D.1. Operational Information Requirements

Introduction

The Tara mine Navan Zn-Pb deposit was discovered in 1970 by the Tara Exploration & Development Company Ltd. Mine development started in 1973 when a decline portal and development shaft collar was constructed in the shallow overburden.

Mine planning, development and equipment procurement continued into 1974 when all work ceased for 13 months while negotiations with the Irish Government on terms of a State Mining Lease were concluded. Construction of the facility continued into 1977 when the first hoisting of ore occurred and production of concentrate started in June the same year. Ownership of Tara passed to Boliden on 1st January 2004.

The current projected 'life of mine plan' extends past 2018. The mine is currently divided into the following sub-areas for management and planning:

- The Central Area, also known as the Main Mine or Knockumber Mine. This was the first area to be mined. It has largely been mined out.
- The South West Extension (SWEX), to the southwest of the Main Mine.
- Nevinstown, to the north of the Main Mine.

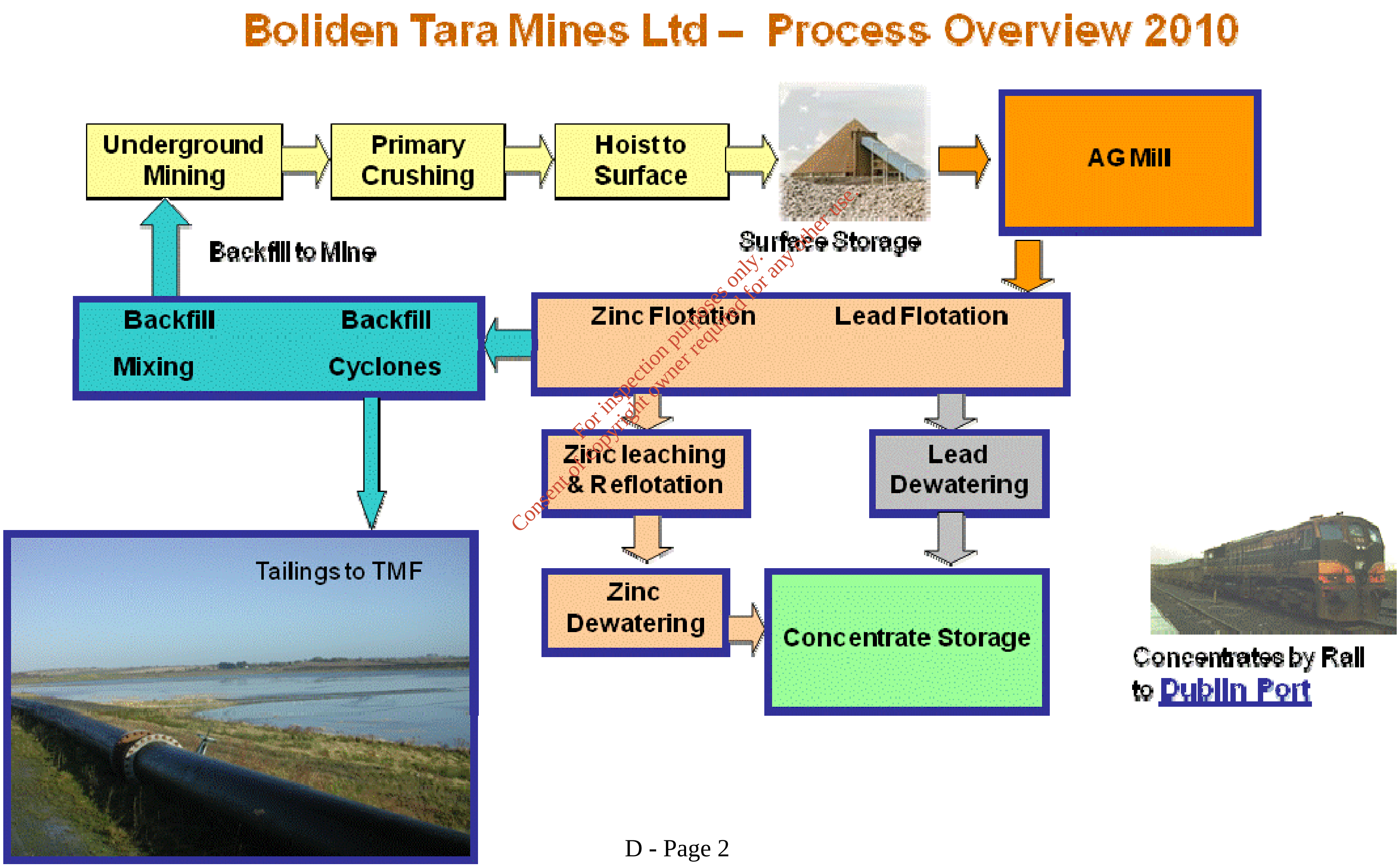
SWEX and Nevinstown are currently being developed and mined by trackless underground methods, principally long-hole open stoping with backfill. Room and pillar methods are applied where the ore is thinner. Access to the SWEX and Nevinstown is from a portal on the Main Mine site.

An expansion of the mine workings is currently being planned into Liscartan and Rathaldron which is currently subject to a planning application to Meath County Council. This section of the orebody is located to the northwest of the Main Mine area. The expansion of the mine workings into new areas in Liscartan & Rathaldron will be from the existing underground workings and will use similar mining methods to those currently being employed.

The original ore reserves (calculated in 1971) in the entire orebody amounted to 69.9 million tonnes grading 10.09% Zn, 2.63% Pb. The estimated current (end December, 2009 known Joint Ore Reserves Committee (JORC)) ***classified mineral resources*** including the Liscartan/Rathaldron orebody are 11.8 million tonnes grading 7.1% Zn and 1.8% Pb while the current ***estimated ore reserves*** are 17.0 million tonnes at 7.2% Zn and 1.8% Pb.

A complete description and assessment of the Nevinstown, SWEX and Liscartan/Rathaldron mine extensions is presented in the EISs for the projects see Attachment B.5 Part I, Part II and Part Iv.

Figure 4 Mining and Processing Flow Diagram



Mining Operations

Mine development

Development takes place in two phases: main development and exploration, in which sections of the orebody are outlined and made accessible for eventual mining and stope development. This involves the drilling, blasting, and recovery of the stope and pillar ore. Both forms of development are important sources of ore, contributing up to 400,000 tonnes annually to the total production target.

Mine development is planned to allow access to the orebody at varying levels and to prepare sections of the orebody for mining. Development drifts are driven at different dimensions depending on their intended purpose, for example a main haulage drift will have larger dimensions than a drift designed for ventilation purposes. It is necessary to have a significant amount of development in place before large-scale ore production commences. All development work is accessed from the existing mine and all ore resulting from development will be transported to the underground crushers / conveyor systems.

Mining Methods

The predominant mining method is 'longhole open stoping' where ore thickness are sufficiently high (up to 12m), together with variations on room and pillar mining in areas of thinner ore (down to 4m). Cemented backfill is used for initial primary stoping and a mixture of cemented and un-cemented backfill for later (secondary) stoping. Longhole open stoping is sequential, that is, stopes are mined in sequence and each stope is filled before mining of adjacent stopes can commence. In room and pillar mining the roof stability is achieved through the design of stable rock pillars, provision of roof support where necessary and the use of regional (larger) pillars where the mining spans are large.

Hanging-wall drifts are driven some 40-50m above the orebody to facilitate exploratory diamond drilling, ventilation and backfill functions.

Mine Production

Mine production is the generation of large tonnages of ore from stopes and pillars. All ore produced is transported by trucks and scoops to the primary crushers located underground.

Generally, stopes and pillars are laid out with their long axes parallel to strike. This has reduced the amount of footwall development in waste and facilitated stope access through haulage pillars that are essentially aligned down dip.

Stope and pillar dimensions have evolved over time from an initial 12.5m width for both, to the present less rigid dimensions. Widths for both are now determined on a case by case basis and are controlled by local features such as ore thickness, bedding planes, faults, joints, and adjacent openings. Heights are also variable and also depend on the thickness of the ore ranging from 10m to 80m. Average unit sizes are in the region of 25,000–30,000 tonnes per stope/pillar, giving rise to a total requirement of up to 80 stoping units per year in order to maintain the production figure of 2,200,000 tonnes from stoping and pillar mining.

