Q2.2.2 Sources of dust

The main sources of dust underground were covered in general terms in section Q1.2.2 of the first course.

Ore/Waste Passes and Storage Bins

A problem arises when discharging rock into vertical or near vertical ore/waste passes and storage bins and is specifically the potential for contamination at the open tipping point with dusty air from the transient reverse airflow pulse caused by the air expansion when tipping stops.

The higher the vertical sections of an ore or waste pass the greater the falling rock induced pressure increase leading to stronger transient back flow.

Drag force and pressure

When a fluid flows around a bluff (not streamlined) object as illustrated in Figure A2.9 in the airflow section of the second course, the boundary layer separates and a turbulent wake are formed. The magnitude of the resultant drag force F_D is dependent on the projected area of the object A_O, its drag coefficient C_D, the fluid density ρ and fluid velocity V as given by Equation A2.7 and repeated :

$$F_D = A_O\, C_D\, \tfrac{1}{2}\, \rho\, V^2 \qquad \text{Equation A2.7}$$

The frictional pressure drop caused by the object is the drag force divided by the cross sectional area of the airway A_A as given in Equation A2.8 also repeated below. The ratio of the area of the object to the cross sectional area of the airway A_O/A_A is also known as the coefficient of fill C_F.

$$Pd = F_D/A_A = C_F\, C_D\, \tfrac{1}{2}\, \rho\, V^2 \qquad \text{Equation A2.8}$$

The drag coefficient of an object is a function of degree of boundary layer separation, which in turn is dependent on the shape of the object and the ratio of the fluid inertial and viscous forces expressed as Reynolds number. For rocks falling in an ore pass, the Reynolds numbers are generally much greater than 10^4 and the drag coefficient would be between 0.2 for a sphere and 1.2 for a cube.

Ore pass over-pressure measurements and tests on similar sized material used in hydraulic hoisting resulted in indicated drag coefficients of between 0.5 and 0.6. This material had already been crushed and had a mass median diameter of 0.022 m. Most rock being tipped into the passes will be an order of magnitude larger and, to allow for a greater diversity in shape, an average value of 0.7 is often used.

Rock size distribution

The material dumped into ore/waste passes is from both development and stoping. Rock size will be a function of blast fragmentation and is likely to be log-normal (see section Q2.2.3) with a mass median diameter D_m of 200 mm and a geometric standard deviation σ_g of 3.0.

For an analysis of pressure increases in the ore pass, the rock mass is divided into size fractions ranging from 15 D_M to $D_M/15$ with the limits of each size range decreasing by 25% (76 size fractions). The arithmetic mean diameter of each size fraction is taken to be representative of that size fraction and mass in each fraction obtained by calculating the areas under the normal distribution bell curve using logarithms of the fraction limits.

Falling rock velocities

In an unventilated ore or waste pass, the fluid (air) is not normally flowing and it is the falling rock that creates the velocity difference.

A rock falling from rest will accelerate to a terminal velocity V_T where the gravitational force is balanced by the aerodynamic drag. For rocks having a density W and a volume diameter d (diameter of a sphere having the same volume as the rock), the terminal velocity can be obtained from Equation Q2.16. The free fall velocity V at any distance h below the discharge point can be obtained from Equation Q2.17.

$$V_T^2 = g\,k \qquad \text{Equation Q2.16}$$

Where $k = \dfrac{4Wd}{3\rho C_D}$

g = acceleration due to gravity (9.807 m/s^2)

$$V^2 = gk\left(1 - e^{\frac{-2h}{k}}\right) \qquad \text{Equation Q2.17}$$

As a consequence of collisions with the ore pass walls, the actual rock velocities in the ore passes are likely to be less than the free fall velocity given in Equation Q2.17. In non-vertical passes, the number of wall collisions would increase and the rock velocities decrease accordingly. The terminal velocities and the expected free fall velocities for the three rock sizes at 100, 200 and 300 m down the pass are given in Table Q2.8 (rock density of 2800 kg/m^3 and an air density of 1.2 kg/m^3).

