fleet and conveyed to surface via the existing conveying system.

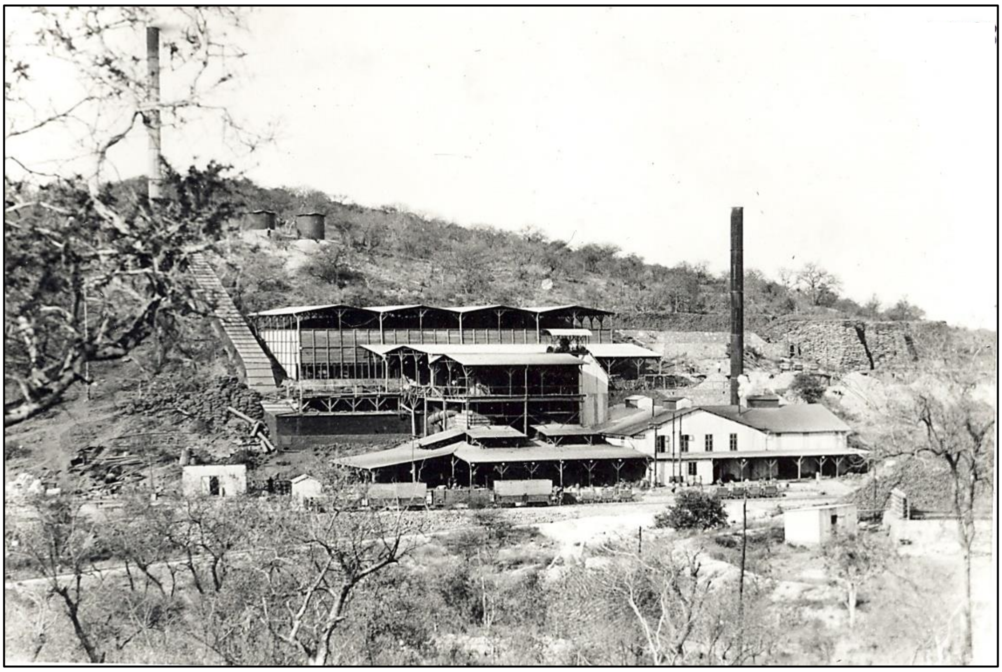
Paste Fill Plant: A paste fill plant designed to operate at both the current 0.7 Mtpa and the 1.3 Mtpa targeted throughput rate has been included. The paste plant construction is assumed to commence in Q1 2022 (approximately 12 months before the expansion project's commencement) as it improves both the safety conditions and economics of the current mine configuration as a standalone project. Paste filling the stopes rather that leaving them void will improve ground stability, increase ore recovery, and reduce dilution. It will also reduce surface tailings as a portion of new and existing tailings will be redirected underground to be used as paste fill. It is also critical to fill existing voids (particularly within WF3) to achieve the increased production target and preferred mining sequence considered as part of the expansion project.

Mobile Equipment: The existing small-scale underground trucks and load-haul-dump (LHD) fleet will continue to be used primarily in the current mining areas. This will reduce capital expenditure associated with purchasing new mobile equipment, increasing development profiles and changing the existing underground crushing and conveying system. As mining extends deeper and average haulage distances increase in WF3, new large-scale trucks and LHDs will be purchased for the more efficient transportation of material to surface reducing costs over the life-of-mine (Speiser, 2021).

Rosh Pinah Mine with the settlement in the foreground (Photo: Chamber of Mines of Namibia)

4. The Tsumeb Smelter (Dundee Precious Metals)

Tsumeb has a long tradition of smelting, as the local people have smelted the copper ores for time immemorial and traded the copper for other goods. The first smelter plant was erected at Tsumeb as early as 1906-07. It consisted of two lead-copper blast furnaces, which were fired with first-class German coke. Only after World War I was coke obtained from South Africa. Third furnace was added in 1923 and a Cotrall precipitator for the recovery of cadmium was installed in 1925. In 1930 a rotary furnace was acquired to roast the cadmium-bearing flue dust. Due to World War II, the smelter came to a standstill in 1940. During the 1950s, the Tsumeb concentrates were smelted and refined at overseas smelters. However, the ever-increasing transportation costs made it necessary to erect a new and larger smelter at Tsumeb in 1960-62 (Schneider & Seeger, 1992).