Blasting is carried out using emulsion based explosives where the broken ore is removed from the stope and taken to one of the crushing stations either by loader or truck depending on the distance of travel. The blasting/mucking sequence continues until the stope is completely mined out, after which it is subsequently filled with tailings sands, cement and/or development waste, the proportions of which are determined by location and adjacent mining plans.

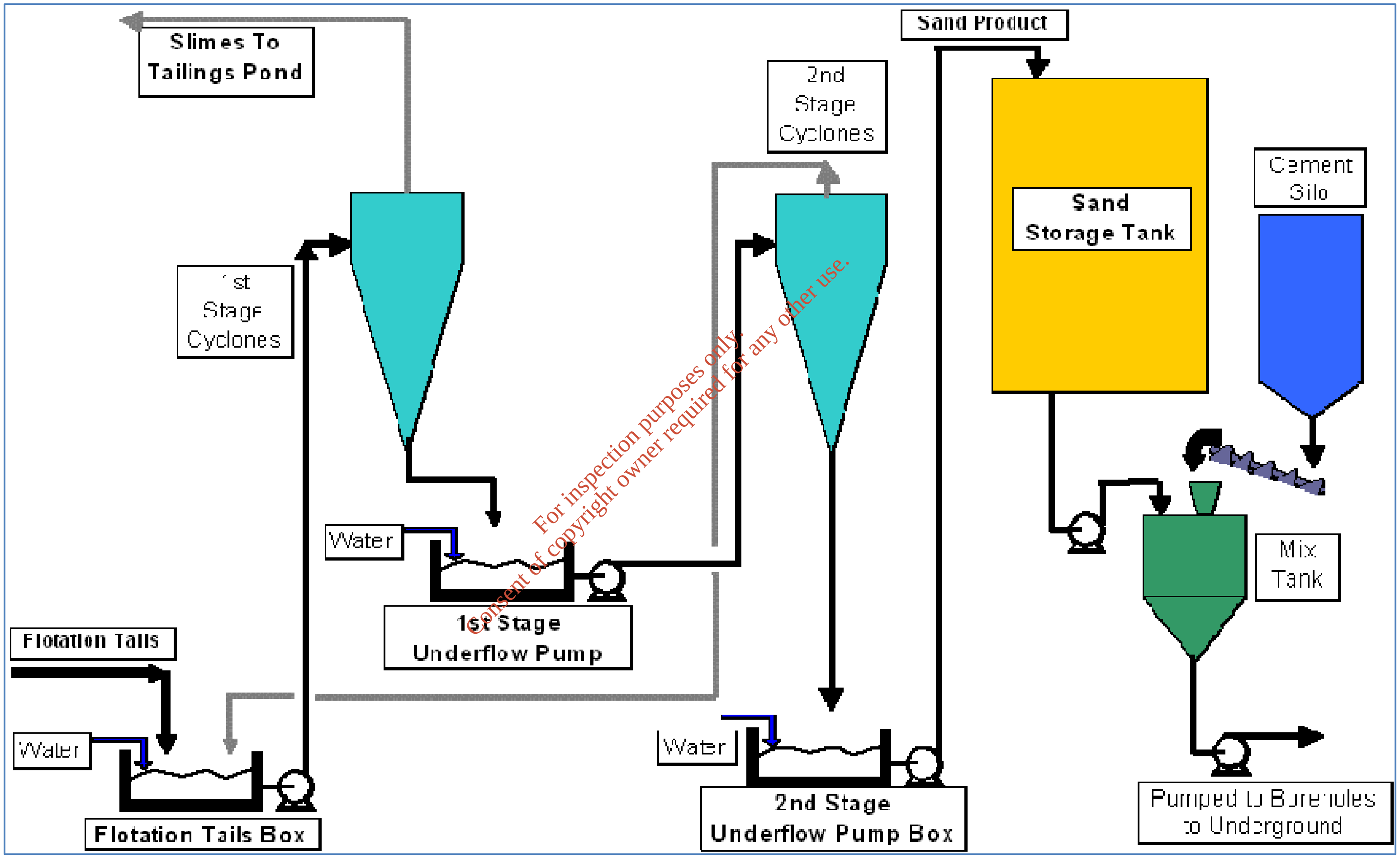
Underground primary crushing results in the reduction of the ore size to less than 150mm, at rates of up to 800 tonnes per hour. Crushed ore is carried by conveyor to a storage bin of 3600t capacity located adjacent to the production shaft, where it is fed to the shaft skip loading pockets. The ore hoisting cycle is automatic, the control of the ore feeders, transfer conveyors and skip loading being regulated by the hoisting cycle and the weigh cells at each loading pocket. Ore is hoisted into two 15.5t capacity bottom dump skips running in balance, tipped into a small bin at the head-frame and then conveyed to a 35,000t surface storage building.

Backfilling

The voids left by the mining of stopes and pillars are filled with cycloned sand, which is produced in the concentrator from tailings and directed to the stopes through surface pipelines and boreholes that connect with the haulage levels. This operation is an essential part of the mining cycle, and close scheduling is required to meet production targets. The recovery of approximately 52% of the mill tailings for backfill, provides sufficient material to replace the ore mined from the stopes and pillars. Maximum fill recovery and utilisation also ensure efficient use of the available volume of the surface tailings management facility.

To facilitate the mining of pillars between the sand-filled stopes, the sand is given cohesion, by the addition of cement at dosage rates of between 3% and 9%, depending on planned exposures. Pillar voids are usually filled with un-cemented sand and/or waste rock. A blended cement consisting of 90%-95% ground granulated blast-furnace slag and 5%-10% normal Portland cement is used. Each stope filling operation is carefully planned to optimise future ore recovery from the pillars, and the economics of designed cement additions are continually analysed.

Figure 5 **Backfilling Flow Diagram**



Waste rock backfilling

The annual tonnage of waste rock mined in pursuit of ore is c.300,000 tonnes, most of which is recovered from remote parts of the mine in locations of future mining. Some waste rock is placed in stopes while some is brought to surface. The surplus waste stockpile has been classified as a reusable product by the EPA; subject to specific conditions (EPA Ref. P0516-01/ap12dh.doc). Much of the surplus rock is now used in the preparation of concrete and shotcrete, both used for backfilling and mine stabilisation purposes.

Concrete and shotcrete

There are two separated concrete plants, also known as batching plants. Both plants combine various ingredients to form concrete products. Some of these inputs include sand, water, aggregate (rocks, gravel, etc.) and cement. There are no air emissions from the plants as all dust is collected in filters and returned to the mixer.

One plant is utilised for the production of large concrete blocks used for underground construction while the other plant produces shotcrete. The large concrete blocks are used to close off access to the stopes prior to filling with backfill.

Shotcrete is a concrete based mix which is sprayed through a hose at high velocity on to a rock surface for support. It can be impacted onto any type or shape of surface, including vertical or overhead areas. The quantity of shotcrete used has increased in recent years as the mine progress deeper into the south western sector of the orebody.

The positive impacts of the onsite production of concrete products include the reuse of surplus mine rock and the reduction in road traffic as previously all concrete had to be imported from offsite sources.

Services and Mobile Equipment

The efficient operation of the mine is critically dependent on a variety of mobile equipment designed specifically for the underground mining. Most of the scoops operate by remote control and are all equipped with noise insulated cabins for operator safety and comfort. The mobile fleet is powered by diesel engines.

The production and the development drilling fleet is all electro-hydraulically driven, incorporating the most up to date features, including the facility to programme the machines to drill while unattended. To assist the efficient running of the operation many service backup vehicles are used for shotcreting, loading explosives, pipe handling, bulkhead building, materials handling and other tasks. Mobile equipment includes 4x4 wheel drive vehicles, some of which are fitted with scissor lifts for general maintenance, while others are used for personnel transport.

Ventilation

The large lateral extent of the orebody, together with the variations in its thickness and depth, calls for a ventilation system that can be adapted to demand as the mining locations change. There are two primary reasons for mine ventilation:

- to provide oxygen rich air for underground operators and machines,
- to remove and dilute concentrations of noxious gases so as to render them harmless.

The extensive use of underground diesel equipment requires large volumes of fresh air to be passed through the mining access routes. Ventilation is by a 'pull' system whereby the fans are on the exhaust end of the system creating a negative flow.