Table Q2.8 - Free fall rock velocities

Rock size (m)	*0.067*	*0.200*	*0.600*
Velocity at 100 m (m/s)	39	42	44
Velocity at 200 m (m/s)	49	57	61
Velocity at 300 m (m/s)	53	67	73
Terminal velocity V_T (m/s)	59	102	177

Ore/waste pass pressure calculations

Typical loaders and truck capacities are between 10 and 50 tonnes and normally discharge into ore/waste passes through a grizzly and finger raise. Allowing for swell the discharge rate *m* into the ore pass varies between 1000 and 7500 kg/s and the feed period normally varies between 2 s and 10 s.

At any cross section in the ore pass, the volume fraction ν of rock in the air will depend on the mass flow rate of the rock, its velocity at the cross section in question and the cross sectional area of the ore pass as given in Equation Q2.18.

$$v = \frac{4m}{\pi W D^2 V} \qquad \text{Equation Q2.18}$$

Using a 50% rock size of 0.200 m and a pass size of 2.5 m x 2.5 m (2.8 m equivalent diameter), the volume fraction of rock at 100, 200 and 300 m below the discharge is 0.0083, 0.0061 and 0.0052 respectively. This is sufficiently low for the fall of each rock to be considered independently.

The fall is not a solid "plug" of rock as may be represented by a piston in a cylinder. It is, however, possible that there will be some rock particles falling within the wake of preceding rock particles. This would reduce the overall drag and, by not taking it into account, a conservative estimate should result.

Pressure increases

With the rock falling through the air in the ore pass, the pressure developed at any point will be determined by the drag force exerted by the falling rocks at and above that point i.e. the pressure created below the falling rock is limited by the energy required for the air to "slip" past.

Ignoring any compressibility effects, the increase in pressure *P* at any point below the discharge into an unventilated ore pass is obtained from Equation Q2.19. The time *t* in seconds taken for a rock to free-fall a given height *h* in metres can be obtained from equation Q2.20.

$$P = \sum \frac{3 M_i \rho C_D V_i^2}{\pi W d_i D^2} \qquad \text{Equation Q2.19}$$

where P = pressure increase (Pa)

M_i = mass of rock in the size fraction (kg)

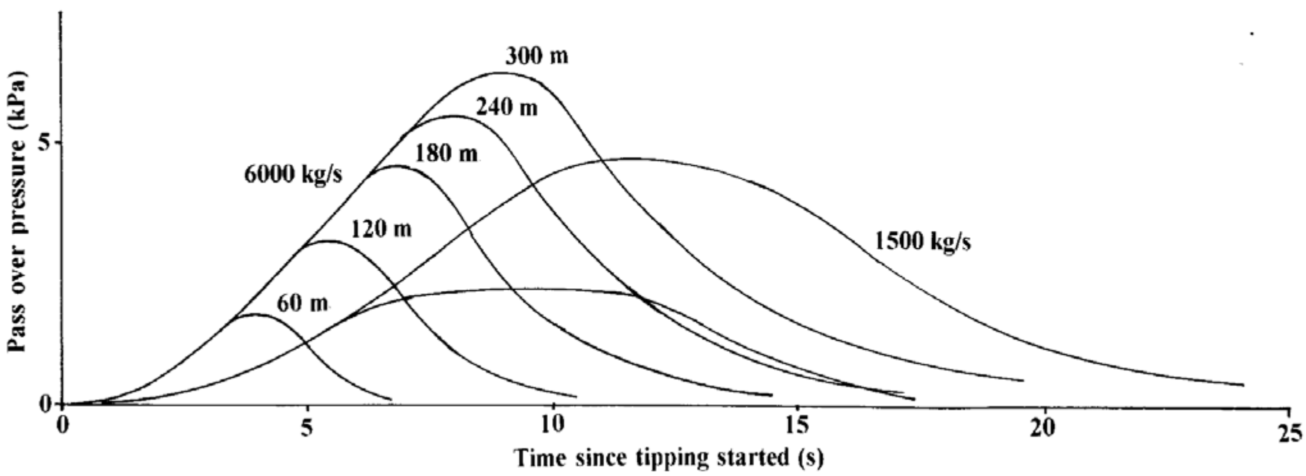
V_i = free fall velocity of rock of size d_i (m/s)