The smelter of the Otavi Minen- und Eisenbahn-Gesellschaft, Tsumeb, 1909 (Photo: National Archives)

Today's facility, which was bought by Dundee Precious Metals in 2010, is amongst the most modern and efficient complex smelters in the World. It consists of a primary smelting furnace, the Ausmelt furnace, two Peirce Smith Converters, bag houses and cooling towers, a slag milling plant, two high voltage distribution sub-stations, a materials handling facility, two oxygen plants, a fume extraction system and a sulphuric acid plant. The smelter is one of only a few smelters in the world that can treat complex copper concentrates. Blister copper and sulphuric acid are the smelter products. The blister copper is delivered to refineries in Europe and Asia for final processing to copper metal. Sulphuric acid is a critical component in the mining industry, particularly for uranium and copper production businesses, and therefore the smelter provides a by-product of copper for other mining operations in Namibia.

The Ausmelt process is one of the most energy efficient smelting technologies available. In this process the direct injection of fuel into the slag bath and ability to utilise the energy released during the oxidation of the feed reduces the fuel requirements compared to alternative processes. The addition of post combustion air with the Ausmelt Lance Shroud System is used to maximize the recovery of energy available in the system. This energy may be used in other sections of the plant or for electricity generation. This further reduces the plant's energy requirements and minimises green house gas emissions.

The Ausmelt TSL Process can handle a wide range of feed materials including low grade concentrates, complex polymetallic ores, materials with high impurity contents and secondary and industrial waste materials. These materials, which are often problematic for conventional smelting technologies, are becoming increasingly important feed sources.

Ausmelt copper smelting entails dropping moist solid feed into a tall cylindrical furnace while blowing oxygen-enriched air through a vertical lance into the furnace's matte/slag bath. The products of the process are a matte/slag mixture and a strong SO_2 offgas. The matte/slag mixture is tapped periodically into a fuel-fired or electric setting furnace for separation. The settled matte (~60% Cu) is sent to conventional converting. The slag (0.70% Cu) is discarded (www.dundeeprecious.com).

The offgas (25% SO_2) is drawn from the top of the smelting furnace through a vertical flue. It is passed through a waste heat boiler, gas cleaning and on to a sulfuric acid plant. A small amount of oxygen is blown through the side of the smelting furnace or lance (about halfway up) to ensure that sulfur leaves the furnace as SO_2 rather than S_2. This prevents sulfur condensation in the gas cleaning system. Most of the energy for smelting comes from oxidizing the concentrate charge. Additional energy is provided by combusting (i) oil, gas, or coal fines blown through the vertical lance and (ii) coal fines in the solid charge (www.totalmateria.com).

Peirce-Smith converters (PSCs) have been used in the copper smelting industries for more than a century for the purpose of removing iron and sulphur through oxidation reactions to obtain blister copper and converter matte respectively. This process step is referred to as conversion. The conversion process used in removing iron and sulphur from matte is a complex phenomenon involving phase interactions, many chemical reactions, associated heat generation, as well as product formation (Kyllo and Richards, 1998a). The PSC is a cylindrical horizontal reactor (circular canal geometry) where air at subsonic velocity (Mach < 1) is injected into matte through submerged lateral tuyeres along the axis of the converter. The converting process is semi-continuous and autothermal. Since there are chemical reactions taking place with products being formed, quality and quantity of mixing are important (Chibwe et al., 2015).

The acid plant, part of the Tsumeb smelter (www.metso.com)

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