The noxious gases to be removed are the emissions from the underground diesel equipment; these are carbon monoxide, carbon dioxide and nitrous oxides. In addition the gases generated as by-products of blasting are also removed and these are carbon monoxide, carbon dioxide, sulphur dioxide, ammonia and hydrogen sulphide. However blasting gases are all removed in the short period following blasting.

Exhaust gases are removed from the mine via Return Air Ventilation Raises (RARs). There are a total of six return air raises currently in operation and a seventh is planned. Since the original IPC licence was granted in 2001 two additional RARs have been installed (RAR No. 4 and RAR No. 5). Figures 3 gives a elevation of RAR No.5 and Figure 4 is a plan of the facility.

RAR No. 5 and the proposed RAR No. 5-2 are similar installations.

Installation of ventilation raise RAR No. 4 (IPPC Licence REF NO. A2-9)

This facility was commissioned in 2003 and involved the conversion of a Fresh Air intake raise to a Return Air Raise to remove air and fumes from the underground mining activity.

The installation consists of 2 fans that operate in parallel with each other and have a fixed speed of 1500RPM. The fans are located 80 m below ground and vent to the atmosphere via a 3.6 m diameter raise. The emissions to surface from the raise are re-directed towards the ground through a deflection hood measuring 6.7m x 3m. The 24 hour daily emissions limit is 1.86×10^7 Nm^3.

Installation of ventilation raise RAR No. 5 (IPPC Licence REF NO. A2-10)

This facility commissioned in December 2006, located approximately 5 km south of the mine site, removes air and fumes from the SWEX area of the mining operation.

The installation consists of 4 fans that operate in parallel with each other and have a variable speed range of 0 - 1500RPM. At a maximum operating speed of 1500RPM, airflow of 370m3/s is generated. The fans are located in excess of 650m below ground and vent to the atmosphere via a 4.5m diameter raise.

Emissions to surface from the raise are re-directed towards the ground through a deflection hood measuring 5.7m x 5.7m. The fans operate in an efficient manner to provide sufficient ventilation with minimum energy consumption. RAR No 5 is inaudible at 200m from the surface discharge point. The design maximum 24 hour daily emissions equates to 3.2×10^7 Nm^3. Air intake is through various openings including the main decline, the Nevinstown decline, the production shaft, the development shaft and 4 No. fresh air raises (FARs).

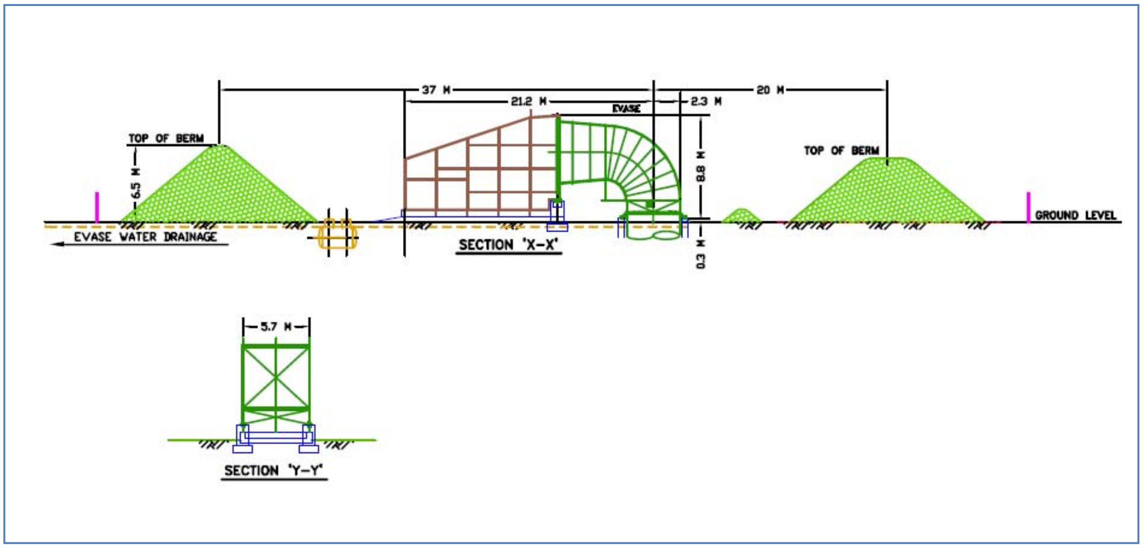


Figure 6   Return Air Raise No. 5. Side elevation

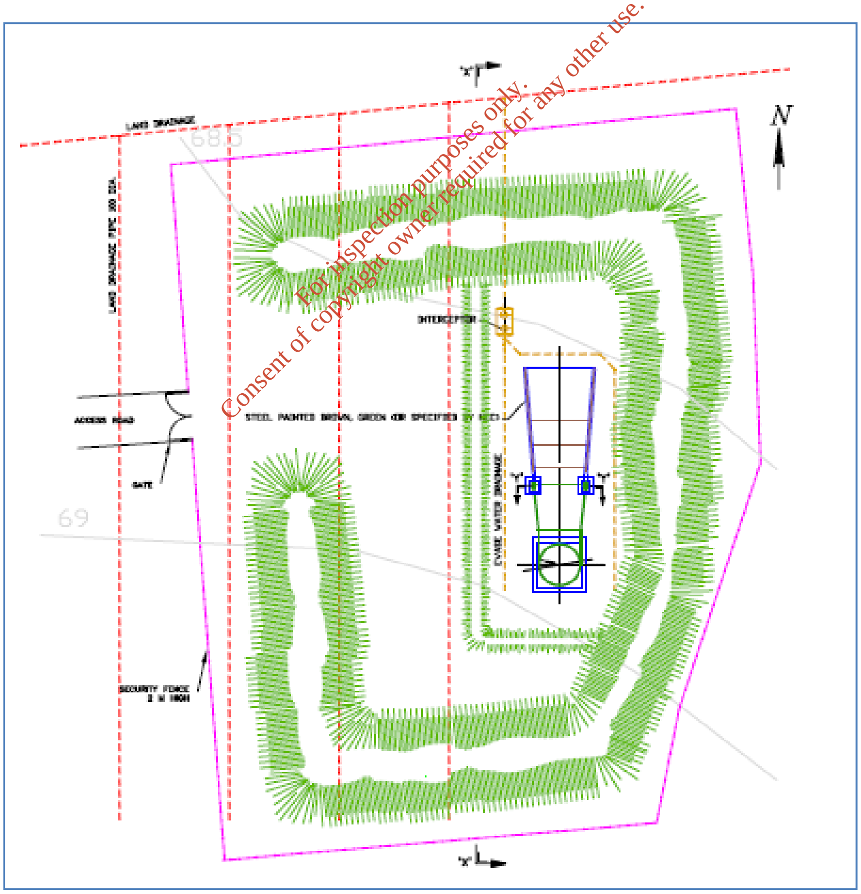


Figure 7   Return Air Raise No. 5. Plan.