d_I = mid size of the size fraction being considered (m)

$$t = 2\, h/V \qquad \text{Equation Q2.20}$$

By using Equations Q2.17 and Q2.20 and the size distribution, an algorithm can be developed where the location of any rock is determined for increasing times after the start of tipping. Equation Q2.19 can then used to determine the pressure at discrete points down the ore pass. The results for a 12 t loader tipping into an ore pass are illustrated in Figure Q2.11.

Figure Q2.11 Tipping from a 12 ton loader at two feed rates and increasing pass height



The maximum pressure developed is a function of the height of the ore pass, the feed rate into the ore pass and the total mass of rock tipped. This latter point is only relevant where the ore pass height is reasonably large and where the falling rock is not leaving the section of the ore pass almost as quickly as it is added. The pressures developed are based on free fall of the rock without wall collisions and do not include any mutual interference between falling rock fragments.

Transient conditions

Tipping into an ore pass is a cyclic process and the resulting pressure perturbations are transient and not steady flow. Using a 180 m high pass and a 6000 kg/s feed rate, from Figure Q2.11 the pressure at the bottom of the pass will increase to 6.0 kPa in about 7.5 s. During this period air will be drawn in the open finger rise as the rock falls down the ore pass.

When the falling tipped rock hits the stationary rock in the ore pass, the air at this point has been compressed and will then expand. The expansion wave front will travel back up the ore pass at a slightly sub-sonic speed. As it travels back up the ore pass some energy will be absorbed by wall friction. If the initial pressure wave energy was high, when the remnant arrives at the tipping finger rise, it can cause air to flow back out of the ore pass. Like the initial tipping pressure increase, this reversal is also transient and dusty air from the ore pass can then contaminate the tipping area.

Q2.2.3 Control of Dusts

Source Control

The source of dust and basic control methods were covered in the first course. Source control is the first step in any control procedure and in mining; this is fundamentally the correct application of water to stop the dust becoming airborne.

The particulate material created by metallurgical operations can also be controlled at the source by judicious design. An example is a submerged arc furnace where the depth of electrodes in the smelt can be related to the dust and fume production rate. Contaminant production can be reduced but inevitably some other penalty must be paid such as increased power and electrode consumption.

Often a change in design, which means a slightly less efficient operation on one item, can result in improvement when the overall plant is examined. An example is a steam generation plant. Changing the feed mechanism to the boiler from a chain grate to a rotary type can increase the fly ash production by between two and five times. The overall boiler efficiency may increase by a few percent, however, considerably more energy must now be expended in cleaning the fly ash, from the flue gases.

Control to prevent dust explosions relies mainly on reducing the concentration of carbonaceous material with an inert stone dust. The dust that becomes airborne then has insufficient carbon to propagate the explosion. In some situations barriers, which may be triggered by the explosion, can be used. The barriers may use water or an inert stone dust. Sulphide explosions are avoided by eliminating time delays in blasting that eliminates the ignition of a dust cloud resulting from blasting.

Dust from tipping at passes

The problem was described in the previous section where the action of rock falling down a pass compresses the air and, when the tipping is complete, the consequent expansion of air in the pass can cause dusty air to flow out of the pass and into the tipping area. Exhausting from the top of the ore pass usually controls it.

Providing the exhaust quantity is sufficient, the open tipping finger rise will always be downcast and it is most unlikely that the transient expansion will cause a flow reversal. An estimate of the transient "back flow" is obtained by considering the rate of pressure decrease as the falling rock hits the stationary rock in the ore pass and assuming an adiabatic expansion.

For example, the maximum pressure caused by the falling rock when tipping from a 40 t truck onto a square grizzly and into a pass having a height of 300 m is 17.1 kPa. Assuming an adiabatic expansion of the air results in a 175 m^3 increase in air volume over a recovery time of 8.8 s.