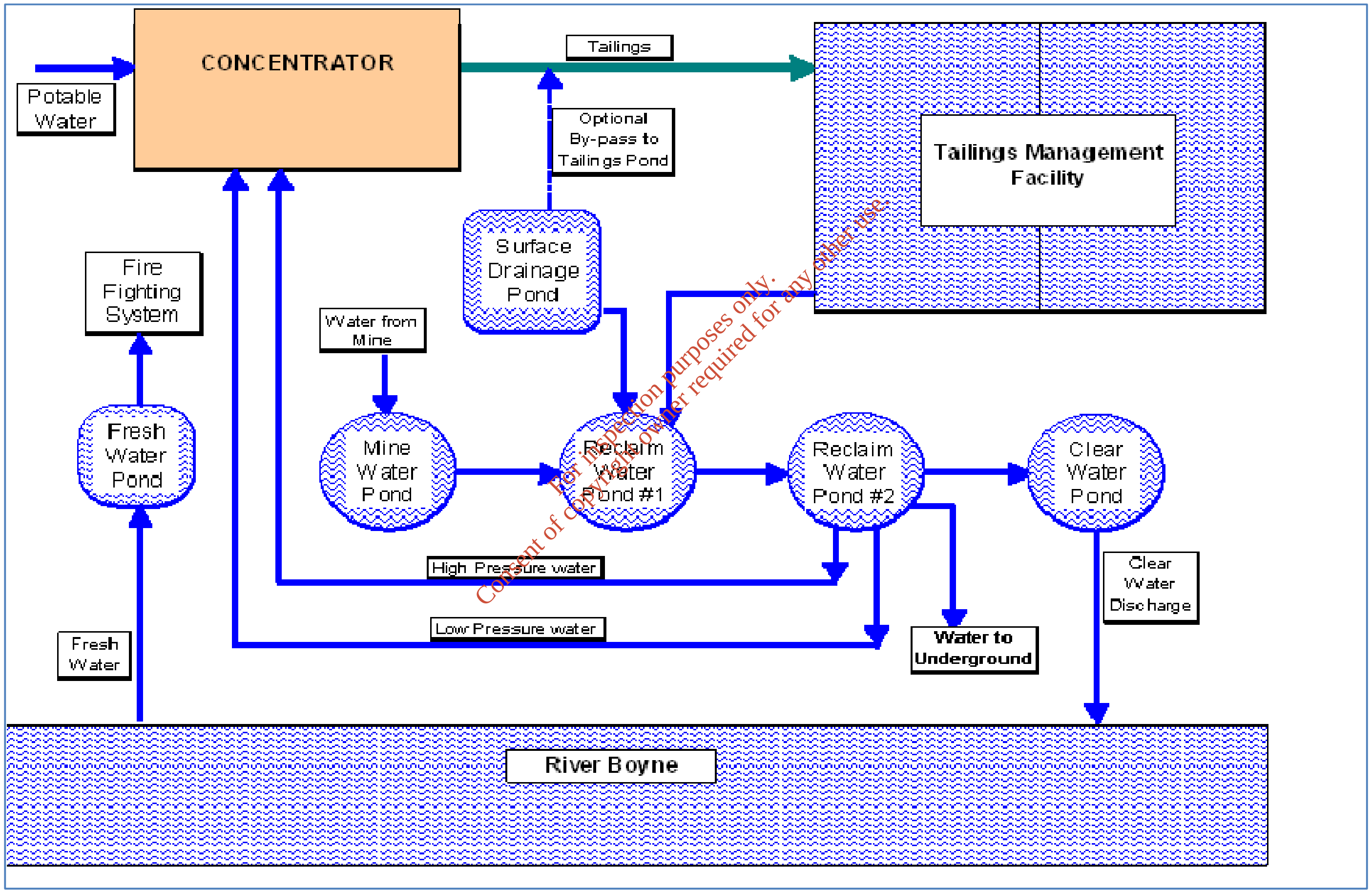
Mine Drainage

Water enters the mine in three ways:

- as natural ground water,
- as service water for the mining operations,
- as transport for the backfill.

The lowest pump station is fully automated, where variable speed slurry pumps operate to remove the dirty water. This water is pumped directly to the main pump station located in the shaft pillar where it is screened, clarified and pumped to the surface. The water entering the clarification system is screened to remove the >3mm material, and clarified with the use of a lamella thickener. An organic polymer is added to induce rapid settlement. A rake at the bottom directs the mud to the central discharge point where a mud pump lifts it to the mill tailings pump box on surface. Clean water overflow collects in two holding sumps, where clean water pumps then lift it 416m to the surface pond where it can be reused. The total pumping capacity of the mine is 21,600 m^3/d while the current total inflow to the mine is 11,979 m^3 / day.

Figure 8 Water system Flow Diagram



Ore Processing

The ore is comprised of zinc and lead sulphide minerals, other minerals and limestone.

Typical composition of ore.

Ore Constituents	Percentage (%)
Lead	1.5 – 3.0
Zinc	7.0 – 9.0
Iron	2.0 – 5.0
Magnesium Oxide	6.4
Barium Oxide	4.4
Copper	0.004
Calcium Oxide	24.0
Silver	17 gm/dmt

The mineral particles must be physically liberated from the host rock in order to selectively recover the lead and zinc metal during the froth flotation stages. The minerals are tightly bound together within the ore and the separation is achieved in a number of processing stages. The process of reducing the ore particle size is known as Comminution.

Grinding

The first stage is the crushing of the ore to minus 150mm in jaw crushers underground. After being hoisted to surface, this material passes to the coarse ore storage building from which it travels to the new *Autogenous grinding circuit*. The ore then passes to the autogenous grinding circuit and is mixed with water. The Autogenous grinding circuit reduces the ore particle size to less than 120 microns, a size range where the mineral particles and the host rock can be separated. Autogenous Mills use large particles of ore instead of steel balls for grinding media. The finely ground ore slurry is then pumped to the floatation stage of the process where the lead and zinc minerals are recovered respectively.

Since processing operations began at Tara in 1977 ore comminution (size reduction) was achieved using crushers and steel grinding mills. The crushing circuit consisted of three hydracone crushers, eleven conveyors to transfer ore, a dust extraction system and ancillary items while the grinding plant consisted of a rod mill, two ball mills plus six conveyors, to transfer ore, and two banks of hydro-cyclones. This system generated fine dust which requires a dust collection system (wet scrubbing plant and air exhaust to atmosphere Emission Point Ref No. A2-2). The new AG milling circuit has replaced all of these crushing and grinding systems, and is designed to deliver a similar product size distribution to flotation. The energy usage is similar to the usage in the crushing and grinding circuit. Autogenous mills operate, mechanically, similar to the ball mill but differ in the media they use to break and grind the ore. Autogenous Mills use large particles of ore instead of steel balls for grinding media. This single mill, which rotates in an enclosed circuit, has a maximum operating capacity of 400 tonnes/ per hour and does not require an emission to atmosphere.

The autogenous grinding circuit is designed to reduce the ore particle size to less than 120 microns. At this size range, the mineral particles are liberated from other minerals and the host rock.

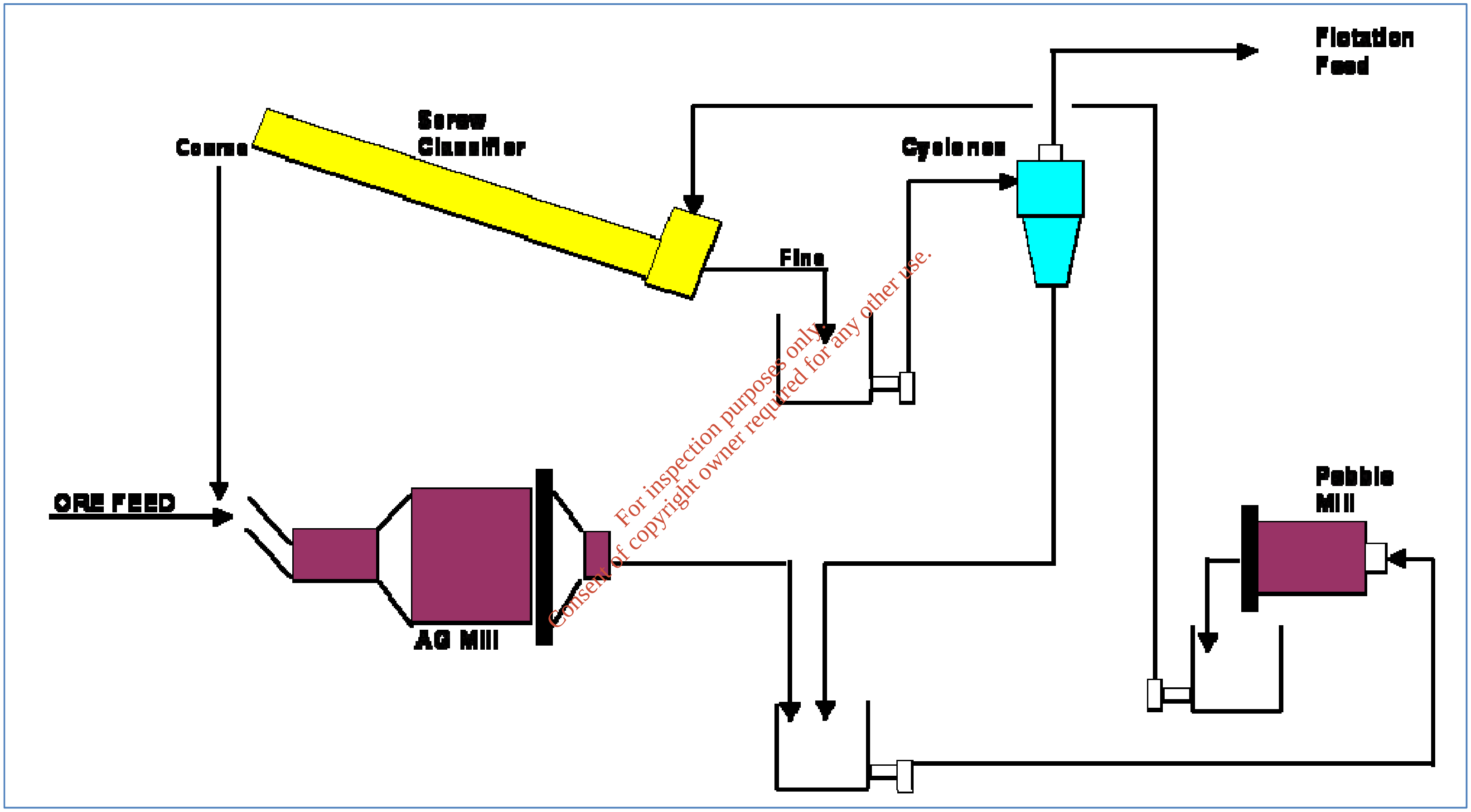
The new AG system removes the need for much of the historical ancillary equipment; the seventeen conveyors in the previous circuit is reduced to just two in the new circuit (The old system was a combination of a wet and dry process and dust generated at the dry crushing process had to be ventilated. The dust from the historical system was extracted and passed through a wet scrubber before being emitted to the atmosphere through the ventilation stack). The new AG circuit is a completely wet system thereby eliminating the source of dust and the point of emission to air. The result is the elimination of one significant atmospheric emission point.

After grinding the finely ground ore slurry is then pumped to the flotation stage of the process where the lead and zinc minerals are separated from the host rock. The finely ground ore slurry is pumped from the grinding circuit to the flotation stage of the process.

Positive Impacts:

- Elimination of dust generation. Improved work environment.
    - **Elimination of IPPC L Emission Point Ref. No. A2-2**
- The main noise sources from the existing grinding system have been decommissioned (3 Cone Crushers, Primary Ball Mill & Rod Mill).
- Reduce consumables used in crushing and milling processes: Steel rods and balls used in the mills by (25,000kg annually), conveyor belt
- Reduction in associated maintenances involved in the various components of the old current crushing and grinding systems.

Figure 9 Grinding Flow Diagram



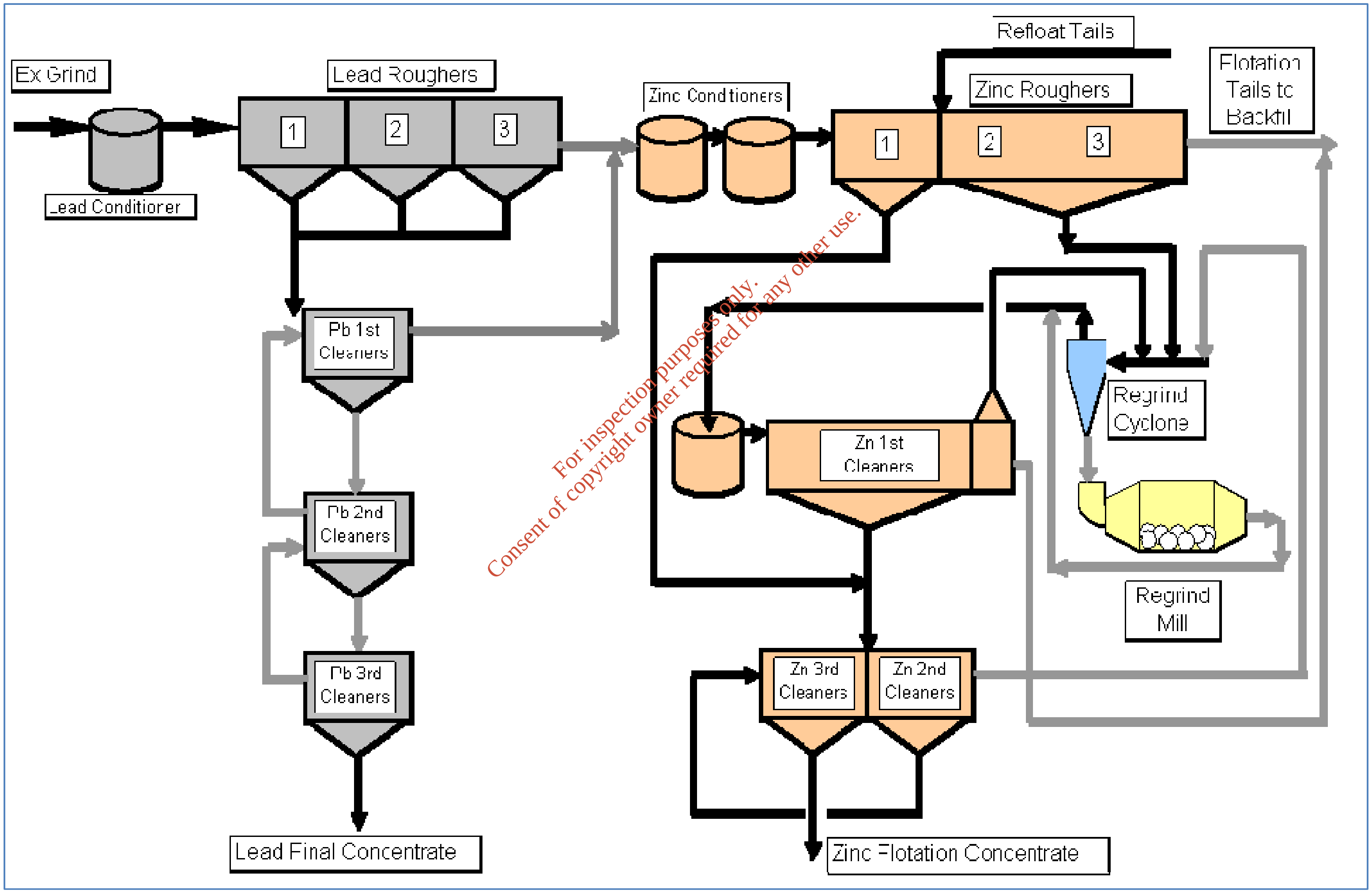
Floatation

The flotation process is divided into two distinct sequential stages. Lead minerals are recovered in the first stage followed by recovery of zinc minerals in the second stage two.

The flotation circuit consists of a series of tank-like cells in which there is a rotating agitator, which stirs the mixture of ground ore and water. Chemical agents are added, one of which promotes frothing. Other agents alter the surface properties of the mineral particles and cause them to be attracted to bubbles, generated by forcing air through the mixture by the agitators. These bubbles, coated with mineral particles, rise to the top of the mixture as a froth, and overflow the lips of the cells into collection troughs or launders.

Different combinations of reagents are used to selectively remove different minerals. Lead minerals are removed in the lead circuit and the remaining material is transferred to the zinc circuit for the removal of the zinc minerals.