Avoiding any back flow out of the open tip would then require a ventilation extraction rate of 18.5 m^3/s from the top of the ore pass. Since only the bottom section of the ore pass is at the highest pressure and has the largest air expansion, this tends to exaggerate the required ventilation extraction rate.

By integrating the air expansion over the full height of the pass, a back flow out of the tip would be avoided with a peak ventilation extraction rate of about 9 m^3/s. The expansion is about 85 m^3 and the recovery period starts when tipping is finished about 7 s after the start of and completed about 15 s later.

Tipping lesser amounts of rock such as 12 t from a loader, has a lower maximum air pressure (6.4 kPa for a 300 m high ore pass), however, the air pressure recovery time is shorter and the ventilation rates are not significantly different to those for a 40 t truck.

With a smaller ring grizzly, with a feed rate into the ore pass of about a quarter that of the square grizzly, the required ventilation top extraction rate is halved.

Re-circulation control

An alternative dust control method is to connect the ore and waste passes just below the lowest finger rise and at the top of the ore and waste passes.

As the tipped rock falls down one pass, the increase in pressure will cause a flow of air through the connections to and from the other pass as well as up the other pass itself. The re-circulating flow reduces the induced pressure increase (the air flows down the pass in the same direction as the falling rock reducing the effective drag) and should also reduce the magnitude of the transient back flow.

By also exhausting from the top of the passes, the open tip will be downcast with dusty air removed from the system and the back flow eliminated.

If it is assumed that the effective length of ore pass is 240 m and that the bottom and top are connected with 4.5 m x 4.5 m drives, the induced maximum re-circulating airflow are given in Table Q2.9 for varying tipping heights.

It is evident that the induced air quantities are too large to exhaust directly to the main exhaust and the re-circulation is essential. The effect on the induced pressures is also given with the values in brackets without re-circulation.

Table Q2.9 - Induced re-circulation (240 m long and 2.5 m x 2.5 m passes)

Tipping height from bottom of ore pass (m)	*Induced maximum airflow (m^3/s) LHD - 12 t*	*Induced maximum pressures (kPa) LHD - 12 t*
60	66	1.1 (2.4)
120	101	2.6 (6.6)
180	124	4.2 (10.5
240	145	5.5 (14.2)

The air removed from the bottom of the ore pass is replaced by air flowing in through the tip finger raise. Although the reverse pressure pulse is reduced, it is not eliminated, however, the air inflow through the tip finger raise should minimise the potential for any contamination of the tip area.

Multi-level tipping

It is evident from the values given in Figure Q2.11 that the induced over-pressures are significant and the potential leakage of dusty air is increased accordingly. For example, if a 40 t truck is tipped into the top of an ore pass, a finger raise and tip 240 m below the tipping point will experience pressures building up to 14.2 kPa about 10 s after tipping starts. At this maximum pressure almost 1 m^3/s will flow out through a hole as small as 100 mm square. Over the 20 s that the over pressure exists, approximately 8 m^3 of dusty air would flow through the 100 mm square opening.

This also demonstrates the problems encountered with multi-level tipping where more than one tip is open. As the rock from the highest tip falls down the ore pass, the high over-pressures will induce air quantities out of the open tips similar to those given in Table Q2.9. Without rock flow controls in the finger raise (chains etc) which block the leakage air route, it is impractical to reduce the induced airflow and subsequent contamination to acceptable values.

Ore pass covers

As illustrated in Figure Q2.11, the air over-pressures that occur when tipping rock in vertical ore passes depend on the amount and feed rate of rock being tipped and the height the rock falls. Other factors are the size of the ore pass and the size distribution of the tipped material.

The peak air pressures that can be expected at various tipping heights for a 12 t scoop tram and a 40 t truck in a 2.5 m x 2.5 m ore pass are given in Table Q2.10. It is evident from the values given in the table that the induced over-pressures are significant and the potential leakage of dusty air through the tip covers below the active tipping point is increased accordingly. The values given in brackets are for a lower feed rate.