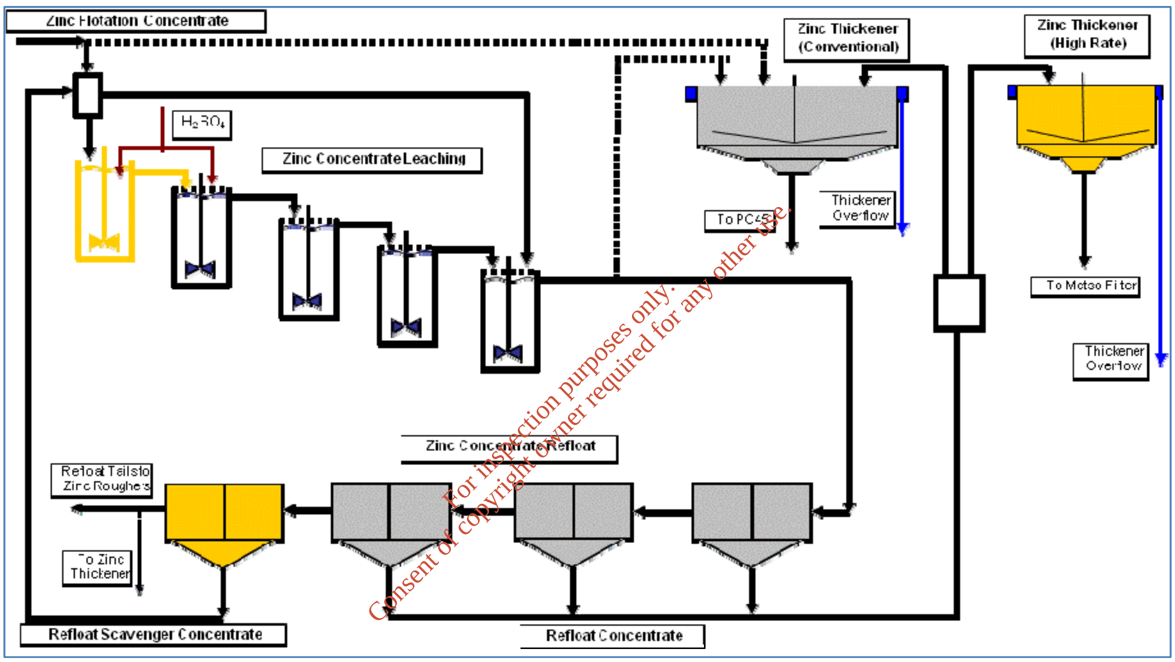
Figure 10 Floatation Flow Diagram



Leaching

To reduce the dolomite content in the zinc concentrates, the bulk of the zinc flotation concentrates are leached with sulphuric acid in enclosed, agitated vessels during which magnesium goes into solution as magnesium sulphate. Calcium carbonate is dissolved and precipitates as calcium sulphate and carbon dioxide is evolved. The leached zinc concentrate is upgraded in flotation cells where calcium sulphate is rejected to produce final zinc concentrate.

Figure 11 Leaching Flow Diagram



Dewatering

Following leaching and re-float, the resultant Pb and Zn concentrates, which have a very high water content, have to be dewatered. The concentrates are dewatered separately using thickening and filtration in *Metso* pressure filters.

Positive Impacts

- The installation of the pressure filter drying systems eliminated the used of thermal drying and resultant emissions to atmosphere.
- Elimination of nitric acid usage. Nitric acid was used as a cleaning agent for cleaning the old *Hoesch* filter plates

Decommissioning of Thermal Zinc Dryer; Elimination of Atmospheric emission point No. A2-1

In September 2006 the thermal system for drying / dewatering zinc concentrate was replaced by a mechanical 'Metso' pressure filter system. In this process, concentrate material which is in a slurry form with a very high moisture content (approx. 30% solids by weight) is pressed between cloths under hydraulic pressure. Moisture is compressed out of the concentrate to produce a final product containing 8% moisture. The filtrate water is collected and returned to the concentrate thickeners. This process imparts zero emissions to atmosphere. This development has resulted in many positive benefits / impacts with the most significant being the elimination of thermal dryer stack emissions of particulates, metals and gases to atmosphere. Total mass emissions of metals to air have been reduced by 98% since 2005.

Currently Zinc concentrate is automatically distributed between a 15m high-rate thickener and / or the 29 m *Eimco* thickener. The High Rate thickener automatically regulates it's feed valve to maintain a constant bed mass, and the balance of zinc concentrate production is automatically diverted to the conventional thickener. Flocculent addition to each thickener is automatically regulated and independently controlled. The overflow water is used in the grinding circuit.

Four independent monopump underflow systems, two per thickener, operate to supply a stock tank. From the Stock tank a centrifugal slurry pump fills each filter to the correct pressure. The two filters , each with an approximate capacity exceeding 40 tonnes per hour, dewater the total Zinc concentrate production to a final moisture conent of < 8.5 %. Filtrate water is returned to a thickener, and the filter cakes are conveyed to the concentrate store.

The lead flotation concentrate is pumped to a thickener, the overflow of which returns to the grinding circuit. The thickener underflow system is fully automated and maintains a fixed level of controlled slurry density in a surge tank. The thickened lead concentrate is then pumped to two *Mesto* Pressure Filters which have a throughput total capacity of approximately 20 t/h with residual moisture of less than 6%. The pressure filter and conveyor system are fully automated.

The *Metso* Pressure Filter System replaced the existing Lead Hoesch pressure filter system, which did not incorporate an air emission. The positive aspect of this development was the elimination of the use of Nitric acid, which was used to clean the Hoesch filter plates. The volume of nitric acid used annually was approximately 150 Tonnes.

Dewatered lead concentrate is conveyed to a storage building which has a capacity of approximately 5,000 tonnes.

Positive Impacts of the new dewatering systems:

- Significant reduction in metal emissions to atmosphere
- Significant reduction in dust in the workplace
- In the case of replacement of the Hoest filter discontinued use of nitric acid
- Elimination of heavy fuel oil usage (approximately 2000m^3 per annum, which was used to run the burner system operating the thermal dryer)
- Reduction in operating costs, more consistent tonnage throughput and residual moisture content
- Elimination of wet scrubber
- Reduction in electric power consumption
- Cleaner air quality in the workplace
- Lower noise levels

Concentrate Loadout

Following dewatering the residual moisture content is less than 9% for zinc concentrate and for 6% lead concentrate. The final products produced at the mine are Zinc concentrate (56% Zn by weight) and Lead concentrate (65% Pb by weight). This material is conveyed to a 30,000 tonne capacity storage building, where it is loaded onto 55 tonne train wagons for rail transport to Dublin. From Dublin the concentrate is shipped to various smelters in Europe.

The concentrate Loadout building has two air extraction points IPPC L Ref A2-3 and A2-4. The Loadout system was upgraded following the installation of the new zinc pressure filter drying system. As a result the larger ventilation stack (Emission Point Ref. A2 –3) has been de-commissioned. The smaller ventilation stack (Emission Point Ref. A2-4) only operates when train loading is in progress.

Concentrate is reclaimed from the storage pile by a front-end loader which feeds a conveyor system. Separate loading hoppers and conveyors are employed for the lead and zinc concentrates, but the shuttle conveyor is common to the two systems and incorporates a lid-lifter to handle the pressed-steel wagon covers. Each wagon stands on a rail-scale during loading, and the entire procedure is automated. Each train passes through an automated wagon washer before leaving the site prior to dispatch to Dublin Port. From time to time concentrate is transported by road to Drogheda using covered trucks, which are loaded inside the concentrate store and washed before leaving the site.

An on-site laboratory checks the quality of the ore and concentrates being produced. Annually the laboratory receives 25,000-30,000 samples on which quality control analyses are carried out.

Figure 12 Zinc Dewatering Flow Diagram

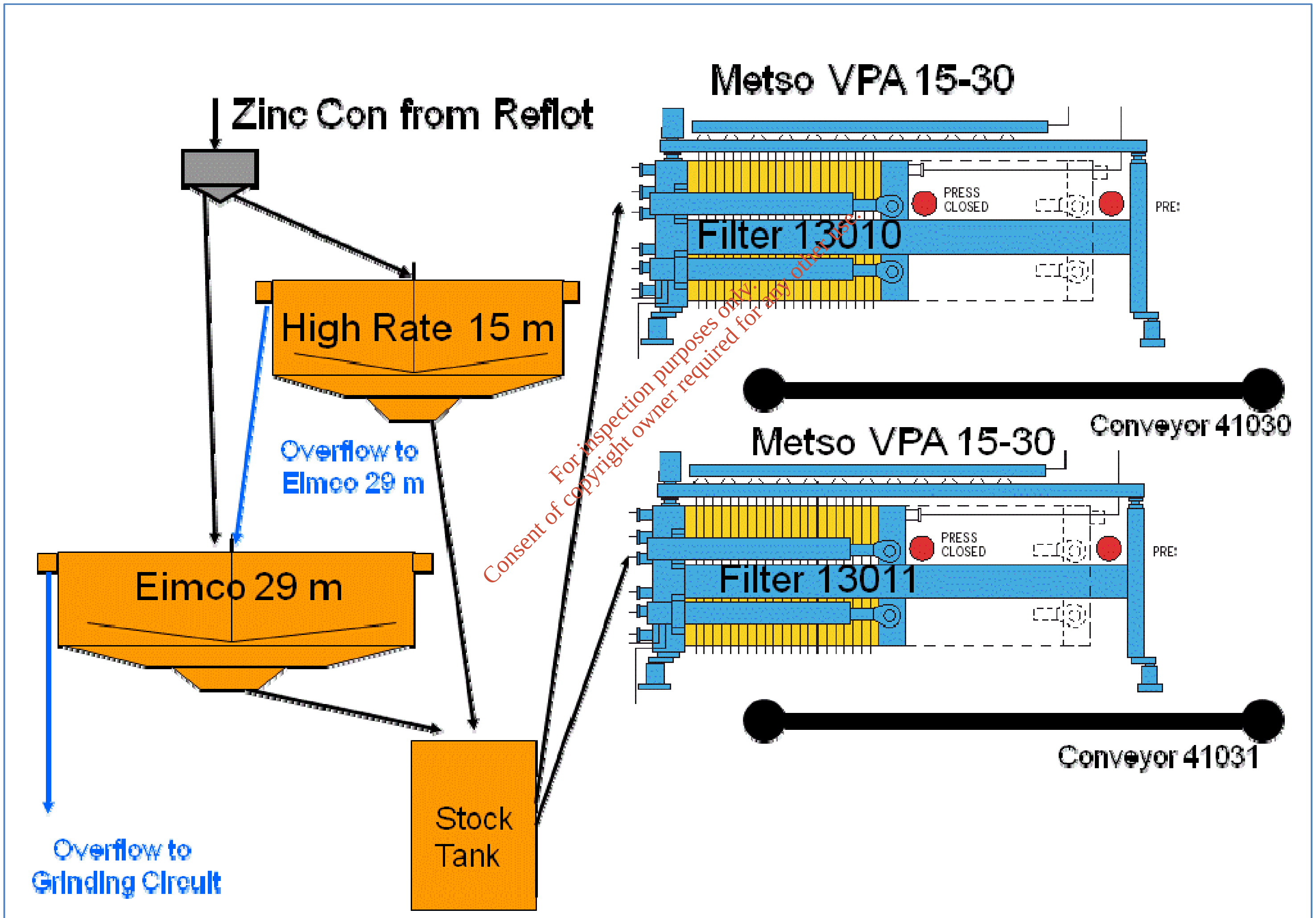
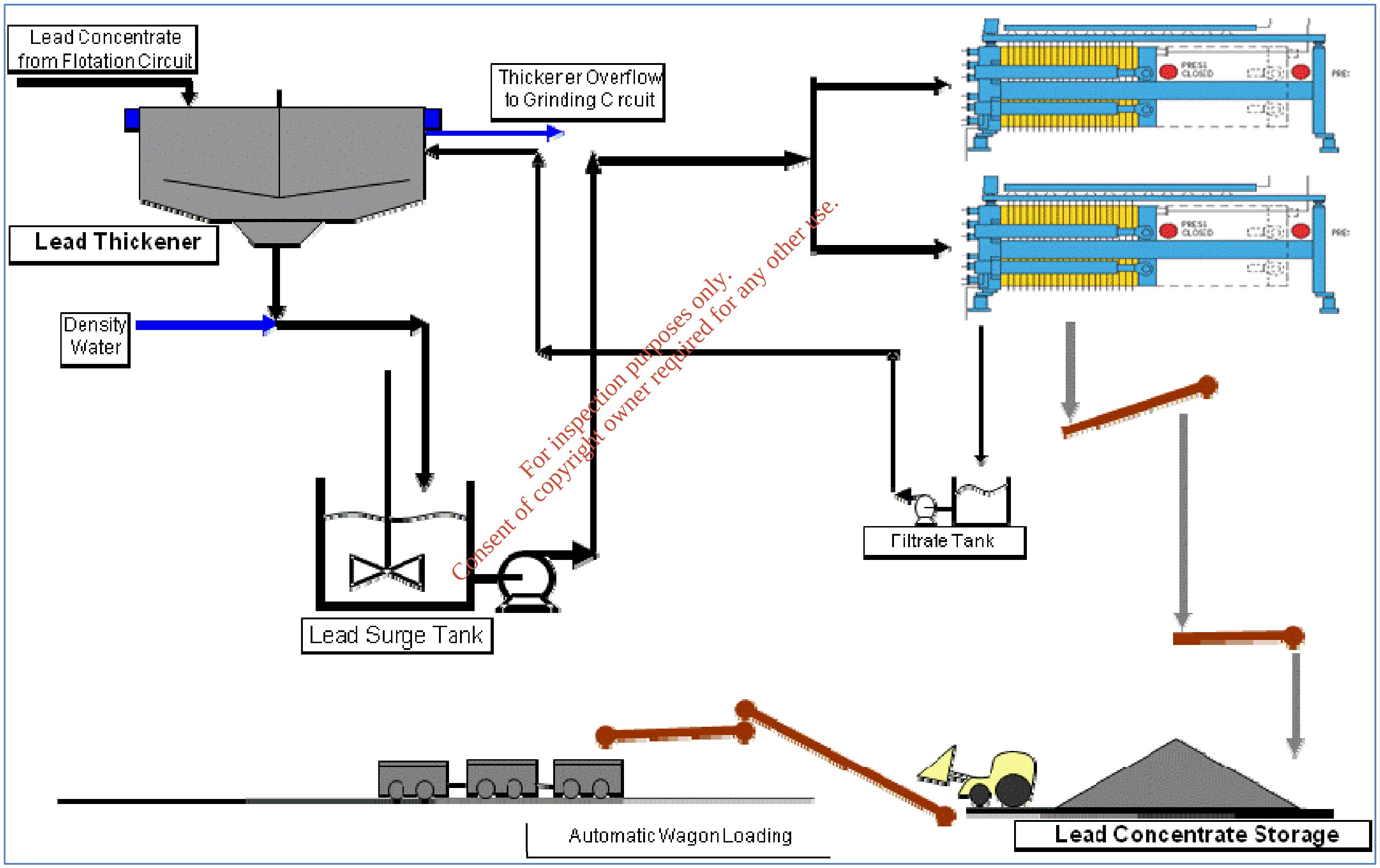


Figure 13 Lead Dewatering Flow Diagram



Tailings and Backfill

Tailings, consisting predominantly of ground limestone and water, is the material remaining after the recoverable zinc and lead minerals have been extracted from the ore.

Mine tailings comprise coarse particles, known as sands, and fine particles, known as slimes. The coarse and fine fractions are separated using cyclones. The sand fraction is typically returned to the mine as backfill to stabilise the worked out areas. The remaining fine fraction, along with any sand not required for backfilling is pumped to the Randalstown Tailings Storage Facility (TSF). On average approximately 1.1 million tonnes of tailings (equivalent to 750,000 m^3) are pumped to the TSF per annum.

The TSF has been constructed in five stages during the period 1974 to 2006.

- Stages 1 and 2 were filled and vegetated by 1988.
- Stage 3 was constructed between 1985 and 1987 and was filled by March 2003.
- Construction of Stage 4A, a raised facility over the existing tailings in Stages 1 and 2, began in late summer 1998, was completed in July 2000 and filled by 2007.
- Construction of Stage 4B, which is founded on the Stage 3 tailings, started in the summer of 2003 and was completed in 2006 and is now the active cell.

Figure 11 shows the layout plan for Stages 1 to 4 and for the proposed Stage 5 extension while Figure 12 shows a cross section through the dam wall including the proposed stage 5 construction.

The total capacity of the combined Stages 1 to 4 is 35.6 million tonnes (equivalent to approximately 25 million m^3). No other tailings disposal facilities have been utilised by Tara Mines since its inception. Stage 4B is currently operational and, at present rates of ore processing, has sufficient void capacity to accommodate tailings until the end of 2013.

Tara has sufficient ore reserves to support operations until at least 2018. It is calculated that additional void space amounting to 5.8 million m^3 (i.e. sufficient void space for the disposal of a further 8 million tonnes of tailings) will be required to sustain life of mine beyond the end of 2018. The proposed extension, to be known as Stage 5, will raise the Stage 4 crest elevation by 4 meters and will provide the required storage capacity. The present average height of the embankment wall is 20m and the proposed development will result in a new average height of 24 meters above ground level or 67.6 mAOD .