Ore pass finger grizzly covers are normally designed for the worst condition that is based on the tipping of the largest truck into the highest tip of the ore pass. For a grizzly having nine 0.8 m square openings, a suitable cover would have an area of about 10 m^2.

The mass of a cover necessary to prevent the over pressure causing it to lift when tipping is therefore about 1.0 t per kPa over pressure. A 6 mm steel plate grizzly cover with suitable bracing to prevent deformation when tipping would have a mass of about 1.0 t.

Table Q2.10 - Ore pass over-pressures for 12 and 40 tonne loads

Tipping height (m)	*12 tonne load (kPa)*	*40 tonne load (kPa)*
60	1.8 (1.3)	2.4
120	3.4 (2.5)	6.6
180	4.6 (3.4)	10.5
240	5.6 (4.2)	14.2

The design of a suitable cover or plug needs to take into account the requirement that air leakage should be reduced to a minimum and the ease with which the cover or plug can be raised and lowered (or installed and removed).

The required force/mass for the grizzly covers can be obtained using: concrete weights (a 15 t door would require a 0.5 m thickness of concrete over its 10 m^2 area), hydraulic rams which are also used to raise or lower the cover or, pins which are used to fix the cover to the grizzly frame when it is maneuvered into position.

If a 1.2 m diameter grizzly ring is used instead of the nine 0.8 m square openings, the required mass of a cover is about one tenth of the above values. Where grizzly rings are used the design of a cover is much simpler with the required mass being much less and a tapered plug providing a reasonable seal and being straightforward to locate in the grizzly opening. A single design with a mass of about 2 t would be suitable for most locations in the ore and waste pass systems restricted to a height of 300 m.

Dust Collection Equipment

The main types of particulate collection equipment are given in Table Q2.11.

Settling Chamber

Settling chambers uses the principle of gravitational sedimentation to remove very coarse particulate material. The gas stream is normally allowed to expand into a large chamber, the particulate falls by gravity from the low velocity gas stream and is collected in a hopper. Referring to Table Q2.6 the settling velocity of a 100 μm dust particle of density 2700 kg/m³ is approximately 0.75 m/s.

Table Q2.11 Particulate Collection Equipment

Type of equipment	*Size collected (μm)*	*Pressure loss (Pa)*
Settling chamber	Over 100	50
Inertial collectors	Over 50	250
Cyclones	10 to 100	250 to 1000
Spray towers	Over 5	250
Venturi scrubbers	less than 1	1000 to 6000
Fabric filters	less than 1	100 to 2000
Electrostatic precip.	less than 1	50 to 250

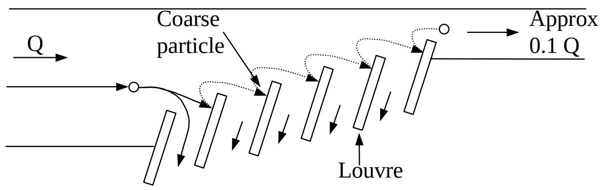
Notes 1. Particle density (2700 kg/m^3)

For large dust particles the settling velocity is proportional to the square of the diameter. For particles smaller than 100 μm, the physical size of the required settling chamber renders the system uneconomic. This method is usually used as a pre-separator to reduce the particulate load on a more efficient secondary collector.

Louvered Collectors

The placing of baffles or making the air change direction in a settling chamber increases particulate removal by introducing inertial impaction. In the louvered separator illustrated in Figure Q2.12 the larger particles leave the air stream entering the louver gap, hit the louver and bounce back into the main air stream. The dust concentration in the remaining air stream builds up and this can be cleaned in a separate collector. The main advantage is that the amount of air to be subsequently cleaned is only approximately 10% of the total quantity of air entering the louvered collector.

Figure Q2.12 Louvered Dust Collector



In other separators, the particles impacting on the baffles or louvers are collected in a hopper below the separator. The rate of impaction of particulate is proportional to the square of the diameter and the velocity and inversely proportional to the distance the particle travels to the impaction surface. (In the louvered collector this would be a function of the width between the louvers).