The coarse sand is stored in surface tanks with a combined capacity of 7,000 tonnes. When backfill is required underground, water and cement are added to the sand and the mixture is then pumped through boreholes into open stopes. Typically, more than 1 million tonnes of sand and 17,000 tonnes of cement are used for backfilling each year. In addition, approximately 45,000 tonnes of waste rock are also used. When the sand has been removed, the residual tailings are pumped from the concentrator to the TMF in which suspended solids are allowed to settle out leaving clear water for recirculation and re-use.

A complete description and assessment of the Stage 5 proposal is presented in the EIS for the proposal see Attachment B.5 Part III.

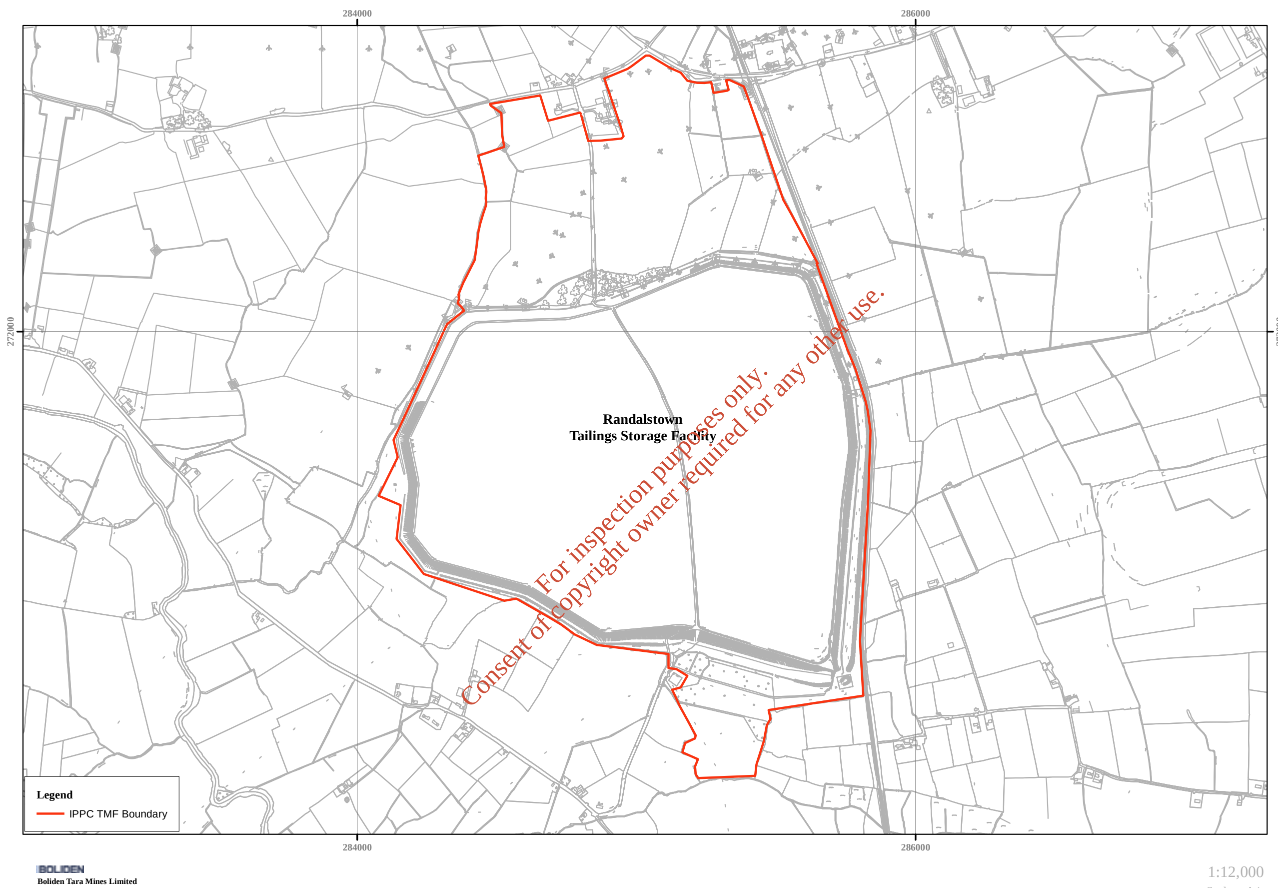


Figure 14   Tailings Storage Facility.

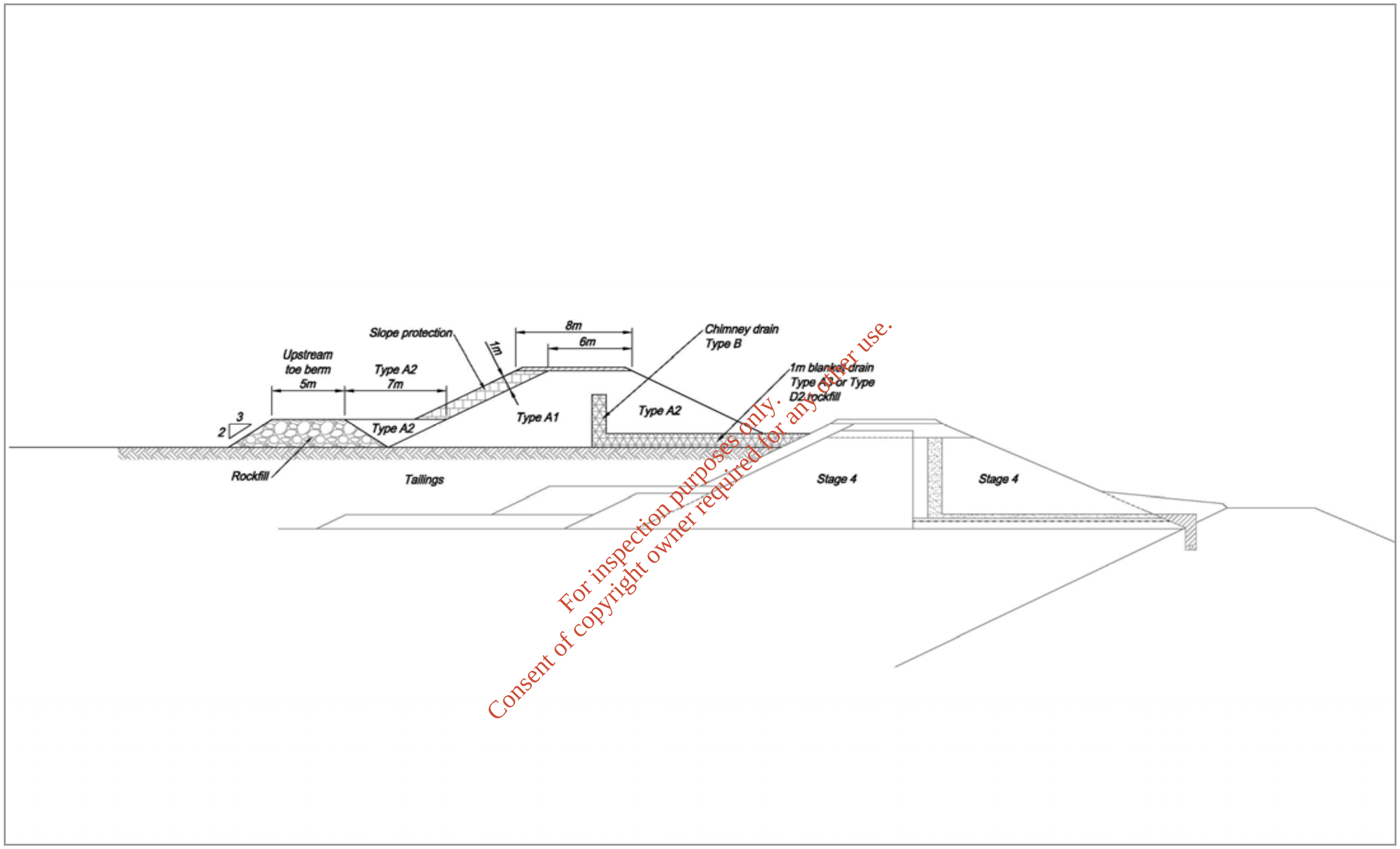


Figure 15 Cross-section TSF Stage 5 Design.